

U. S. DEPARTMENT OF COMMERCE

R. P. LAMONT, Secretary

BUREAU OF MINES

SCOTT TURNER, Director

Bulletin 356

SAMPLING AND ESTIMATION OF ORE DEPOSITS

BY

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UNITED STATES
GOVERNMENT PRINTING OFFICE
WASHINGTON : 1934

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ing, beneficiation, and metallurgical treatment has generally recognized importance to the mineral industries.

Volumes have been written upon this important and complex subject and its varied phases. In 1904 T. A. Rickard discussed the subject comprehensively in his book, *The Sampling and Estimation of Ore in a Mine*. This bulletin attempts to discuss methods employed at a large number of mines and under a wide range of conditions in connection with the exploration and mining of ore deposits and does not include methods employed in ore-dressing and metallurgical plants. From a background of considerable personal experience in sampling the authors have discussed methods employed at various properties, as described in published articles by engineers in charge of the work, and methods observed during visits to various districts.

Sampling is employed in connection with a number of different operations and with several different purposes in view. Although the method to be used varies primarily with the type of mineralization, mode of occurrence, and other geological features of the deposit, the end sought should be to obtain samples which will, when properly combined, represent as accurately as possible or practicable the entire volume of the deposit within the area sampled.

The degree of accuracy required or desired will depend upon the nature and value of the ore and the purpose for which the sampling is done. The degree of accuracy possible will depend upon the uniformity and type of mineralization and other geological features and will be limited from a practical standpoint by the amount of money available or warranted for the work.

The correct interpretation of the results of sampling and their use in computations of tonnages and grades of ore require skill and judgment based upon wide experience. An understanding of the geology of the deposit is essential; and the type and shape of the deposit, mode and habit of ore occurrence, distribution of valuable minerals, etc., must be given careful consideration.

APPLICATIONS OF SAMPLING

Since at its inception a mining enterprise requires a preliminary program of prospecting and exploration, the first use of sampling may be said to be in connection with these operations. Samples taken during this period may be used as a basis for determining the limits of ore bodies and the distribution of valuable minerals within them and for estimating the tonnage and value of ore within the explored area.

In the transition from the prospect to the development stage of a mineral property transfer of ownership or of a portion thereof for obtaining additional capital is often involved, and resampling of the exploratory workings is commonly required. Thus, sampling by a prospective investor, a vendor, or both is perhaps the second important application of ore sampling. Thus sampling is employed as a basis for evaluating the developed ore and for forecasting probable and possible additional ore.

Visual examination of the samples with the naked eye or the microscope, and chemical analyses may assist the skilled geologist

to formulate an opinion on probable extension of the ore and its grade at greater depth than is reached by existing exploration and so to predict the possible value of the deposit in addition to that represented by proved ore.

During the development period of a mining property, sampling is again employed to add to the knowledge of tonnage and grade of ore revealed during the exploration period, as development progresses, and to guide further development and mining of the deposit.

During the production stage sampling is usually required for control of stoping operations and for exploration and development, which generally continue, in some degree at least, throughout the period of ore production.

EXPLORATION AND PROSPECTING

Exploration work ranges from that performed by the individual prospector employing only the simplest tools and equipment to elaborate campaigns involving the expenditure of large sums of money and the use of churn or diamond drilling equipment, large test-pitting crews, or sometimes a regular mining organization. In any event the object is to determine as accurately as possible, with the funds available, the size, shape, and other physical characteristics of the ore bodies and their average value or grade.

An individual prospector is usually limited to small expenditure, and if he can determine with the limited means at his disposal the existence of a large enough deposit, its location and grade considered, to give some indication that exploitation will be profitable, he will have achieved no mean result. He will seldom be able to determine the ultimate limits of the ore laterally or at depth. His methods will usually include trenching, shallow test pitting, and possibly some tunneling and drifting.

To interest capital in further development of a property it is important that a prospector present the facts regarding his discoveries so far as his finances have permitted him to determine them. Among these facts assay values of representative samples of all ore exposures have prime importance.

The value of such information often is not realized by the prospector until he tries to interest capital in developing his property. At this point he finds that his speculations, backed up by a few haphazard samples of high-grade material, have no value whatever, although he may actually have a property of merit.

Assuming he has such a property, a rough sketch showing accurately the location and dimensions of the mineralized zone as far as they have been determined, with data on the location, length and width, and the method of taking and assay values of samples, will seldom fail to interest an examining engineer. Such information does not need to be prepared by an engineer or draftsman or to be attractive in appearance, as long as the facts are clearly shown. If proper sampling methods appear to have been employed and the minimum proved size of the deposit in relation to assay values is promising, it is seldom difficult to interest an engineer or geologist in making an examination of the property. The examining engineer will usually take his own samples, and it is within his province to

examine geological conditions and speculate upon the possibilities as to size of deposit and extension and grade of ore at depth.

SAMPLING METHODS

Methods of sampling prospect openings, such as trenches, test pits, shafts, tunnels, and drifts, are the same as for sampling development headings in operating mines. Examples of methods employed, with satisfactory results, at operating mines in ore deposits of various types, structure, and character of mineralization are described in later sections of this paper. Suffice it to say that different types of deposits require different sampling methods.

Experience and judgment are required in selecting the method of sampling best adapted to a given set of geological conditions. Once the method has been selected, consistent results can be obtained only by following this method mechanically, eliminating the human equation as far as possible in the cutting of samples. Cold-blooded sampling is not easy, as it is the human tendency to favor the selection of good-looking pieces of ore. On the other hand, the too-conscientious sampler may, in his endeavor to combat this human tendency, avoid pieces of high-grade ore which properly belong in the sample, thus obtaining results that fail to measure up to the actual grade of the material sampled.

Large, regular deposits in which the valuable minerals are distributed uniformly are the simplest type to sample accurately by the ordinary drill, channel, muck-pile, and car-sampling methods. With more erratic mineralization sampling becomes increasingly difficult; and in some types of deposits, especially veins containing precious metals irregularly distributed in the gangue, the taking of each individual sample may be a problem in itself, even to the question of inclusion or rejection of visible metal in the sample after it has been cut.

The size of sample to be cut is an important consideration. A well-recognized principle is that the finer the size of ore and particles of gangue and the more evenly distributed the mineralization, the smaller may be the sample and the greater the interval between samples to obtain results of equal accuracy.

The mineralization of some ore bodies is so erratic that only a mill test of a considerable tonnage of ore will indicate grade accurately. No method has yet been devised for sampling in place the native copper ores of northern Michigan. Sometimes a mill test of a block of ground sampled previously by other methods may be employed to establish a factor which may be applied with confidence to assay values of other samples in the same deposit for estimating the grade of the ore.

Part 1.—DRILL SAMPLING

Churn drills, core drills, and hammer or piston drills are employed extensively for quickly and cheaply obtaining samples of formations and ore in mineralized areas.

Drilling is sometimes used merely to determine geological structure or to delimit mineralized zones, and in some types of deposits can not be relied upon to give results upon which to base accurate calculations of the grade of ore. In other types of deposits, however, drill samples accurately indicate the grade of ore and check closely with those taken by other methods and with actual mill tests of the ore.

Drilling methods give the most accurate results when employed for sampling large ore deposits in which mineralization is regular and uniform, such as the disseminated or porphyry copper deposits, the iron ores of the Lake Superior districts (especially those of the Mesabi range), the disseminated lead ores of southeast Missouri, the lead-zinc deposits of the Tri-State district, and other bedded deposits, such as the copper ores of northern Rhodesia.

In deposits of these general types drill samples properly taken and interpreted can be relied upon for making estimates of grade and value that will check closely with actual results from later mining operations. Drilling has been widely employed on a large scale for sampling such deposits.

When employed for sampling deposits irregularly and erratically mineralized, where the values are "spotty," results are not to be depended upon as a basis for making accurate estimates of grade of ore. In such deposits drilling has value principally for determining geological structure, formations passed through, nature of mineralization, and other collateral information useful as a guide to exploration. This type of deposit is illustrated by such well-known examples as the gold ores of Porcupine and Kirkland Lake, in Ontario, the amygdaloid copper deposits of Keweenaw Peninsula in Michigan, and contact metamorphic deposits of Arizona, New Mexico, and Utah.

CLASSIFICATION OF DRILL-SAMPLING METHODS

Drill-sampling methods may be classified as follows on the basis of the type of equipment employed:

1. Drive-pipe method (for sampling placers).
2. Churn drilling (cable tool—percussion type).
 - (a) Spring pole (hand).
 - (b) Portable power-driven rigs.
 - (c) Standard rigs.
3. Rotary drilling.
 - (a) With solid or fishtail bits.
 - (b) Core drilling.
 - (1) Diamond drilling.
 - (2) Calyx drilling.
 - (3) Shot drilling.
4. Percussion rock drills of the hammer or piston type (test-hole drilling).

It is not within the scope of this paper to discuss details of drill construction and operation, but the application of the different methods will be considered briefly.

Each of the above methods has its particular field of usefulness, depending upon such considerations as the type of ore deposit, its size, depth, and inclination; characteristics of the material to be cut through; hardness; condition of ground—whether solid, fractured and broken, or loose; angles of dip of the holes; size and scope of the drilling campaign; financial ability of the operators; type of information sought; and relative desirability and cost as compared with other methods, such as sinking, drifting, crosscutting, or geophysical prospecting.

DRIVE-PIPE METHOD

The drive-pipe method is frequently employed for sampling shallow alluvial deposits, particularly gold placers and soft or partly consolidated material.

OPERATION

The method consists of driving a pipe, provided with a steel cutting shoe beveled on the inside, into the ground to be sampled and removing the "core" or plug of material from the inside of the pipe in a series of measured lengths; the core constitutes the sample, from an examination of which the value of the mineral content of the ground is calculated. The length of each sample is usually the depth from surface to bedrock, divided for convenience in operation and purposes of examination into units of 1 foot, corresponding to the depth of each driving of the pipe. After each drive the core may be pumped directly from the hole with a vacuum sand pump if in loose material or may first be broken up by churn drilling within the pipe. It is common practice to employ a light, portable churn-drill outfit (either hand or power operated) for drilling and for handling casing in drive-pipe work.

As the core or "plug" is pumped from the hole (water being used and the contents of the pipe being thoroughly mixed therewith) it is caught in a pan as it issues from the sluice box. The light, fine material (slimes) overflowing into a tub or box may be panned separately, or the entire core may be caught in a pail set in a tub below the sluice box. In any event the volume of material pumped from the hole is measured as a check against the measurement of the depth of drive and length of core in the pipe to indicate whether any of the core was lost or whether material not properly belonging in the sample has been included. To prevent the latter contingency drilling out of the core is usually stopped 1 to 4 inches above the bottom of the cutting edge of the shoe. Two pumpings are generally made, one before and one after drilling.

After the core is pumped out the drill is lowered into the pipe until it rests on the bottom of the hole, and a measurement is taken to determine the depth of core left in the pipe. The pipe is then driven another foot (or whatever the sample interval is) for the next sample.

Usually the material from each pumping is concentrated by panning, removing all but black sand and heavy minerals, the "colors" are counted, and their weight in milligrams and their grade are

estimated and recorded. The gold from all the pannings for each hole is accumulated in one globule of mercury. When the hole is finished the tailings from all the pannings are run through a rocker, together with the slimes from the sluice box and tub, and any values recovered are added to the mercury. The amalgam is later treated with dilute nitric acid, and the remaining gold is annealed and weighed.

PRECAUTIONS

Some of the precautions to be observed in placer sampling by this method are as follows:

1. **Water.**—In dry ground no more water should be added than is necessary to make the core material of such consistency that it may be pumped from the hole, but in wet ground the water level in the pipe should be kept above the water level outside to prevent an inrush of material from outside the pipe. While the drill is being pulled it should be washed to carry any adhering material back into the hole.

2. **Drilling and pumping.**—Care should be taken not to drill beyond the bottom of the pipe, unless it is absolutely necessary to break a boulder or to clear the pipe of plugged core in heavy ground or coarse gravel. The drill should be turned during drilling to cut up the core rather than to pack it into the pipe. In loose and running ground little if any drilling may be required, and in pumping care must be taken not to draw material from outside into the pipe. A careful record of depths of drives and the corresponding volume of material removed is necessary. The object is to remove all the material and no more than that representing a cylinder of the same inside diameter as that of the shoe and of a height equal to the depth of the drive. Drilling should be stopped 1 to 4 inches above the bottom of the shoe to prevent the entrance of extraneous material into the pipe. Holes are usually driven to bedrock, where the greater concentration of gold and heavy sands usually occurs; the pipe should usually be driven a short additional distance (a few inches at least) into the bedrock to make sure all the valuable ground is included in the sample.

3. **Protection of hole between shifts.**—When drilling is stopped at the end of the shift, it is advisable to throw in some barren tailing material. Then the first operation on the following day will be to pump this out clean and check the depth measurement. This will serve as a protection against possible salting of the hole.

RECORDING AND CALCULATIONS

A careful log of each hole should be kept for future reference and as a basis for calculating the value of the deposit. Figure 1 shows specimen field logs suggested by the Keystone Driller Co.⁴

Table A shows more detail than Table B and is designed to give a close check on the condition of the hole at all stages.

Table A.—Column A gives the depths to which the casing has been driven. Column B shows the depth of core after driving, column C the feet of core left in the pipe after drilling is stopped, column D

⁴ Stines, Norman C., *The Prospecting and Valuing of Dredging Ground: Min. and Sci. Press*, Feb. 10, 1906, p. 90.

the core left in the pipe after pumping, and column E the depth of the hole (A—D) after pumping. In column F are given estimated amounts of gold (3⁷ means 7 milligrams of No. 3 gold),⁵ and in column G the amount of material obtained by pumping as estimated from the pan for a rough check on the volume of the core.

The estimates involve, first, calculation of the value of the material represented by each hole and then the combining of the values from the holes, giving weights to each hole based on the volume of ground represented by each.

The value of placer ground is usually expressed in cents per cubic yard and is calculated for each hole by the formula

$$\frac{27 \times \text{cents}}{A \times D} \times \text{milligrams} = \text{theoretical value in cents per cubic yard,}$$

in which

- 27 = cubic feet per cubic yard,
 Cents = value of gold, cents per milligram,⁶
 A = area of drive shoe, square feet,
 D = depth of hole, feet, and
 Milligrams = number of milligrams recovered from hole.

For a standard 7½-inch drive shoe this formula becomes

$$\frac{27 \times \text{cents}}{0.3068 D} \times \text{milligrams} = \text{value in cents per cubic yard.}$$

⁵ No. 1 gold, "color" over 7 milligrams in weight; No. 2 gold, 2 to 7 milligrams in weight; No. 3 gold, colors large enough to be individually counted but less than 2 milligrams in weight.

⁶ This value will depend upon the fineness of the gold as determined by assay; for gold 850 fine, value of gold is 0.06 cent per milligram based on a value of \$20.67 per ounce for gold 1000 fine.

This theoretical value is an approximation, since the volume of material actually recovered in the core (and hence the amount of gold recovered) may vary considerably from that represented by the area of the cutting edge times the depth. Sometimes the core is deficient, and again it may be excessive, the latter condition being caused by loose sand and gravel running into or being drawn into the hole.

Various factors based on experience have been employed to compensate for this error.

Some engineers have used 0.3333 instead of 0.3068 in the formula, with the idea of leaning to the side of safety. Such an arbitrary figure is hardly justifiable from the standpoint of accuracy. The "Radford" factor of 0.27 instead of 0.3068 has been found to yield quite accurate results in a large number of instances, probably due to the fact that unless ground runs into the pipe from the outside, giving excessive cores, gold is apt to be lost through settling to the bottom, being lost in bedrock crevices at the bottom of the hole or lost in panning.

Avery⁷ has discussed the computation of drill-hole data in placer prospecting and suggested the use of factors based upon measurements of core rise for each hole rather than the use of arbitrary factors.

In calculating the average value per cubic yard for an entire property the usual methods of weighted averages are employed, the weight given to each hole being proportioned according to the relative volume of ground represented thereby. Thus, where holes have been spaced equidistant at the corners of squares, each hole is weighted according to its depth. For unequal spacing, the values of triangles or polygons bounded by lines connecting the holes may be calculated. For other cases, such as lines of holes across channel deposits, cross sections may be plotted on each line, the average value for the section being computed from the weighted value of each hole, and this value applied to the yardage extending on each side of the section halfway to the adjacent sections. In special cases the calculations may be varied to suit the conditions by application of ordinary arithmetical principles.

Again, certain erratic high values may be treated differently, depending upon the judgment and experience of the engineer.

For sampling gold placers one hole to every 2 to 4 acres is common practice. In channel deposits lines of holes are usually laid out as nearly as possible at right angles to the direction of the channel, the holes in each line being spaced uniform distances apart, the distances between holes depending in each case upon the judgment of the engineer; the aim of course is to space them so as to obtain an average of the entire property to be sampled. Appraisals of several channel deposits have been based upon holes spaced 150 feet apart on lines as much as 1,000 to 1,500 feet apart. Sometimes holes are required at the corners of 25-foot squares to test the deposit accurately. The characteristics of each deposit will determine the spacing. At the outset it is probable very little will be known concerning these characteristics, so that a tentative layout of holes is made and one hole

⁷Avery, William W., *Computing Drill-Hole Data in Placer Prospecting: Eng. and Min. Jour.*, vol. 129, No. 10, May, 1930, pp. 493-495.

in each line may be drilled first. From this information additional holes are planned, and ultimately it may prove to be advisable to drill other intermediate lines of holes.

In wide blanket deposits holes are usually regularly spaced at the corners of squares, the final distance between holes or the size of the squares being ultimately determined from preliminary drilling at more widely spaced points.

It is of interest to note here that, assuming one drive-pipe hole per acre in ground 50 feet deep, the actual amount of material yielded by the hole for examination is only 1/640,000 part of the total volume represented in the acre of ground.

In sampling gold placers by this method, considerable skill is required in taking the samples, and experience and judgment are needed in interpreting the results. A few cents in the value per yard often may mean the difference between profit and loss in exploiting a gold placer, and often placer operations have failed financially because of misleading results from prospecting.

ACCURACY OF RESULTS

In placer deposits the gold is usually distributed very irregularly; and the recovery from different closely spaced test holes, even though one is drilled alongside the other, will often vary within wide limits. Likewise, the results from larger excavations by test pitting around drill holes may vary widely from the results of drilling.

This range of variation in certain specific instances is well illustrated by Table 1. Column 6 gives the values from the drill holes showing the highest values, and column 7 those from the drill holes showing the lowest values.

Table 2 gives some comparisons between values, as estimated from test drilling and actual recoveries by dredging. In view of the wide discrepancy between individual drill holes and between drill-hole and test-pit results, as illustrated by Table 1, it is remarkable that the actual recoveries in Table 2 have checked within the limits shown.

TABLE 1.—Comparison of drill results with those of test pits¹

1	2	3	4	5	6	7	8	9
Location	Size of casing, inches	Number of holes	Depth of gravel, feet	Size of pit, feet	Maximum value from holes, cents	Minimum value from holes, cents	Average value from holes, cents	Value from pits, cents
Korea.....	4	16	19.8	30 x 30	76.14	0.19	15.22	29.00
Do.....	4	16	13.8	30 x 30	41.03	.45	8.51	-----
Do.....	4	16	19.6	30 x 30	44.36	1.11	10.28	-----
Do.....	4	16	18.0	30 x 30	30.72	.93	6.33	-----
Do.....	4	16	19.3	30 x 30	30.13	2.20	7.31	-----
Do.....	4	16	20.0	30 x 30	51.51	.72	6.85	-----
Colombia.....	2	2	-----	-----	22.6	17.7	-----	25.00
Do.....	1	1	-----	-----	-----	-----	12.20	20.00
Do.....	2	2	-----	-----	68.2	17.0	-----	151.50
Do.....	3	3	-----	-----	34.4	25.7	-----	56.70
California.....	1	1	-----	-----	-----	-----	2.70	3.93
Do.....	1	1	-----	-----	-----	-----	1.90	5.80
Do.....	15	15	-----	(?)	20.6	3.60	8.63	\$ 12.03

¹ Data taken from Gardner, Charles W., Drilling Results and Dredging Returns: Eng. and Min. Jour., vol. 112, No. 17, Oct. 22, 1291, pp. 646-649, et seq., which see for further data.

² Pits around each hole.

³ All pits.

TABLE 2.—Comparison of estimates based on drive-pipe samples and actual dredge recovery¹

1 Example	2 Area, acres	3 Average depth, feet	4 Acreage per hole	5				
				Size of shoe, inches	Area factor used	Value per cubic yard, cents		Recovery, per cent
						Estimate from samples	Dredge recovery	
Oregon ²	121.0	18.0	2.4	7½	0.3333	16.8	15.63	93.0
California.....	118.5	(?)	3.1	7½	.3068(?)	29.88	31.55	105.6
California A.....	173.5	22.5	3.2	7½	.27	6.8	7.82	115.0
California B.....	84.0	44.5	4.2	7½	.27	5.9	6.70	113.0
California C1.....	183.0	51.8	1.5	7½	.30	11.1	9.64	87.0
California C2.....	108.0	60.6	2.6	7½	.30	11.2	9.44	84.0
California C3.....	135.0	56.4	2.3	7½	.30	11.6	11.30	97.0
Average.....						9.48	9.12	96.2
California:								
1918.....	19.94	32.1	1.8			10.39	10.64	102.4
1919.....	20.90	34.4	2.1			9.69	9.22	95.2
1920.....	20.43	29.8	2.9			10.69	14.34	134.0
Natoma.....	421.0			7½	.30	10.25	11.39	111.1
Montana.....	300.0	40.0	3.9			15.83	13.55	85.6
California.....	559.0	22.5	1.0	7½	.27	7.58	9.61	126.7
Do.....	420.5	35.9	2.9	7½	.27	7.25	8.18	112.9

¹ Data taken from Gardner, Charles W., *Drilling Results and Dredging Returns*: Eng. and Min. Jour., vol. 112, No. 17, Oct. 22, 1921, pp. 646-649 et seq.

² Area divided into 8 blocks. Values estimated for individual blocks gave much wider variations from actual dredge recovery than the average, ranging from 49.4 per cent in excess of the estimate to 63.2 per cent less than the estimate. Method of calculation, cross sections on lines of holes.

It is obvious that, provided the work is done carefully, with equal skill and judgment employed in interpreting results, the greater the number of tests drilled the more reliable will be the estimates based thereon.

In dredging gold may be lost in crevices in the bedrock, or recovery of gold from the ground excavated may be poor, in which event the estimates of recovery based upon test drilling will naturally be high unless loss of gold is allowed for correctly. This emphasizes the importance of thorough knowledge of dredging and gold recovery in making up the final estimates of recoverable gold from a prospective dredging area.

The sampling and estimation of placer ground are therefore perhaps the most uncertain and risky of any type of deposit. It is impossible to treat this subject exhaustively within the scope of a paper of this length. The authors' object will have been attained, however, by describing in a general way the methods employed, the principal sources of error, and some of the results obtained by drive-pipe sampling of placers and by emphasizing the importance of obtaining the most skilled and experienced talent for supervising and interpreting the results of a prospecting campaign.

OTHER REFERENCES

The reader is referred for further information on prospecting and sampling placer ground to the following bulletins, each of which contains a bibliography on placer mining.

JANIN, CHARLES. *Gold Dredging in the United States*. Bull. 127, Bureau of Mines, 1918, 219 pp.

WIMMLER, NORMAN L. *Placer-Mining Methods and Costs in Alaska*. Bull. 259, Bureau of Mines, 1927, 236 pp.

CHURN DRILLING

Churn drilling has a wide application in sampling mineral deposits but is limited to the drilling of vertical holes.

Under "Drive-pipe method" it has been stated that the pipe is driven first and then the material inside the pipe is drilled out and pumped to the surface as a sample. In churn-drill sampling the hole is drilled ahead of the pipe (if any pipe is used), the cuttings from each "run" (usually 5 feet in length) of the tools being pumped to the surface as the sample. Obviously the drive-pipe method is applicable to loose or soft material which would cave into a hole drilled by churn-drilling methods and into which the pipe can be driven without excavating ahead of it, whereas the churn-drill method is suited to firm rock, which may range from soft to very hard and into which pipe could not be driven ahead of excavation of the ground by drilling.

The sources of error in churn-drill sampling, the method of collecting samples, and the precautions to be observed are similar, whether the drilling is done by hand, by portable rigs, or by standard rigs, and will be discussed under "Portable rigs," which are of the type most widely employed for exploring ore deposits.

HAND DRILLS

JUMP DRILLS

The simplest form of churn drill is a long piece of steel or a pipe with a chisel bit on the end ("jump drill") which may be raised and dropped by hand. Two or more men alternately raise and drop the tools, rotating them a part turn each time. The use of drills of this type is limited to drilling very shallow holes in loose or partly consolidated material or in soft rock.

SPRING-POLE DRILLS

Intermediate between this and the lighter power-operated rigs is the spring-pole drill (fig. 2), which has been used in remote regions and by prospectors whose resources will not permit the employment of power-driven equipment, and for drilling shallow vertical holes in loose material or soft rock, such as shale, slate, some sandstones, or decomposed igneous rocks.

With this equipment two or three men can work to 100 to 150 feet in depth and in exceptional instances where the rock is soft but not fissured up to 200 or 300 feet has been drilled. Casing is put down to bedrock if possible by driving and using earth augers or sand pumps to remove the material inside the pipe. When ready to drill the drill tools, which may be screwed onto a string of pipe as shown in Figure 2 or may be secured to a cable, are hung from the spring pole as shown, and the tools are raised and dropped by springing the pole up and down, the play ranging from 6 inches at the surface to 18 inches at the bottom of the hole. After 2 or 3 feet is drilled the tools are hoisted by the windlass, and the cuttings constituting the sample are brought up with a sand pump. When the sand pump

brings up only clear water the tools are put back and drilling is resumed.

For sampling ground where hand-operated equipment may be used it is an open question and must be decided for each individual case whether it would not be more economical, quicker, or more desirable from the standpoint of accuracy to test pit the ground rather than to drill. For very shallow depths where water is not present test pitting may be nearly as cheap as drilling and in most cases would supply samples that are much more accurate. Where water is present in appreciable quantity in the ground to be sampled test pitting would have obvious disadvantages in comparison to drilling methods.

POWER DRILLS—PORTABLE RIGS

APPLICATION

For more elaborate drilling campaigns and for depths up to around 1,000 feet more or less, the portable churn-drill rig is widely employed for exploring and sampling ore deposits by drilling from the surface. These rigs are driven by steam engine, gas engine, or electric power and are mounted on wheels or caterpillar traction so that they may be readily moved under their own power from one drilling site to the next. This type of drill has been employed extensively for sampling the porphyry copper deposits of the Southwestern States, in

the tri-State zinc and lead district, and elsewhere. The size of hole drilled usually ranges from that drilled by a 10 or 8 $\frac{1}{4}$ inch bit to 6 $\frac{1}{4}$ or 4 $\frac{1}{4}$ inches for finishing the hole, though sometimes larger drills are employed. Portable rigs are also employed for drilling vertical holes from underground stations.

Portable rigs differ principally from the standard cable-tool rigs in that they are mounted on self-propelled traction and have a mast mounted on the machine, which may be raised for drilling and lowered while moving instead of employing a separate derrick, as with standard rigs. They are not built for as heavy work or as deep holes as are standard rigs.

EQUIPMENT

The essential parts of these machines are: Source of power, prime mover actuating a walking beam and a hoist through suitable clutches, drill line (hemp or wire rope), and string of tools consisting of a rope socket (between the rope and string of tools proper), set of jars, drill stem, and bit. For pumping out the hole a sand

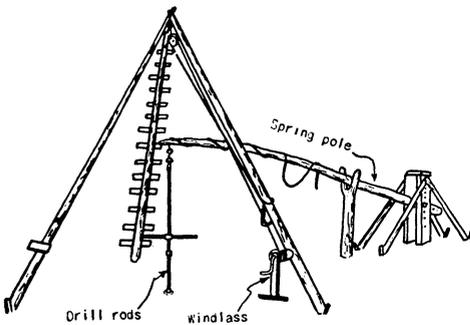


FIGURE 2.—Spring-pole drill

pump is required, and sand-pump lines operated by the hoist mentioned above are necessary. Other equipment consists of fishing tools, driving clamps, and apparatus for driving and pulling casing pipe, drill-tool wrenches and jacks, and various small tools. Portable churn-drill equipment is described in detail in the catalogues of the manufacturers, and their description here would be superfluous.

ARRANGEMENT OF HOLES

For surface drilling the tract to be explored is usually laid out in the form of a checkerboard if the area is wide; and holes are drilled at the corners of the squares, the spacing of the holes or size of the squares being determined by the purpose of the drilling, nature of the deposit and uniformity of mineralization, the judgment of the engineer, and the money available. Often the first holes may be spaced farther apart than the ultimate spacing, the latter being determined by the results of the preliminary holes. Where the area to be explored and sampled is long and narrow the holes may be drilled on a series of lines running at right angles to the long dimension thereof.

The arrangement and spacing of holes are usually planned by the geological department if the work is being done by a large company, and the sampling, inspection of sludge, etc., are under the same supervision.

ACCURACY OF RESULTS

Since drilling, as briefly discussed in this paper, is for the purpose of obtaining samples of ore in place, speed and cost of drilling are secondary in importance to accuracy of the samples. The degree of accuracy required in sampling may differ in different instances, but in any event it is important to bear in mind that accurate sampling is sought rather than feet of hole.

CAUSES OF INACCURACY

The principal inaccuracies in results from drill sampling are due to: (1) Caving of the hole with attendant salting of the samples; (2) salting of the sample due to the knocking off of pieces from the side of the hole above the sample by the drilling tools, pump, or bailer during lowering and raising, and by the rope slapping against the sides of the hole; (3) settlement of the heavier minerals to the bottom of the hole and failure to remove them with the sample to which they properly belong; (4) escape of part of the sample through cracks and crevices in broken and fractured ground or into vugs; (5) sliming of the sludge and floating off and loss of the lighter material with the drilling water; (6) failure to clean the hole properly before drilling is resumed after casing pipe is driven; (7) partial solution of ore or gangue minerals in the drilling water decanted from the sample.

Since water is used in churn drilling the samples are wet and must be dried, usually at the drill. In drying, a frequent source of error is introduced, due to burning of the samples, which changes

their chemical composition and weight; the most common change is that from sulphides to oxides of the metals, an error introduced in treatment of the sample rather than in the taking thereof.

Caving of the hole requires that it be cased with pipe or well tubing. In any event it is a safe precaution when the top of the ore is reached to case the hole to prevent contamination by capping material of the samples to be taken from the ore. This of course is imperative where soil, glacial drift, or other loose material is passed through.

In anticipation of possible caving of the hole it is well to start with a large bit to allow for one or more strings of casing and tools. The large sizes of bits often will drill faster, and the cost of a large hole is not necessarily more than that of a smaller hole. The practice followed at Miami⁸ is quite common. There the holes were started with a 10-inch bit, succeeded by the following sizes of casings and bits: 7 $\frac{5}{8}$ -inch casing; 7 $\frac{5}{8}$ -inch bit followed by 6 $\frac{1}{4}$ -inch casing; 6 $\frac{1}{4}$ -inch bit followed by 4 $\frac{1}{4}$ -inch casing; 4 $\frac{1}{4}$ -inch bit. It may not be necessary, of course, to use all these sizes and often is not; on the other hand, if the 4 $\frac{1}{4}$ -inch hole caves badly before the bottom of the ore is reached it may be necessary to ream the hole and reset the casing to a point below the caving ground.

After each driving of the casing it is important to pump the hole clean before drilling is resumed.

Salting of the sample by pieces falling from the upper part of the hole will always occur to some extent, but not necessarily enough seriously to vitiate the results. Casing the hole will of course minimize this, since caving into the hole will then be limited to that portion below the casing. It is a matter of judgment therefore as to when casing should be used to prevent salting (either raising or lowering value of sample) from this cause.

Failure to remove all the cuttings after each run will result in salting the next sample with the material left in the bottom of the hole. If the ordinary dart-valve bailer is used, the bottom of the bailer will never rest on the bottom of the hole, as it is kept off a distance equal to the height of that portion of the dart below the bottom of the bailer. By stirring the cuttings and diluting them with water most of them may be recovered; and unless there is considerable heavy mineral, such as metallic sulphides, magnetite, free gold, and native copper, salting due to this cause need not as a rule be serious.

For recovering heavy minerals from the bottom of the hole a suction sand pump, having a valve with a stem to which the rope is attached, running the length of and out the top of the pump, will usually give better results than a dart bailer.

One source of error most difficult to overcome in churn-drill sampling is that caused by open fissures in the rock, into which the cuttings escape and are thus lost. This can not be helped entirely during drilling through the fissured portion, although it may be kept to a minimum by drilling with a thick sludge, using the least possible amount of water. Having passed through the fractured ground deeper cuttings may be prevented from escaping from the

⁸ Bowen, H. P., Data on Miami Churn Drilling: Eng. and Min. Jour., vol. 97, No. 18, May 2, 1914, pp. 903-904.

hole below by casing or by cementing the hole, allowing the cement to set, and then drilling through it.

Floating off and loss of lighter material from the sample can be guarded against by providing proper settling facilities so that the slimes may have ample time for settling before the water is decanted and by not decanting completely, but leaving the last of the water to be evaporated in the drying process. If loss is suspected after all precautions have been observed, it may be advisable to take occasional samples of the overflow water for analysis. Sometimes alum, lime, or wood ashes may be added to hasten settling of the slime.

Solution of some of the minerals and their partial loss by passing off with the wash water, while not commonly serious, may sometimes occur. Thus, if acid water is used there may be appreciable solution of a limestone gangue or of some oxide copper minerals. The use of neutralizing reagents to counteract this action would introduce another error, in that upon evaporation of the water during drying some of the reagent would remain with the sample.

DRILL-HOLE LOGS

Since a drilling campaign costs considerable money it is desirable to obtain therefrom all the information possible at the time of drilling, and to this end the data should be carefully and painstakingly recorded. Data are of three principal types: (1) Drilling; (2) geological; and (3) chemical analysis.

The drilling data furnished by the drillers' reports will not only give a record of speed of drilling, materials and supplies used, and other information from which the cost sheets are made up, but, if the driller is experienced, important information on the character of the ground which may be useful for anticipating probable future mining conditions. Geological data will show the petrographic and mineralogical character of the deposit at each horizon, useful in locating future holes and for working out geological structure and anticipating future milling and metallurgical problems.

The chemists' data will of course give the assay value of the samples upon which are based the estimates of grade of ore to be mined and will serve as a guide to future metallurgical treatment.

Figure 3 shows a condensed churn-drill log used at Utah Copper and Figure 4 a graphic churn-drill-hole log used at Chino. Figures 5 and 6 show forms for drill reports and for hole records. These are applicable either to churn or diamond drilling. More elaborate and detailed report forms have been described and illustrated by Harding.⁹

It is common practice to assign a sampler to look after the sampling of each hole (sometimes if the rigs are close together one sampler can look after more than one hole at a time). For this work it is desirable that if the sampler is not a technical man he should at least have a working knowledge of mineralogy and petrology so that he can, by panning the sludge, note the rocks passed through and the principal ore and gangue minerals.

⁹ Harding, James E., *Information Obtainable from Churn-Drill Sampling: Eng. and Min. Jour.*, vol. 117, Mar. 29, 1924, pp. 523-527.

EXAMPLES OF CHURN-DRILL SAMPLING PRACTICE

TRI-STATE ZINC AND LEAD DISTRICT

Churn drills have been used for many years to prospect the flat-lying, shallow deposits in this district. The practice has become well standardized. The ore occurs in irregular sheets and in shear zones in brecciated flint and dolomite beds. Diamond drilling is not

CHURN DRILL HOLE NO. 134 Coordinates S. 973.75 W. 2396.39 El. 6708.0				
Casing Left in Hole	Casing Record	Adopted Assays	Character of Material	General Remarks Minerals
		CU		
23" Casing 76 with shoe 20" " 100" " 1 1/2" " 323" " 12 1/2" " 130" " 200	23" casing 20" casing 1 1/2" casing 12 1/2" casing	Elev. 6690 0.88%	Silic. Porph. Silic. Porph. Silic. Porph.	8-14-28, Spudded in 23" Bit Pyrite, chalcopryrite, chalcocite, covellite, bornite, molybdenite. Chalcopryrite, pyrite, chalcocite, covellite bornite, molybdenite. Chalcopryrite, pyrite, bornite, chalcocite.
400		Elev. 6389	Silic. Porph.	Covellite, molybdenite. Chalcopryrite, pyrite, chalcocite, bornite, covellite, molybdenite.
600		1.16%	Silic. Porph.	Chalcopryrite, pyrite, chalcocite, molybdenite, bornite, covellite.
800			Grey Porph.	Chalcopryrite, chalcocite, pyrite, covellite, bornite, molybdenite.
1000		Elev. 5833	Dark Porph.	Chalcopryrite, chalcocite, pyrite, covellite, bornite, molybdenite
1200		0.81% Elev. 5573 0.52% Elev. 5543	Limestone Dark Porph.	Altered, chalcocite, chalcopryrite, pyrite, covellite, bornite, molybdenite Chalcopryrite, chalcocite, pyrite, covellite, bornite, molybdenite.
1400		0.79% Elev. 5468 0.94% Elev. 5423 0.80%	Limestone Mixed zone of limestone and porphyry Dark Porph.	Black, large amounts of chalcocite in limestone in porphyry primary minerals only, chiefly chalcopryrite, bornite, pyrite, unusual amount of covellite. In limestone chalcopryrite, chalcocite, covellite. In porphyry, chalcopryrite, bornite, pyrite, molybdenite, 2-10-29 - Hole completed - Depth 1475'
		Elev. 5233		

FIGURE 3.—Condensed churn-drill log

suited to this type of ground. In some places the ore occurs at several horizons, but more often there is only one productive horizon. The ore minerals—sphalerite and galena—are found as irregular seams cementing fractures and as masses filling vugs and cavities, associated with flint, calcite, and highly siliceous dolomitic limestone.

Samples are usually taken of 2 or 3 foot runs after the ore is reached, the sludge being split with a Jones splitter. Exploration is usually started by drilling several rows of churn-drill holes across the tract from north to south or from east to west at intervals of

200 to 400 feet until a favorable area has been found, when the holes are drilled closer together.¹⁰ The earlier holes on the tract described by Netzeband were drilled $5\frac{7}{8}$ inches in diameter, but later the standard size was changed to $6\frac{1}{4}$ inches. Logs were kept of all holes and assays made of all values above 2 per cent of blende or 1 per cent of galena; values below this are recorded as "shines." On the 200 acres of No. 1 mine 128,204 feet of drilling was done, all on contract at prices ranging from \$1 to \$1.25 per foot, depending upon the character of the ground. Estimates of ore reserves are based on

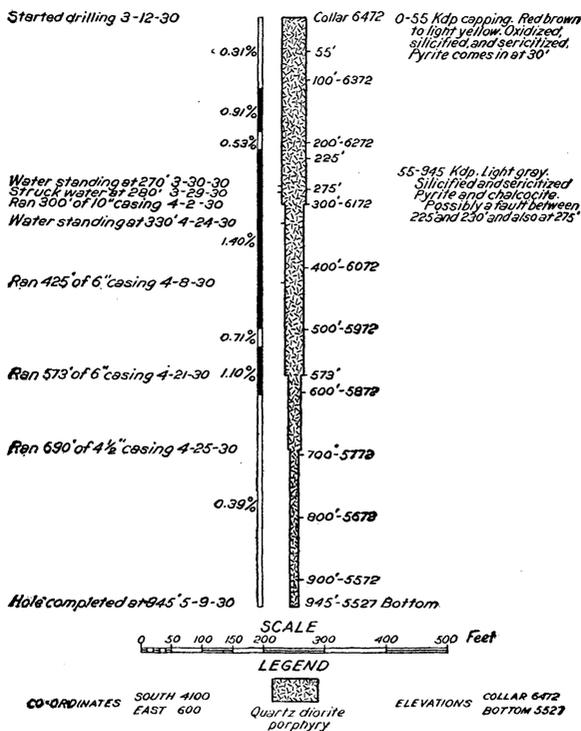


FIGURE 4.—Graphic churn-drill-hole log

careful analyses of the churn-drill records. Past experience has shown that an ore body mills out about 10 per cent better in grade than the estimate.

At mine No. 2,¹¹ after the mine was opened up and the general character of the ore deposit had been determined, drilling operations were concentrated along the trend of the shear zone, which was very difficult to drill before the water was drained off, and in consequence the cost of drilling was high. The early drilling cost \$1.50 per foot, and many of the holes had to be finished on company time at \$25

¹⁰ Netzeband, Wm. F., Method and Cost of Mining Zinc and Lead Ore at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Okla.; Inf. Circ. 6113, Bureau of Mines, 1929, 11 pp.

¹¹ Netzeband, Wm. F., Method and Cost of Mining Zinc and Lead at Mine No. 2, Tri-State District, Picher, Okla.; Inf. Circ. 6121, Bureau of Mines, 1929, 11 pp.

The first drilling on a new piece of ground is on a hit-or-miss basis. After ore has been found, it is followed and delimited by closer spacing of the holes. Recent practice has been to drill on cross-section lines normal to the known trend of the ore-runs.

On the property of the Missouri-Kansas Zinc Corporation 640 holes with a total footage of 151,352 feet have been drilled.

Underground churn drilling has been done when the overlying surface was inaccessible because of water ponds, slime ponds, tailing piles, or other obstacles. The average cost of underground drilling has been \$1.923 per foot.

At the Barr mine¹³ the ore bodies are on a 60-acre tract around the flanks of a limestone dome in a brecciated zone in massive flint at three horizons.

The 60-acre tract upon which this mine is located has been prospected by 205 churn-drill holes. The first drilling is laid out in rows of holes spaced at more or less regular intervals across the property and drilled primarily to locate the low spots in the structure where the ore is localized in this district. This drilling is followed by closer drilling where good ore or favorable conditions warrant.

No samples are taken after mining operations are started to determine the extent of the ore bodies. Estimates are made after a sufficient number of churn-drill holes are drilled into an ore body to prove it up and prior to beginning mining operations.

The drill cuttings are carefully preserved and logged, and such samples as show as much as 1 per cent of ZnS and PbS are assayed. From a careful analysis of these assays an ore body is blocked out. An estimate is then made of the tonnage available and the probable extraction. Experience has proved that these estimates are lower than actual mill tests run.

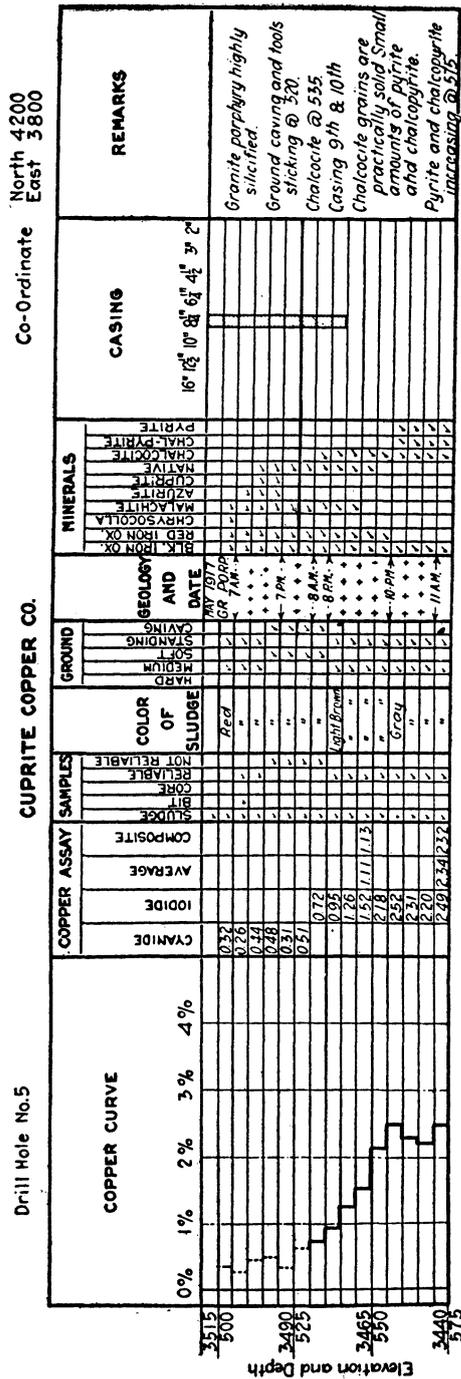


FIGURE 6.—Hole record as shown in standard form report

¹³ Keener, Oliver W., Method and Cost of Mining at Barr Mine, Tri-State Zinc and Lead District: Inf. Circ. 6159, Bureau of Mines, 1929, 9 pp.

Figure 7 shows a number of drill holes and an ore body sketched in. The following is the method of ore estimation used:

Hole No.	Ore face	Height	Average assay		Products assay, feet	
			ZnS	PbS	ZnS	PbS
26	180-200	20	5.70		114.00	
27	150-200	50	4.40		220.00	
28	172-188	16	4.05		64.80	
32	165-200	35	4.04		141.40	
33	180-205	25	6.14		153.50	
37	155-175	20	7.55	2.00	151.00	40.00
40	154-218	64	7.34		469.76	
49	170-205	35	3.91		136.85	
50	165-205	40	10.40		416.00	
51	172½-217	44½	21.40		952.30	
53	170-187½	27½	15.30	1.02	420.75	28.05
54	190-220	30	9.25	1.19	277.50	35.70
55	190-200	10	2.14		21.40	
		13)417			417)3,539.26	103.75
		32			8.49	.25

\$40 per ton Zn concentrates.

\$90 per ton Pb concentrates used.

8.49 × 85 per cent extraction × \$40 = (net) ----- \$2.89

0.25 × 90 per cent extraction × 90 = ----- .23

Value per ton ----- 3.12

Less 8 per cent royalty ----- \$0.25

Mining and milling cost ----- 1.60

1.85

Net value per ton ----- 1.27

Area 17,040 × 32 feet = ----- 41,945

13 cubic feet per ton

Less 10 per cent pillars ----- 4,194.5

Net tonnage ----- 37,750.5

Net tonnage × \$1.27 = \$47,943.14, value ore body.

Additional estimates are prepared from time to time as drilling progresses.

At Crestline¹⁴ the ore bodies are closely associated with shale pockets in boulder ground 140 to 190 feet below surface.

After the character of the ore bodies was determined by the early drilling it was found that this property was adapted to what is locally known as "shale drilling." By this method the contact between the Cherokee shale and the Boone formation is determined by a series of holes which are drilled through the shale but only about 10 feet below it. Holes are drilled at regular intervals, usually on 200 or 400 foot centers. From the data obtained the base of the shale is contoured, and deep drilling is concentrated around the edges of the deep shale pockets. By this method considerable territory can quickly be eliminated as unfavorable.

The shale holes are drilled for 40 to 60 cents per foot, depending upon the length of move and the topography of the tract. Deep holes are drilled for \$1.10 to \$1.50 per foot. Much of the drilling cost \$1.50 because of the bowldery character of the ground, which made it difficult to drill.

¹⁴ Netzeband, William F., Method and Cost of Mining Zinc and Lead at No. 3 Mine, Tri-State District, Crestline, Kans.: Inf. Circ. 6174, Bureau of Mines, 1929, 10 pp.

MESABI IRON RANGE

On the Mesabi iron range the ore bodies occur as large flat or basin-shaped deposits lying at comparatively shallow depths, to which prospecting from the surface by vertical holes is well adapted.

Diamond-drilling equipment was used for drilling these holes, but for much of the footage the drills were used as churn drills, employing a chopping bit and a water jet for drilling through surface material and in soft ore, and only using the diamond equipment for drilling through seams of hard ore, taconite, and quartzite. The ore is mostly soft hydrated hematite and limonite, too soft to core well, but some bands of hard ore occur.

Churn drilling with this equipment, often termed "Mesabi-type" equipment, differs somewhat from the practice with the walking-beam type of churn-drill rig discussed previously. The method employed involves chopping with a chisel bit connected to the bottom of a string of hollow drill rods reaching above the collar of the hole, a stream of water being pumped down through the rods and bit; the bit has a hollow shank with openings or perforations for the outlet of water in the sides of the bit, pointed so that the water jets against the bottom of the hole. A casing is kept close to the bottom of the hole; and the

water, rising around the drill rods and inside the casing, carries the cuttings to the surface. A "tee" is screwed on the top of the casing, the drill rods being run through the straightaway of the tee and the cuttings passing out through the branch. A pipe screwed to the branch directs the water and cuttings into barrels where the cuttings are settled out to form the sample. Each barrel has a stoppered hole

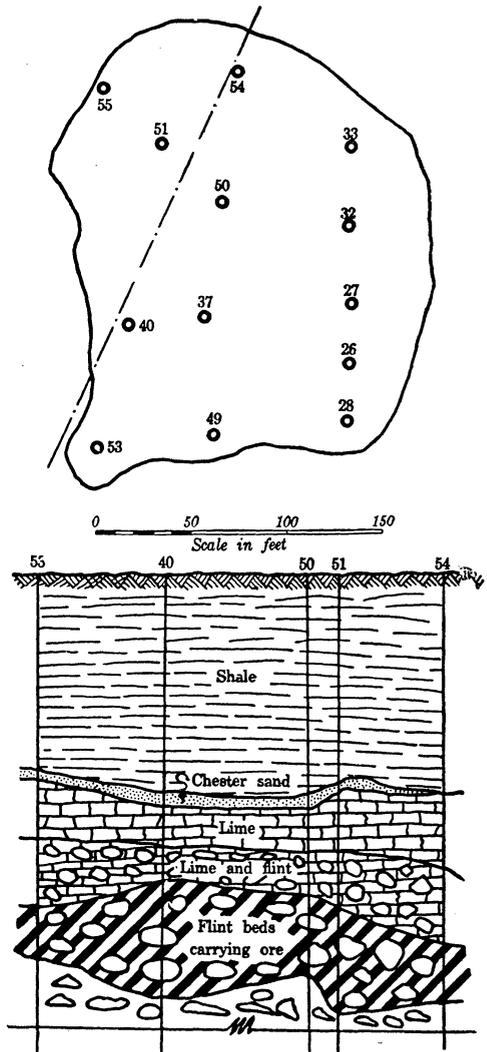


FIGURE 7.—Drill holes and ore body, projected on section at A-A, Tri-State district

about 8 to 10 inches above the bottom through which the water may be drawn off after the sludge has settled. The sudge forming each sample from the barrels is scraped out into a tub and settled further, the water is decanted or siphoned off, and the sample is dried. The usual length of sample is 5 feet, and all the cuttings from these 5 feet are collected and dried. The number of barrels required for each sample depends upon the nature of the ore. It is best, however, from a precautionary standpoint, not to refill any of the barrels during a run but to provide enough barrels so that all the sludge and water may be collected in them without refilling.

In churning with the Mesabi rig, a rope is passed from the rods over the crown pulley and brought down to the rotating "niggerhead" on the machine. Several coils of the rope are wound on the "niggerhead" and by alternately tightening and loosening the coils the rods are raised and dropped.

In passing through glacial drift a 3-inch pipe is ordinarily used for casing, although larger pipe may be used if this material is deep. The casing pipe is kept close to the bottom of the hole so that the sample will not be contaminated by material dropping in from above. While the casing is being driven and washed down the wash water and cuttings are of course discarded. Care must be taken after each driving of the casing to wash the hole clean before cutting of samples is resumed.

Sometimes the water will come up outside the casing instead of inside; and in this event the sample may become contaminated, or more likely, some of the sample will be lost. Packing around the outside of the pipe with rags will sometimes stop this. Hay or sawdust introduced in the hole may stop it from the bottom.

One of the principal precautions required in this drilling is to settle the samples properly. There are examples on record of very serious errors in sampling, due to failure to collect all the sludge, which were not discovered until too late; that is, until the mine was developed. The loss of light material with the wash water resulted in high sludge assays, the iron minerals having been concentrated in the sludge.

It may be said that sampling of Mesabi range iron ores by the methods employed will, if carefully done, give very reliable results. The samples are dried, and the assays will then give "dried" rather than "natural" iron. Since water must be used in drilling "moisture" can not of course be determined from drill samples. To determine moisture other means, such as test pitting, must be employed to obtain the samples.

COPPER MINES

Churn drills of the portable walking-beam type have been extensively employed for prospecting and sampling the disseminated copper deposits of the Southwest. Used successively at the Utah Copper, Inspiration, Miami, Ray, Chino, Cananea, Sacramento Hill, and other localities, a more-or-less standardized technique has been developed for sampling deposits of this nature.

Churn-drill sampling is particularly well adapted to deposits of this type; and if care is used in drilling, sampling, and treatment of samples, a high degree of accuracy is possible, due to the comparatively uniform distribution of the copper minerals. Joralemon¹⁵ states that "with careful work, involving casing below all caving ground, assays from churn drilling can be depended upon in nearly all cases to within 0.1 per cent copper."

The methods employed at these mines have been described in considerable detail by E. R. Rice.¹⁶

Rice advises that in starting a drill-exploration campaign in a deposit of this type, churn drills rather than diamond drills should be used. Churn drills can be used in formations difficult to drill with diamond drills, in both hard and soft ground, regardless of whether the ore will "core" or not. After the general character of the formations has been determined by churn drilling the decision can then be made whether to continue with the churn drill or to use a diamond drill or some other rotary type.

Samples are taken every 5 feet, as a rule, after the ore is reached. They are bailed from the hole with an ordinary dart bailer or by a suction sand pump, and dumped into a launder leading to a splitter which automatically cuts out one-half, one-fourth, one-eighth, or one-sixteenth of the total sludge and water as a sample. (Fig. 9.) A 50 to 75 pound sample is usually sought. If the hole is not caving a 4¼-inch hole will give about a 70-pound sample from a 5-foot run. An 8¼-inch drill will give about 350 pounds, so that an eighth or one-fourth split will give about the final sample desired.

L. S. Breckon¹⁷ describes a cutter used by the Utah Copper Co., with which is employed a small gyratory crusher to break down the coarser cuttings before splitting.

The samples are caught in tubs and dried over an open fire on a sheet-iron stove or steam drier. Rice, in the article previously mentioned, describes a drying pan with a false bottom arranged for admitting steam from the drill boiler between the two bottoms. In drying over an open fire or on a stove care must be exercised to avoid burning the samples.

At some of the operating mines standard or modified standard churn-drill rigs are employed and larger and deeper holes are drilled than with the lighter portable rigs. Thus at the Utah Copper Co. mine holes are started with a 26-inch bit if any considerable depth of hole is anticipated.

UTAH COPPER MINE, BINGHAM CANYON, UTAH

The following account of the methods employed at this property is taken from Information Circular 6234, Bureau of Mines.¹⁸

Prior to the advent of steam shovels at the Bingham property many miles of underground work had been driven by the Boston Consolidated and other

¹⁵ Joraleman, Ira B., Sampling and Estimating Disseminated Copper Deposits: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 611-612.

¹⁶ Rice, E. R., Churn Drilling of Disseminated-Copper Deposits: Eng. and Min. Jour., vol. 111, June 25, 1921, pp. 1058-1064; vol. 112, July 2, 1921, pp. 11-18.

¹⁷ Breckon, L. S., Sampling Churn-Drill Sludge at the Utah Copper Mine: Eng. and Min. Jour., vol. 126, No. 13, Sept. 29, 1928, pp. 491-492.

¹⁸ Soderberg, A., Mining Methods and Costs of the Utah Copper Co., Bingham Canyon. Utah: Inf. Circ. 6234, Bureau of Mines, 23 pp.

companies operating in the district, so that in the beginning most of the proved tonnage was developed by this means. After the organization of the Utah Copper Co., both diamond drills and churn drills were used in exploration. Unsatisfactory results were noted from the very few diamond-drill holes drilled in the porphyry, as there was such a wide variation between core and sludge samples that diamond-drill work was discontinued and prospecting continued solely by means of the churn drill. This prospecting has, with the exception of short intervals, been continued to the present day. Figure 8 shows the relative development of the ore body by underground workings and churn-drill holes.

Two types of churn drills are used at present: One, a modified standard rig for drilling holes to depths of 1,000 feet or less, and two standard oil rigs for drilling deeper holes; all are operated by electric power. When any considerable depth is anticipated, the holes are started with a 26-inch bit and a 26-inch stovepipe casing is carried down 80 or 100 feet, depending upon the extent to which the ground caves. At this point a 23-inch casing is inserted and carried down until it is "frozen" in the hole by caving ground. Underreaming is often resorted to when the casing will not follow the bit. Smaller sizes of casings are inserted as required, and in some instances casings

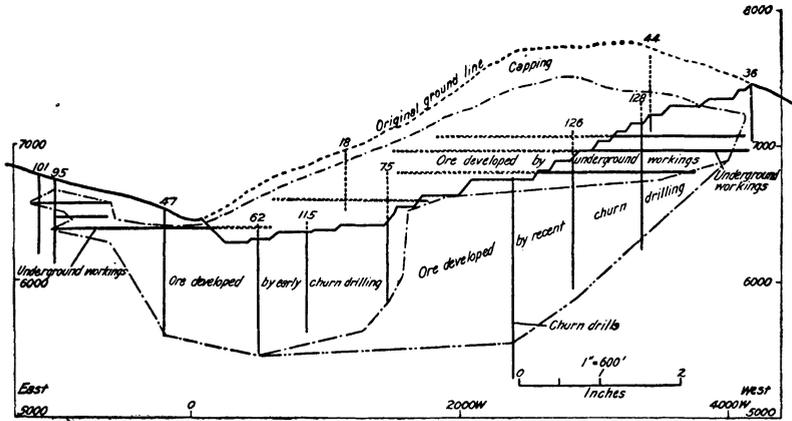


FIGURE 8.—Illustrative section showing comparative ore development by underground workings and churn drills, Utah Copper Co.

as small as 4 inches have been used. It became a rule early in this work not to run an open hole more than 50 feet in advance of the casing, to eliminate the danger of salting the bottom samples by cavings from the bore of the hole above, and to minimize the need of underreaming, consequent delays, and increases in drilling cost. It is thoroughly understood that an accurate sample is wanted, even at the sacrifice of depth of hole if necessary.

All churn drilling is done by contract on a footage plus labor and supply cost basis. Following is a cost statement of one of the recent holes completed:

Cost of drilling hole with Standard electric rig

[Spudded in with 23-inch casing; elapsed time, 455 days; average progress per day, 3.23 feet; average drilling progress per day, 3.92 feet; total footage drilled, 1,468.8]

	Cost	Per foot
Moving and setting up drill: Labor, power, and water supply.....	\$1,667.08	\$1.14
Actual drilling: Amount paid contractor plus labor and supplies.....	19,578.33	13.33
Casing hole: Labor and supplies.....	4,468.47	3.04
Sampling and assaying.....	4,367.55	2.98
Total.....	30,081.43	20.49

Sampling at the churn drills is under the supervision of the geological department. A sampler is assigned to each rig.

Samples are taken every 5 feet and the cuttings removed by a suction bailer. The content is discharged into a launder, passed over a screen 8 to 10 feet long, from which the large sizes are sent through a small gyratory crusher, then combined with the undersize, and the whole is passed into an inverted cone-shaped tank. Sludge in the tank is agitated for 20 minutes by a mechanical agitator that revolves near the bottom, aided by compressed air that enters through a 1-inch line near the bottom of the tank. From here the sludge passes to a cutter so constructed that three separate samples are obtained. One is sent to the mine-assay office, one to the assay office at the mills, and the third and smallest sample is cut down to 5 pounds and placed in a 5-gallon wet-sample can. This last sample is used as a part of a composite sample for every 100 feet drilled to serve as a check on the 5-foot samples; it is also used for making experimental flotation tests. Samples taken for immediate analysis are thoroughly dried on a large sheet-iron stove, care being taken not to burn or break down the sulphide. A 2-pound specimen is saved of every 25 feet of the hole drilled in porphyry and of every 5 feet when the hole is near a porphyry-quartzite contact. These specimens are examined by the geologist to determine the minerals contained and character of the rock.

Daily reports are made and a complete log of the hole is kept by the geological department. (Fig. 3.) A record of the assay returns from the mine and mills is kept, and when a variation of over 0.05 per cent exists between the mine and mill assays, duplicate analyses are made of the pulp and averaged for the adopted assay.

Owing to the irregularity of the surface, holes were not drilled at the actual intersections of predetermined squares, but are drilled as nearly as possible at the corners of equilateral triangles. Where values are consistent, a spacing of 400 feet is considered safe, but as the limits of the ore body are approached, holes 200 feet apart have frequently been drilled.

CHINO MINES, SANTA RITA, N. MEX.

Drill-sampling practice at the Chino mines, Santa Rita, N. Mex., has been described by Thorne¹⁹ as follows:

Churn drills of the "spudder" type, capable of drilling to a depth of 500 feet, were used on the first holes; later, churn drills with a range of 1,200 feet were used. The first drills were all actuated by steam power. At present drill rigs powered by gasoline engines are in operation. In the beginning holes were drilled at 100-foot centers on coordinate lines. This distance was increased to 200 feet, which gave dependable data for estimating tonnage and grade of ore.

The prospect holes drilled in later years have averaged about 900 feet in depth. They are usually started with a 13-inch bit and continued with this size until caving becomes serious. Casing is then inserted and drilling is continued with a small-diameter bit. In some instances tools as small as 3 inches in diameter are used. Underreaming is not used except as a last resort. The hole is cased when ore is first encountered, at any depth that caving is apparent or suspected, and again upon leaving the ore zone. Every precaution is taken to obtain an accurate and representative sample of such depth interval as was selected for the hole.

During the first half of the prospect-drilling campaign samples were taken at intervals of 3 feet; however, samples at every 5 feet as taken now are proving satisfactory. All of the sludge from a 5-foot interval is bailed out and passed through a large Jones-type sampler equipped with splash boards and a mixing chamber. See Figure 9.

The sample is carefully split, one part being preserved in the wet state and the other part dried.

The wet sample is saved until assay returns are received on the dry part. Composites of wet samples in the ore bands are then made up and metallurgical tests are run on them.

¹⁹ Thorne, H. A., Mining Practice at the Chino Mines, Nevada Consolidated Copper Co., Santa Rita, N. Mex. : Inf. Circ. 6412, Bureau of Mines, 1931, 28 pp.

The dry sample is divided up and used for the following purposes:

- (a) Sample for mine assay office.
- (b) Sample for mill assay office.
- (c) Sample washed clean of sludge and dried for geological department.
- (d) Remainder of pulp retained for possible future use.

The assay laboratory at the mine determines the copper by the potassium-iodide method, whereas the mill assay office runs them electrolytically. The returns from these two assays usually check within very close limits, but should a discrepancy exist the samples are rerun by each office.

The sample washed clean of sludge is used in two ways—one part, in a small glass vial, is sent to the geological department where it serves for rock determinations; the other part is used on the sand board. The sand board is a long, narrow strip of wood on which the particles of drill cuttings are glued; the cuttings from each interval of the hole are put on to scale. This is kept up to date as the hole is drilled. The bottle samples are kept in specially constructed racks and are thus always available for future reference.

Daily reports are made by the drill crews as well as the assay office. Records

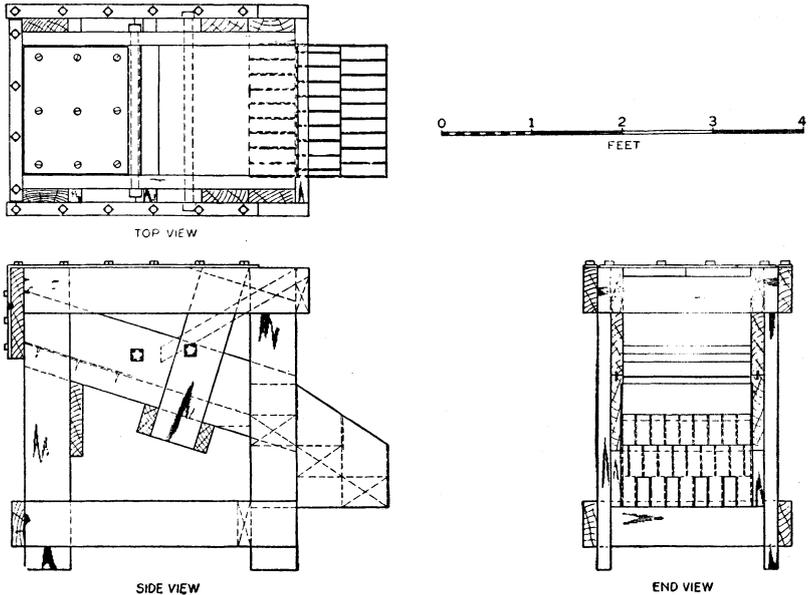


FIGURE 9.—Sketch of sampler for material from drill holes, Chino, N. Mex.

are kept posted daily and upon the completion of a hole a log is constructed by the geological department. (Fig. 4.)

Following is a copy of the progress record for two holes recently drilled, one with a steam rig and the other with a machine equipped with a gasoline engine:

Progress report of two holes drilled by different power

	Drill hole 824, steam	Drill hole 825, gaso- line
Elapsed time.....days.....	66	45
Shifts worked, total.....	112	83
Shifts moving and setting up.....	71	3.66
Shifts drilling.....	102.08	76.42
Shifts casing.....	2.92	2.92
Footage drilled.....	945	965
Average progress per shift, total shifts.....	8.44	11.62
Average progress per shift, drilling.....	9.25	12.62

Following is a segregated cost statement for the same two holes:

Segregated cost statement for two holes drilled by different power

	Drill hole 824, steam		Drill hole 825, gasoline	
	Total cost	Cost per foot	Total cost	Cost per foot
Moving and setting up, labor.....	\$159.51	\$0.168	\$84.65	\$0.088
Drilling, labor.....	2,326.16	2.463	1,764.26	1.828
Casing, labor.....	66.46	.070	67.34	.070
Fuel.....	421.44	.446	221.35	.229
Supplies.....	67.30	.071	42.06	.044
Sampling and assaying.....	475.73	.503	409.79	.487
Totals.....	3,516.60	3.721	2,649.45	2.746

Permanent records are made of each drill hole showing the assay of each interval sampled, color or sludge, principal minerals, and the rock formation as reported by the geological department. Accompanying each set of sheets for a finished hole is a summary sheet showing depth and average assay of capping, or ore, and of any waste bands. The elevations of the top and bottom of each band of ore or waste are shown. Cross-section sheets are also developed on a scale sufficiently large to record the individual assays.

INSPIRATION MINE, INSPIRATION, ARIZ.

The methods employed at Inspiration have been briefly described by Stoddard²⁰ as follows:

The probable and possible ore-bearing areas of the various divisions of the company's property were explored and prospected by churn-drilling; the holes were put down wherever practical on the corners of 200-foot squares.

The original churn-drill sampling was done in as careful a manner as possible. From each 5-foot interval the sludge was bailed and run through a splitter, the sample going into a tub. At this time the sample was examined for its mineral constituents, and the results were placed on the log sheet. If the examination seemed to show a preponderance of copper sulphide, the ore was classed as "sulphide"; if oxides and carbonates were mixed with the sulphides, it was shown as "mixed"; and if only the oxides and carbonates showed, it was marked "carbonate."

This classification was done with care, but it can be assumed that "sulphide" was favored slightly, as this was at the time the only ore of economic value. Toward the end of the churn-drilling period, while the samples were still classified according to their visible mineral constituents, they were also assayed for acid soluble and insoluble copper. This was a tremendous step in truly classifying the ore.

The sludge in the tub was brought to dryness over an open fire. If the sampler were careless and dried and heated the sample too much, probably some oxidation took place and the sulphide got the worst of it.

In recent drilling the sample has been taken in the same way, but has been brought to dryness on a steam sample-drier and the dried sample assayed for acid soluble and insoluble copper.

In the last few years considerable diamond drilling has been done, particularly to determine the class of ore, whether sulphide or oxide, and also to determine accurately the bottom of the ore bodies. In addition, some prospect drilling has been done with the diamond drill. In accuracy, speed, and cost diamond drilling compares most favorably with churn drilling.

²⁰ Stoddard, Alfred C., *Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.*: Inf. Circ. 6169, Bureau of Mines, 1929, 23 pp.

RAY MINES, RAY, ARIZ.

At Ray, Ariz., the first prospecting was done by sinking shafts, followed by a churn-drilling campaign using portable rigs, the holes being drilled at the corners of 200-foot squares. The samples were put through a split divider, and a careful record of each sample was kept by the sampler of the rock being drilled, the color of the sludge, the various mineral constituents, the weight of material cut for each 5-foot sample, the size of bit, and the depth and size of casing in the hole. The size of bit, taken in conjunction with the total weight of the 5-foot sample, indicated the extent to which the hole might be caving and hence the reliability of the sample.

LA COLORADA MINE, SONORA, MEXICO

At La Colorada mine, Cananea, Mexico,²¹ considerable churn-drill exploration has been done. Catron states as follows:

Prospecting and exploration at first consisted of sinking shafts or driving tunnels on mineralized outcrops or in the vicinity of the prominent iron-stained gossans which were considered indications of secondary enrichments below. Later, and continuing to the present time, a great deal of drilling has been done in the district. Prospect drilling of new areas from the surface has been done by churn drills; diamond drilling has been carried on underground, generally in exploring or developing known ore.

A considerable amount of churn drilling was done along the southwesterly extension of the Ricketts fault in an attempt to find a lateral extension of the Oversight ore. The Colorada and Sonora Hill areas were churn-drilled on coordinates in a systematic exploration campaign. Holes were drilled on the corners of 200-foot squares. At first, when interest was concentrated chiefly on possible secondary ores, holes were drilled to depths of only 300 to 500 feet. Later, upon discovering the existence of the rich primary ores, many holes were drilled to depths of 800 to 1,600 feet.

The holes did not need to be cased, except a few in soft ground, as in the Oversight area. The holes were sampled at 5-foot intervals, all the way from the surface. All the sludge was run through a splitter, saving a ¼-part sample. This sample was dried, without settling or decanting, over a fire, and the sample assayed for total copper. Composite samples of 40 to 50 foot lengths were assayed for gold and silver. A geologist inspected the sludge and noted the kind and character of rock being drilled through.

OTHER REFERENCES

Other references of interest follow:

- NOTMAN, ARTHUR. Churn-Drilling Costs, Sacramento Hill. *Trans. Am. Inst. Min. and Met. Eng.*, vol. 52, 1915, pp. 444-457.
 GRUNOW, WILLIAM R. Churn-Drill Prospecting at Morenci, Ariz. *Eng. and Min. Jour.*, vol. 101, June 3, 1916, pp. 969-974.
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 MOSES, FREDERICK G. The Sampling of Churn-Drill Prospect Holes. *Eng. and Min. Jour.*, vol. 100, Aug. 21, 1915, pp. 301-304.

²¹ Catron, William, *Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico*: Inf. Circ. 6247, Bureau of Mines, 1930, 40 pp.

COSTS

TABLE 3.—Typical churn-drilling costs

District	Mine	Material drilled	Size of holes, inches	Approximate depth of holes, feet	Cost per foot	Remarks
Tri-State	No. 1	Shale, sandstone, limestone, chert, flint.	5¾ and 6¼	250	\$1 to \$1.25	Contract price.
Do	No. 2	do	6¼	300	\$1 to \$1.50	Do.
Waco	Acme	do	6¼	300	\$1	Do.
Tri-State	No. 3	do	6¼	190	\$1.10 to \$1.50; \$0.40 to \$0.60 in shale.	Do.
Bingham, Utah	Utah Cons. Copper Co.	Porphyry	23	1,469	\$20.49	Data on 1 hole.
Chino, N. Mex.	Nevada Cons. Copper Co.	do	13 to ?	945	\$3.72 (hole 824)	Steam.
Do	do	do	13 to ?	965	\$2.75 (hole 825)	Gasoline.
Bisbee, Ariz. ¹	Sacramento Hill	do	do	Ave. 400	Initial cost..... \$0.17 Operating..... 1.43 Roads..... .37	60,000 feet of drilling.
					Total..... 1.97	
Morenci, Ariz. ²	Phelps-Dodge	do	10¾ to 4¾	Ave. 627	Operating..... \$1.422	
				Max. 1,039	Total..... ³ 3.257	
Nye County, Nev. ⁴		Clay and crystalline gylusite.	5½	770 to 906	Drilling only..... \$1.033	6 holes.
					Total..... 3.371	
Ludlow, Calif. ⁵	Pacific Mines Corp.	Porphyry	8¼ to 6	Ave. 258	\$1.043	24 holes; total, 6,456 feet.
Miami, Ariz. ⁶	Miami	do	10 to 4¾	Ave. 605	Drilling only..... \$2.75	73 holes; total, 44,500 feet.
					Total..... ⁷ 3.26	
Burro Mountain, N. Mex. ⁸	Savanna Copper Co.	do	10 to 4¾	Ave. 415	\$1.684	29 holes.
				Ave. 681	\$2.487	28 holes.
				Ave. 545	\$2.177	57 holes.
Silver Bell, Ariz. ⁹	Imperial Copper Co.	Rhyolite porphyry and granite.	7¾ to 4¼	Ave. 230	\$2.52	62 holes.

¹ Notman, Arthur, Churn Drilling Costs, Sacramento Hill: Trans. Am. Inst. Min. and Met. Eng., vol. 52, 1915, pp. 444-457.

² Grunow, William R., Churn Drill Prospecting at Morenci, Ariz.: Eng. and Min. Jour., vol. 101, June, 1916, pp. 969-974.

³ Drilling labor, \$0.585; supplies, \$0.453; tool sharpening, \$0.043; power, \$0.201; repairs, \$0.140; total operating, \$1.422. Sampling and assaying, \$0.396; general expense, \$0.230; total operating, assaying, and general expense, \$2.048. Preparatory expense, \$0.973; geology and engineering, \$0.018. First cost of equipment, \$0.218. Total cost, \$3.257.

⁴ Free, E. E., Drilling Costs in Potash Prospecting: Eng. and Min. Jour., vol. 100, July 17, 1915, pp. 108-109.

⁵ Palmer, C. H., jr., Churn-Drilling Costs: Eng. and Min. Jour., vol. 99, Jan. 2, 1915, p. 20.

⁶ Bowen, H. P., Churn-Drill Costs at Miami: Eng. and Min. Jour., vol. 99, Jan. 2, 1915, p. 21.

⁷ Drilling only, \$2.75; roads, \$0.44; water, 0.04; repairing, \$0.03; total, \$3.26.

⁸ Stauber, I. J., Churn Drilling in New Mexico: Eng. and Min. Jour., vol. 94, Sept. 14, 1912, pp. 501-503.

⁹ Gentry, Martin Butler, Details of Churn Drill Operations at Silver Bell, Ariz.: Eng. and Min. Jour., vol. 90, Oct. 29, 1910, pp. 850-851.

CORE DRILLING

Core drilling includes the methods of shot drilling, calyx drilling, and diamond drilling. A method sometimes used is a combination of shot and calyx drilling. In all these methods an annular groove is cut by the bit, leaving in the center a solid core which passes through the bit into a core barrel and is extracted from the hole. The core affords an excellent means of determining many of the physical and geological features of the formations drilled and provides as well, under favorable conditions, an accurate sample for assay. Core drilling is especially suitable for penetrating relatively firm, hard material of homogeneous texture. Irregularities in hardness, brecciated or fractured ground, open fissures, and soft altered material all tend to increase the difficulties of drilling, lower the percentage of core recovery, and vitiate the accuracy of samples.

The conditions under which these methods are applicable and the results obtained by their use will be treated for each method under separate captions.

DIAMOND DRILLING

Diamond-core drilling is widely employed at American metal mines for prospecting, exploring, developing, and sampling ore bodies.

The equipment consists essentially of a boring column composed of jointed steel rods of standard lengths (5, 10, or 20 feet), a hollow "core barrel" usually 5 or 10 feet long, a core lifter and lifter shell, an annular bit set with diamonds (carbonados), and a geared driving mechanism that rotates the column. Small machines for use underground are air or electric driven. Larger sizes for surface use may be powered with electric, steam, gasoline, or compressed-air drives. Rods are jointed, and the drilling column is raised and lowered by means of an auxiliary winding drum and a derrick or other handling rig. Water must usually be pumped to the drill, but sometimes sufficient head may be obtained in underground work by tapping one of the mine-pump columns. Feed is either hydraulic or of the differential screw type. Bits and core barrels are of various dimensions, the common ones delivering cores $\frac{3}{4}$ to $2\frac{1}{8}$ inches in diameter. In soft ground saw-tooth steel bits may be used, and solid bits are sometimes employed in passing through formations from which core is not required. The use of borts (diamond chips) and tungsten alloys where rock characteristics will permit their use has increased in recent years due to the mounting cost of good carbons. Sizes of equipment range from light outfits for underground or shallow surface work, limited to depth capacities of 350 to 400 feet, to large, expensive surface rigs that can drill 5,000 feet or more. A more detailed discussion of equipment or of the mechanics of drilling is beyond the scope of this paper. The subject is fully treated in technical handbooks and manufacturers' catalogues.

APPLICATION

Diamond drilling is an accurate and economical means of sampling large, homogeneous, rather flat-lying deposits of fairly hard or even-

textured material, such as some of the hard Lake Superior iron ores, some of the disseminated low-grade coppers, the lead deposits of the southeast Missouri district, and other extensive bedded or massive ores. Tonnage and grade estimates made from diamond-drill results at such deposits are generally reliable and accurate if judgment based on experience is used in their interpretation.

Diamond drilling is usually more expensive than churn drilling, but is not limited to holes of vertical inclination, and in favorable ground gives more complete information by recovery of solid cores. When the ore body is at a considerable depth below the surface the unit cost can be reduced by deflecting or wedging the hole at any desired point and thus cutting the ore with several holes having a common upper portion through the barren capping.

At many mines diamond drilling can not be relied upon to give accurate information regarding the grade or amounts of ore but is extremely valuable for obtaining such geological data as the position of favorable beds, thickness of strata, location of dikes or structures controlling ore deposition, location of faults, presence of blind or parallel lodes, limits or boundaries of mineralized ground, and the like.

Used underground the diamond drill is perhaps most valuable for such service as that just described. However, at some mines drill holes provide such accurate information that they are used in the same way as crosscuts for blocking out ore and are given the same weight in figuring reserves.²²

Diamond drills have their best application where the ground is relatively firm, hard, and unbroken or where long inclined holes are necessary. In soft or fractured ground the recovery of cores is usually low, and caving of the hole with consequent dilution or salting of the sample is a source of trouble and error. If the ground is not homogeneous or is vuggy or fractured diamond losses through breakage or even loss of bits may be considerable, due to jarring or uneven pressure on the bit. Deflection of the hole is partly due to such conditions and gives misleading results unless the hole is surveyed.

OPERATION

For surface work the machines range in size from small, light outfits for prospecting in rough or inaccessible regions, through medium-sized portable rigs mounted on wheels or trucks, to the larger-capacity steam units which require a set-up, including boilers and rigs or derricks that can handle relatively long sections of rods. Thus the cost of a surface-drilling campaign includes not only the direct drilling expense plus capital charges, but a variable and often considerable item for moving into location and setting up. With a truck-mounted machine in flat country this item is negligible, but when heavier steam equipment is used in a region of mountainous or rough topography the expense for roads and moving will enter very noticeably into the final cost.

Surface holes are ordinarily located either on a geometric (usually rectangular) system or in lines normal to the trend of a known

²² Catron, William, *Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico*: Inf. Circ. 6247, Bureau of Mines, 1930, p. 4.

structure, although in areas of extremely rugged topography a somewhat irregular arrangement may be unavoidable.

The purpose of the drilling (whether it is systematic sampling of a large ore body or is more of an exploratory nature), the local geologic conditions, the funds appropriated to the work, and the judgment of the man in charge will determine the layout of holes. Holes may be vertical or inclined, and a fan of holes differing in azimuth and inclination may be put down from one point. Sometimes deflecting wedges are used in diamond-drill holes to change their course or start a new hole at depth. The use of wedges in the latter instance often effects appreciable savings by taking advantage of an existing hole for part of the distance rather than putting down a new one from the surface.

In surface work the upper part of the hole is often in unconsolidated material, such as glacial drift, alluvial fill, or residual soil and clay. In such instances the hole is put down to bedrock by churn drills or jetting drills. Casing must be used from the surface to solid rock to prevent caving, spalling, dilution of sludge, and loss of water.

Underground operation must be modified somewhat because of the limitations imposed by lack of space. Small air or electric operated machines are mounted on special compact frames or are set up on columns and crossbars. In most instances stations must be cut to permit the handling of rods; for down holes these take the form of short raises, and sprags and staging are necessary.

It is frequently economical, when an extensive program of underground drilling is undertaken, to "fan" the holes drilled from each set-up—that is, to drill a number of holes at different azimuths and inclinations from each station and have fewer stations, rather than to cut the same area with a smaller number of holes from a greater number of stations. The first procedure will increase the footage drilled but will sometimes give better results from the standpoint of cost than the second, unless carried to extremes, since the greater footage drilled to reach the same objective is offset by the reduced cost for cutting stations, moving, setting up, and lost time. These observations will apply to surface drilling in some degree, as, for instance, deep drilling with heavy equipment on rough terrain.

Open cavities or short porous sections which cause loss of water and consequent vitiation of sludge samples were formerly cased or plugged with manure, bran, or tallow. More modern practice is to force quick-setting cement into the hole, keep it under air pressure long enough to prevent water from interfering with setting, and then drill through the cement when it has hardened sufficiently. Long stretches of porous or broken material may sometimes be cased, but frequently the hole must be abandoned when such ground is encountered.

Short sections of casing are often needed in soft ground to prevent caving of the hole or dilution of sludge from pieces knocked off by the rods above. Reaming must be done before the casing is put in, or else a smaller bit must be used below the casing.

DEFLECTION OF HOLES

Deflection of holes is a feature of diamond drilling which may seriously influence the results obtained, especially in holes 500 feet or more deep. Deflection may be caused by uneven texture and hardness of the rock; slips, faults, joints, or planes of bedding or schistosity; vugs, cavities, and fissures; contacts of formations of divergent character; or uneven pressure on the bit. In flat holes the sag of the boring column due to weight of the rods may cause deflection, especially if the core barrel is worn, the bit dull, and the runner careless or inexperienced. In general, holes at a flat angle to bedding, fault, or other planes tend to turn parallel with those planes; holes nearer normal to such planes tend to assume perpendicularity with them.

Deviation may be prevented or minimized by using sharp bits with small diamond clearance and new core barrels and by running at slow rotative speed with light or moderate pressure on the bit. Holes are sometimes brought back onto their courses by the use of deflecting wedges.

In homogeneous rocks deflection generally is not great enough to demand consideration, except in rather deep holes and in flat holes.²³ In some formations or series of formations the behavior of the diamond drill with regard to deflection can sometimes quite accurately be predicted on the basis of experience. White²⁴ describes the factors controlling curvature of holes on the Marquette iron range and explains the use of graphic curves in predicting the course a hole will take with different initial dips. By taking advantage of this knowledge holes can be started at the proper location and inclination to intersect the desired objective approximately.

In most formations which are not homogeneous the position of a drill hole can not be predicted, due to irregularities of faults or slips. In such cases uncertainty and possible grave errors in interpretation can only be overcome by surveying the hole.

Drill-hole surveying is a subject in itself and one that has received abundant attention in the handbooks, manufacturers' manuals, and technical press.²⁵ Suffice it here to say that hydrofluoric-acid etching in a glass tube is still the principal method of determining dip, with either the Maas-compass (compass floating in gelatin in a tube) or the photographic-compass method generally employed in non-magnetic formations to ascertain azimuth. The importance of this feature of drilling exploration has frequently been overlooked or scoffed at. It should be self-evident that the location of a sample is equally as important as its value—the one is useless without the other—and that if accurate samples are obtained with care and expense, equal pains should be given to mapping the position of the hole when deflection is appreciable.

²³ Walker, Harlan A., *Diamond Drilling in El Potosi Mine*: Eng. and Min. Jour., vol. 114, 1922, pp. 896–902.

²⁴ White, E. E., *Controlling Curvature of Diamond-Drill Holes*: Eng. and Min. Jour., vol. 90, 1910, p. 546. See also Ebbutt, F., *Diamond-Drilling Practice at the Britannia Mines*: Eng. and Min. Jour., vol. 126, 1928, pp. 446–447.

²⁵ White, E. E., *Surveying and Sampling Diamond-Drill Holes*: Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 69–90. Lundberg, Sven, *Borehole Surveying by the Kiruna Method*: Eng. and Min. Jour., vol. 117, 1924, pp. 690–693. Haddock, M. H., *Deep Borehole Surveys and Problems*: McGraw-Hill Co., New York, 1931, 296 pp.

DRILL-HOLE LOGS

Practice in logging holes varies in different districts and with different operators, but in general the following data are recorded in some form or other:

Field data should show number of hole, location, elevation of collar, direction and inclination, date started, daily record of footage drilled, core recovered, formations cut, bits used, casing placed (if any), operating delays, condition of hole, grouting (if any), operating remarks.

Geological data include number, location, and direction of hole, dates, daily log of formations, color of sludge, core and sludge recovery, core and sludge assays, type and degree of mineralization and structural features recognizable, such as faults, contacts, joints, etc. Further data may include notes on surveying of the hole.

Typical log sheets and reports used in connection with this work are illustrated in Figures 10, 11, 12, 13, 14, 15, and 16.

Hole No. 1106		Sto. Domingo, July 26, 1921	
EL POTOSI MINING CO. DRILL RUNNER'S REPORT			
		Day x Night —	
Ore encountered at 415 ft.	Terminating at 423 ft.		
Commencing at 415 ft.	To 422 ft.		
Ore encountered at — ft.	To — ft.		
Ore encountered at — ft.	To — ft.		
Remarks: Rods pulled twice. Pump repacked.		[Signature]	
		_____ Drill Runner.	

FIGURE 10.—Drill runner's daily report

Figure 10 is a drill runner's daily report as used by El Potosi Mining Co., and Figure 11 an operating report for an individual hole. Figure 13 shows typical drill logs and maps used by the same company. Figure 12 is a form used on the Mesabi range for recording analyses of samples from dia-

mond and churn drill holes. Figure 14 is a diamond-drilling record employed by the Britannia Mining & Smelting Co. (Ltd.). Figure 15 illustrates a detail diamond-drill hole record as kept by the Corrigan, McKinney Steel Co. and Figure 16 typical drill-hole sections as prepared by the same company.²⁶

SAMPLING PRACTICE

When diamond drilling is undertaken for sampling ore deposits too much attention can not be devoted to obtaining reliable samples. Contract drillers, in their zeal to make footage, sometimes minimize the importance of good core recovery when it means reduced progress and are apt to be careless and hasty in handling core and recovering sludge. The expense involved in a drilling campaign demands for its justification that every effort be made to obtain complete and accurate samples of both core and sludge. As pointed out by Longyear²⁷—

In diamond-drill work a true sample consists of all the material cut by the bit—both core and cuttings. As the recovery of this sample is the object of diamond drilling the utmost care should be taken to secure it.

²⁶ Gillies, Donald B., McNeil, E. D., and Blackwell, Frank, Standardization of Mine Maps: Presented at annual meeting, Am. Inst. Min. and Met. Eng., 1930, not published.

²⁷ Longyear, R. D., Diamond-Drill Sampling Methods: Trans. Am. Inst. Min. Eng., vol. 68, 1923, p. 423.

To obtain a complete recovery of core is, of course, desirable, but in the majority of cases it is impossible to do so. The core alone would constitute a perfect sample if recovery were perfect and the diameter uniform. In actual practice, however, the gangue and ore minerals are generally of different degrees of hardness or brittleness, so that

EL POTOSI MINING COMPANY											
DIAMOND DRILL HOLE REPORT											
DRILL BASE NO. 0		SHAFT NO. 1		LEVEL NO. 4		HOLE NO. 1105					
DATE	FEEET DRILLED PER SHIFT	Total per Day	LIME	ORE	DATE	FEEET DRILLED PER SHIFT	Total per Day	LIME	ORE	MFT. DRILLING PER HOUR	
D					30	D	2				
N					30	N	38	40	40		
D					July	D	60				
N					1	N	60	120	120		
D					2	D	50				
N					2	N	50	100	100		
D					4	D	50				
N					4	N	40	90	90		
D					5	D	45				
N					5	N	35	80	79 1/2	1/2	
D					6	D	26 1/2				
N					6	N	10 1/2	37	20	17	
D					8	D	9				
N					8	N	11 1/2	20 1/2		20 1/2	
D					9	D	32 1/2				
N					9	N	20	52 1/2	52 1/2		
D					11	D	35				
N					11	N	3 1/2	38 1/2	38 1/2		
D					12	D	10				
N					12	N	11 1/2	21 1/2	21 1/2		
D						D					
N						N					
D						D					
N						N					
Totals											

BIT REPORT			
Bit No.	Wt. Kts. Out	Wt. Kts. In	Loss Kts.
1105	10.69	9.75	0.94
TOTALS			

Total Supply Cost	\$8.08
Total Labor Cost	\$87.50 (U. S. currency)

REMARKS:

Bit No. 1105 drilled 600 feet.

FIGURE 11.—Report for an individual hole

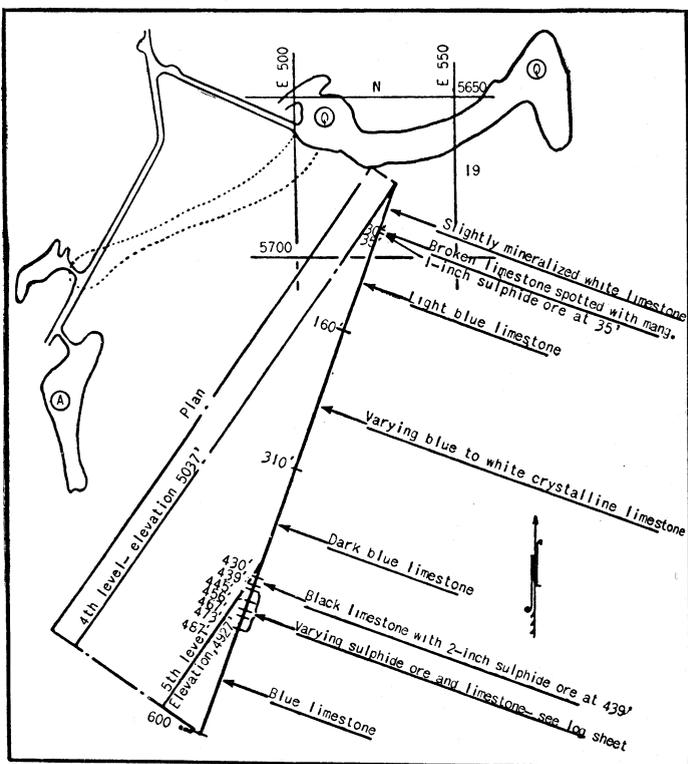
one or the other is apt to be ground out of the core and mixed with the sludge in disproportionate amounts. Core frequently breaks along planes of gouge or seams of metallic minerals, grinding up the material along the break and salting the cuttings with it. Thus the solid core recovered is very frequently not representative of the material drilled, and assays thereof fail to indicate the grade of

EL POTOSI MINING COMPANY DIAMOND DRILL HOLE LOG						
			July 12 th 1921			
SHAFT NO. 1	LEVEL NO. 4	DRILL BASE NO. Δ				
COLLAR COORD. LAT. S 671.90 M	DEPT. E 525.30 M	HOLE NO. 1105				
COLLAR ELEVATION 5037.25 FT	DIRECTION. S 35° 07' W	BLOCK NO. 19				
DIP 15° NAT. SIN. 0.259	NAT. COS. 0.966					
FEET	Class of Sample	Silver %	Lead %	ZINC %	REMARKS	
0-50	Lime				Slightly mineralized, white	
50-53	Lime				Broken, spotted with manganese and 1-inch streak sulphide ore at 53'. No sample	
53-160	Lime				Light blue	
160-310	Lime				Varying blue to white crystalline	
310-430	Lime				Dark blue	
430-445	Lime	7.0	5.0	1.0 (Core)	Black, with 2-inch sulphide ore	
445-456	Lime	11.0	7.2	1.8 (Core)	Lime and ore mixed	
	Ore	16.0	13.1	9.7 (Sludge)		
456-467	Ore	16.0	15.8	6.9 (Core)	Sulphide	
		24.0	19.9	9.4 (Sludge)		
467-477	Ore	10.0	7.1	9.8 (Core)	Mixed sulphide ore and limestone	
	Lime	16.0	6.3	4.7 (Sludge)		
477-487	Ore	9.0	5.0	4.2 (Sludge)	Sulphide mixed with blue limestone	
487-600	Lime				Blue	

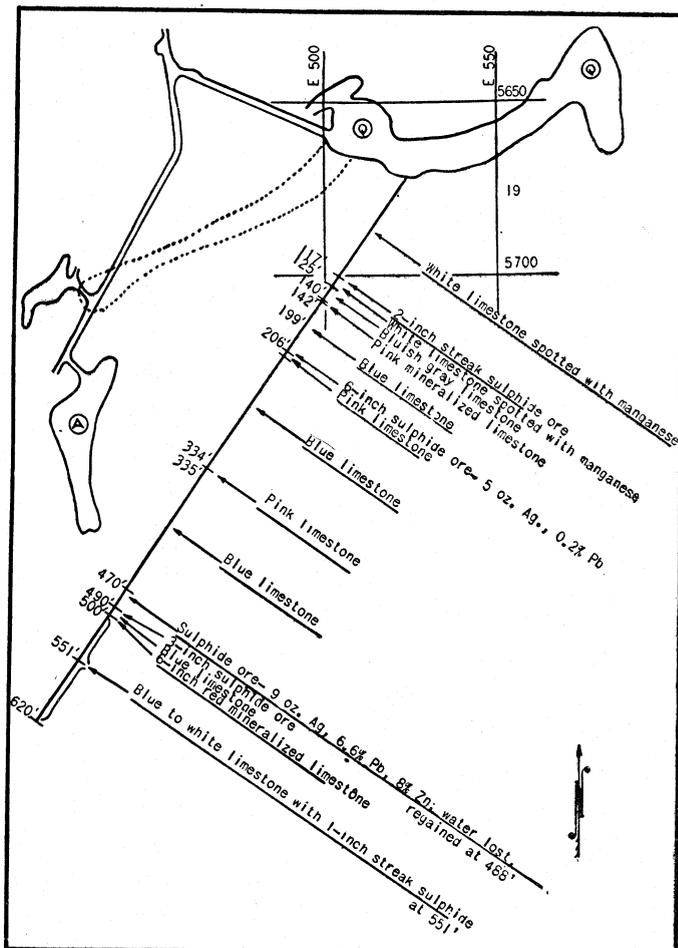
Log of drill hole 1105

EL POTOSI MINING COMPANY DIAMOND DRILL HOLE LOG						
			June 30 th 1921			
SHAFT NO. 1	LEVEL NO. 4	DRILL BASE NO. Δ				
COLLAR COORD. LAT. S 671.90 M	DEPT. E 525.30 M	HOLE NO. 1104				
COLLAR ELEVATION 5038.0 FT	DIRECTION. S 35° 07' W	BLOCK NO. 19				
DIP 15° NAT. SIN. 0.259	NAT. COS. 0.966					
FEET	Class of Sample	Silver %	Lead %	ZINC %	REMARKS	
0-125	Lime				white, spotted with manganese and 2-inch streak sulphide ore at 117'. No sample	
125-140	Lime				Bluish gray	
140-142	Lime	1.0	0.1	0.4	Pink, mineralized	
142-199	Lime				Blue	
199-206	Lime	5.0	.2	(Core sample)	Pink, with 6-inch sulphide ore at 199'	
206-470	Lime				Blue, with 1 1/2' pink at 334'	
470-490	Ore	9.0	6.6	8.1	Sulphide, impregnated with iron. Water lost. Regained at 488'	
490	Ore				(No sample)	
490-500	Lime				Blue, with 6-inch red mineralized limestone at 500'	
500-620	Lime				Blue to white, with 1-inch sulphide ore at 551'	

Log of drill hole 1104



Plan and section, drill hole 1105



Plan and section, drill hole 1104

FIGURE 13.—Typical drill logs and corresponding maps. (Eng. and Min. Jour.—Press, vol. 114, no. 21, p. 900)

paper this subject has been ably discussed by Matson and Wallis,²⁹ who point out that, for Rhodesian copper ores, sludge is not so important when core recovery is high (90 to 100 per cent), but when core recovery is low sludge recovery must be nearly perfect to be of value, since it is now partly composed of core. They also conclude that unless the return of water from the hole is practically complete sludge assays are misleading, since loss of water indicates that fine solid material is being sucked into the cracks and crevices through which water is leaving the hole. In the work they describe holes were cemented or cased if loss of water exceeded 5 per cent.

Loss of water not only implies the passage of part of the cuttings from the hole but also reduces the velocity of flow in the hole and results in the heavier or coarser cuttings being left on the bottom. In pumping out sludge, pressure should be high enough to lift all particles and thus prevent error due to classification, and flow should be continued until no more particles appear. If the ground is tight enough to stand considerable pressure without loss of water "reverse pumping" (pumping down the casing with upward discharge through the rods) will deliver the cuttings with less water, since the smaller cross section within the rods produces increased velocity of flow, with consequent greater lifting power.

Care should be taken to keep the rods as free from oil as possible, since sulphides are apt to be floated off from the settling tubs by oil.³⁰

Sludge is run from the casing head through pipes or launders into receptacles of various kinds. The easiest and least accurate method simply involves running the sludge into a tub and allowing the excess water to overflow. Results are poor because slimes are carried off in the overflow. When sludge boxes fitted with baffles are used results are better if the water is not siphoned off until the cuttings have settled. A more elaborate but very satisfactory method of collecting sludge is to employ a series of barrels³¹ or sludge boxes³² with a swinging launder, so that as one receptacle is filled the discharge from the hole is diverted to the next. About the time the last receiver is nearly full the first may be safely decanted and filled again, and so on. After sufficient time for settling the clear water is decanted by siphoning or discharging through bungs, and the sludge is collected in a tub and dried slowly over a fire. Riffled splitters of the Jones type are sometimes employed to reduce the volume of drill-hole sludge collected for examination. If care is taken to maintain a uniform split so that total sludge recovery can be calculated correctly, this practice is satisfactory. Various modifications may be advantageously employed under different conditions. The important thing is to clean the hole and rods completely to prevent loss of slimes, and to obtain as near 100 per cent recovery of cuttings as possible.

Drying should be done carefully, if a fire is used, to prevent oxidation of sulphides. At Cananea (see p. 47) and elsewhere com-

²⁹ Matson, H. T., and Wallis, G. A., *Drill Sampling and Interpretation of Sampling Results in the Copper Fields of Northern Rhodesia*: Trans. Am. Inst. Min. and Met. Eng., Tech. Pub. 373, 1930, 17 pp.

³⁰ Joralemon, I. B., *Sampling and Estimating Disseminated Copper Deposits*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 610-611.

³¹ E. J. Longyear Co., Catalogue 8, pp. 10-11.

³² Matson, H. T., and Wallis, G. A., *work cited*.

DATE	FEET DRILLED	TOTAL DEPTH	SAMPLE			ANALYSIS			COMBINED ANALYSIS			REMARKS	SECTION - DRILL HOLE NO. 15					
			FROM	TO	FEET	IRON-DRY	PHOS	IRON-DRY	PHOS	IRON-DRY	PHOS		GEOLOGY	FOOTAGE ALONG	COMPLETE ANALYSES			
OCTOBER																		
Day 21																		
Day 23	13	13																
Day 24	19	32		0	11	11												
Day 25	8	40																
NITE 25	11	51																
Day 26	10	61																
NITE 26	12	73																
Day 27	10	83																
NITE 27	8	91																
Day 28	10	101																
Day 30	14	115																
NITE 30	14	129																
Day 31	14	143																
NITE 31	12	155																
NOVEMBER																		
Day 1	14	169																
NITE 1	9	178																
Day 2	7	185																
NITE 2	10	195																
Day 3	17	212																

CONTINUED ON SHEET 2

LOCATION 260 FT. N AND 40 FT. E. OF SW. CORNER OF SECTION
 ELEV. COLLAR OF HOLE 1441 FT. ABOVE SEA LEVEL
 DATUM SEA LEVEL
 DIRECTION AT START BEARING N. 15° W.
 DIP 65°
 STARTED OCTOBER 23, 1922
 COMPLETED DECEMBER 2, 1922
 ULTIMATE DEPTH 420 FT.
 PROPOSED DEPTH 500 FT.
 DRILLED BY JOHN DRILL & SONS
 MATERIAL CLASSIFIED BY P. F. JONES, GEOLOGIST

THIS IS A JOINT HOLE WITH THE
 I. U. MINING COMPANY
 CHURN DRILLING THE FIRST 35 FEET
 AND THEN DIAMOND DRILL
 THE CASING IS LEFT DOWN TO 35 FEET.

THE IDEAL MINING CO.
 RECORD OF
 DIAMOND
 DRILL HOLE NO. 15
 MODEL MINE NO. 6
 SEC. 16, T. 47 N., R. 46 W., MICHIGAN
 GOGEBIC RANGE
 ENGINEERING DEPT. DATE
 PENNSCOKE, MICHIGAN

FIGURE 15.—Drill-hole record, Gogebic iron range

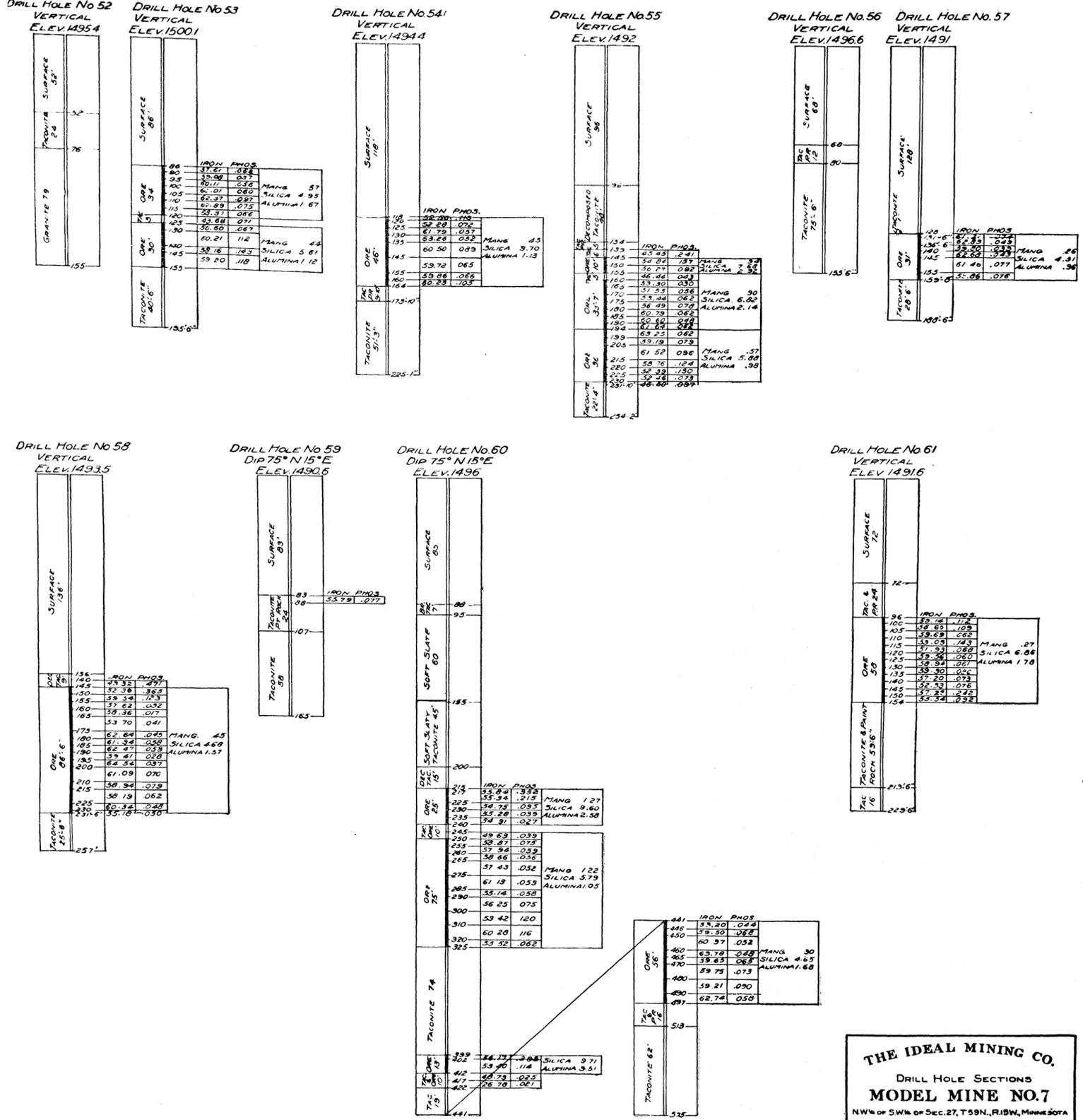


FIGURE 16.—Drill-hole sections, Model Mine, Minnesota

THE IDEAL MINING CO.
 DRILL HOLE SECTIONS
 MODEL MINE NO. 7
 NW 1/4 OF SW 1/4 OF SEC. 27, T. 59 N., R. 18 W., MINNESOTA
 VERTICAL SCALE 1"=50'
 ELEVATIONS ARE ABOVE SEA LEVEL
 ENGINEERING DEPT. DATE
 PENNSCOKE, MICHIGAN

pressed-air filters have been used successfully in place of decanting for preliminary unwatering.

CALCULATIONS IN COMBINING SAMPLES

Core and cuttings should be assayed separately and the results combined according to the proportionate recovery of each class of sample. In the exceptional case of complete recovery of both an assay of the combined core and sludge would be correct, but since in practice the two are recovered in different proportions, consideration must be given this fact.

Practice varies with respect to handling core and sludge; and in the past, particularly, careless methods were often used. At some mines core only is assayed; at others sludge only is considered; at many the two are combined for assay, but not always have correct weights been given to core and cutting assays before they are combined, so as to represent the value of the portion of hole from which they were taken. This last method is the only accurate one, except when all material is recovered for treatment as one sample, or when the metallic and nonmetallic minerals are so similar in habit and characteristics that sludge and core are representative even when not wholly recovered. Such a condition is rare, but doubtless it is in some instances rather closely approximated—sufficiently so, at least, that practical results obtained with some of the incorrect sampling methods in use are close enough to suit the management or to justify the financial savings they effect over the more accurate, but more costly, correct methods. Assaying sludge only is probably the least accurate of all methods, due to concentration and classification of the particles according to size and weight, loss of slimes, salting or dilution from core and caving walls, and sludge loss in cracks and fissures.

The correct method of weighting assays involves consideration of the volumes or weights of each class of material recovered. Matson and Wallis³³ give several formulas and examples of good and bad practice, the one they prefer involving factors for volume and specific gravity. Thus they allow, in estimating tonnage, for variation in the gravity of the material, which is theoretically correct and doubtless justified for the purposes of their work. For average work these formulas are too perfect, since the physical errors of sampling which are bound to occur from such sources as minor variations in the outside diameter of holes, undetected sludge loss (which can occur if sludge in the bottom of the hole is lost in crevices, or not raised, while a similar volume is caved in from above) are greater than the refinements used in the formulas.

Table 4 gives a series of factors used by the E. J. Longyear Co. for evaluating core and sludge assays; these are based on the relative volumes of core and sludge for a given linear recovery of core, and they assume a constant diameter of core and hole and complete recovery of sludge. Results in practice will be vitiated to a greater or less degree by changes in diameter of core due to wear in the barrel; variations in diameter of hole due to caving, alternating hard and soft layers, etc.; loss of sludge and concentration of heavy

³³ Matson, H. T., and Wallis, G. A., work cited.

particles in porous or fissured ground,³⁴ and "lag" of sludge (this can always be eliminated by care in cleaning holes). It will be seen that cuttings range from a minimum weight, depending on relative cross sections of hole and core, to a maximum of 100 per cent at zero core recovery, and that core weighting ranges from zero at no recovery to a maximum dependent on relative cross sections.

TABLE 4.—Percentages of volumes of core and cuttings in a 5-foot diamond-drill sample for each inch of core recovered, to be used as multipliers in combining analyses of core and cuttings

[After E. J. Longyear]

Inches of core	EX		AX		BX		NX	
	Core	Cuttings	Core	Cuttings	Core	Cuttings	Core	Cuttings
1.....	0.6	99.4	0.6	99.4	0.8	99.2	0.9	99.1
2.....	1.2	98.8	1.2	98.8	1.6	98.4	1.7	98.3
3.....	1.8	98.2	1.8	98.2	2.4	97.6	2.6	97.4
4.....	2.4	97.6	2.4	97.6	3.2	96.8	3.4	96.6
5.....	3.0	97.0	3.0	97.0	4.0	96.0	4.3	95.7
6.....	3.5	96.5	3.6	96.4	4.8	95.2	5.1	94.9
7.....	4.1	95.9	4.2	95.8	5.6	94.4	6.0	94.0
8.....	4.7	95.3	4.8	95.2	6.4	93.6	6.8	93.2
9.....	5.3	94.7	5.4	95.6	7.2	92.8	7.7	92.3
10.....	5.9	94.1	6.0	94.0	8.0	92.0	8.5	91.5
11.....	6.5	93.5	6.6	93.4	8.8	91.2	9.4	90.6
12.....	7.1	92.9	7.2	92.8	9.6	90.4	10.2	89.8
13.....	7.7	92.3	7.8	92.2	10.4	89.6	11.1	88.9
14.....	8.3	91.7	8.4	91.6	11.2	88.8	11.9	88.1
15.....	8.9	91.1	9.0	91.0	12.0	88.0	12.8	87.2
16.....	9.5	90.5	9.6	90.4	12.8	87.2	13.7	86.3
17.....	10.1	89.9	10.2	89.8	13.6	86.4	14.5	85.5
18.....	10.6	89.4	10.8	89.2	14.4	85.6	15.4	84.6
19.....	11.2	88.8	11.4	88.6	15.2	84.8	16.2	83.8
20.....	11.8	88.2	12.0	88.0	16.0	84.0	17.1	82.9
21.....	12.4	87.6	12.6	87.4	16.8	83.2	17.9	82.1
22.....	13.0	87.0	13.2	86.8	17.6	82.4	18.8	81.2
23.....	13.6	86.4	13.8	86.2	18.4	81.6	19.6	80.4
24.....	14.2	85.8	14.4	85.6	19.2	80.8	20.5	79.5
25.....	14.8	85.2	15.0	85.0	20.0	80.0	21.3	78.7
26.....	15.4	84.6	15.6	84.4	20.8	79.2	22.2	77.8
27.....	16.0	84.0	16.2	83.8	21.6	78.4	23.1	76.9
28.....	16.6	83.4	16.8	83.2	22.4	77.6	23.9	76.1
29.....	17.1	82.9	17.4	82.6	23.2	76.8	24.8	75.2
30.....	17.7	82.3	18.0	82.0	24.0	76.0	25.6	74.4
31.....	18.3	81.7	18.6	81.4	24.8	75.2	26.5	73.5
32.....	18.9	81.1	19.2	80.8	25.6	74.4	27.3	72.7
33.....	19.5	80.5	19.8	80.2	26.4	73.6	28.2	71.8
34.....	20.1	79.9	20.4	79.6	27.2	72.8	29.0	71.0
35.....	20.7	79.3	21.0	79.0	28.0	72.0	29.9	70.1
36.....	21.3	78.7	21.6	78.4	28.8	71.2	30.7	69.3
37.....	21.9	78.1	22.2	77.8	29.6	70.4	31.6	68.4
38.....	22.5	77.5	22.8	77.2	30.4	69.6	32.4	67.6
39.....	23.1	76.9	23.4	76.6	31.2	68.8	33.3	66.7
40.....	23.7	76.3	24.0	76.0	32.0	68.0	34.2	65.8
41.....	24.2	75.8	24.6	75.4	32.8	67.2	35.0	65.0
42.....	24.8	75.2	25.2	74.8	33.6	66.4	35.9	64.1
43.....	25.4	74.6	25.8	74.2	34.4	65.6	36.7	63.3
44.....	26.0	74.0	26.4	73.6	35.2	64.8	37.6	62.4
45.....	26.6	73.4	27.0	73.0	36.0	64.0	38.4	61.6
46.....	27.2	72.8	27.6	72.4	36.8	63.2	39.3	60.7
47.....	27.8	72.2	28.2	71.8	37.6	62.4	40.1	59.9
48.....	28.4	71.6	28.8	71.2	38.4	61.6	41.0	59.0
49.....	29.0	71.0	29.4	70.6	39.3	60.7	41.8	58.2
50.....	29.6	70.4	30.0	70.0	40.1	59.9	42.7	57.3
51.....	30.2	69.8	30.6	69.4	40.9	59.1	43.5	56.5
52.....	30.7	69.3	31.2	68.8	41.7	58.3	44.4	55.6
53.....	31.3	68.7	31.8	68.2	42.5	57.5	45.3	54.7
54.....	31.9	68.1	32.4	67.6	43.3	56.7	46.1	53.9
55.....	32.5	67.5	33.0	67.0	44.1	55.9	47.0	53.0
56.....	33.1	66.9	33.6	66.4	44.9	55.1	47.8	52.2
57.....	33.7	66.3	34.2	65.8	45.7	54.3	48.7	51.3
58.....	34.3	65.7	34.8	65.2	46.5	53.5	49.5	50.5
59.....	34.9	65.1	35.4	64.6	47.3	52.7	50.4	49.6
60.....	35.5	64.5	36.0	64.0	48.1	51.9	51.2	48.8

³⁴ Mead, W. J., Assaying Diamond-Drill Samples: Eng. and Min. Jour., vol. 91, 1911, pp. 913-914.

BIT SIZES, INCHES

[Standards of Diamond Core-Drill Manufacturers Association]

	Outside diameter	Inside diameter	Approximate diameter of hole	Approximate diameter of core
EX.....	1 $\frac{7}{8}$	$\frac{7}{8}$	1 $\frac{1}{2}$	$\frac{7}{8}$
AX.....	1 $\frac{7}{8}$ ³²	1 $\frac{1}{8}$	1 $\frac{7}{8}$	1 $\frac{1}{8}$
BX.....	2 $\frac{1}{16}$	1 $\frac{3}{8}$	2 $\frac{3}{8}$	1 $\frac{3}{8}$
NX.....	2 $\frac{1}{16}$	2 $\frac{1}{8}$	3	2 $\frac{1}{8}$

FILING SAMPLES

To preserve a complete record of the information gained by diamond drilling holes should be fully logged, as already noted; in addition, samples of core and cuttings should be filed in core boxes, drawers, or other suitable containers. At some mines a few samples of core representative of the various formations drilled are kept, while at others all core available is filed. When core is mineralized it should usually be split in a core splitter, one-half going to assay and the other to permanent file. Some companies, however, prefer to assay the entire core, relying on the notes made at the time of drilling for future reference and study. Needless to say, core should always be inspected carefully by a competent man at the time of drilling, but especially is this true when all core is assayed. Instead of splitting some companies prefer to retain a few small pieces of core most representative of the ore, assaying the balance.

EXAMPLES OF PRACTICE

Examples of practice at North American mines are summarized below.

IRON MINES

MINEVILLE DISTRICT,³⁵ NEW YORK

Over 100,000 feet of diamond drilling has been done for Witherbee, Sherman & Co., where the ore is magnetite occurring as rather uniform thick lenses in gneissoid rocks. In this material, taking a 1-inch core, average drilling speed for 31,725 feet was 10 feet per 8-hour shift at a contract cost of \$3.92. Samples are accurate.

MINE NO. 1, MARQUETTE RANGE,³⁶ MICH.

This deposit is hard specularite, folded and faulted. Formerly much ore was found by diamond drilling from the surface. At present horizontal holes are drilled underground, since the existence of extensive workings makes this a more economical mode of attack. One machine is constantly used to drill many relatively short holes,

³⁵ Cummings, A. M., Method and Cost of Mining Magnetite in the Mineville District, New York: Inf. Circ. 6092, Bureau of Mines, 1928, 12 pp.

³⁶ Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Inf. Circ. 6138, Bureau of Mines, 1929, 14 pp.

averaging somewhat over 2,000 feet total footage per year. Drilling supplements drifting and raising in exploration.

MINE NO. 2, MARQUETTE RANGE, MICH.³⁷

Soft hematite occurs in irregular masses, lenses, or chimneys in troughs formed by folds or by intersecting dikes and slates. Diamond drilling is used for surface and underground exploration and is extremely useful as an adjunct to drifting; frequent flat holes perpendicular to long drifts are put through to the hanging wall.

MINE NO. 1, MENOMINEE RANGE, MICH.³⁸

Ore bodies are irregular and steeply dipping; the grade of ore is variable, with commercial boundaries. Surface diamond drilling follows magnetic surveys, and results are used in connection with estimating reserves. Underground drilling is done supplementary to ordinary development for exploratory purposes.

EUREKA-ASTEROID MINE, GOGEBIC RANGE, MICH.³⁹

Ore is soft hematite occurring in troughs formed by intersecting dikes or slate-dike intersections. Ore bodies are irregular or flat blankets on the footwall. Diamond drilling is used for surface exploration, together with churn drilling, but is chiefly important underground, where it is employed to locate dikes and secure other geological information.

MONTREAL MINE, MONTREAL, WIS.⁴⁰

Clean-cut iron ore occurs in chimney ore bodies or in troughs formed by intersections of dikes with slate or quartzite members. Diamond drilling is done at depth to locate dikes, which are an important control of ore. Drilling is expensive, since the rich ore caves readily and lean ore is abrasive and full of vugs. One hole 1,000 feet deep cost the contractor \$12 per foot. Total drilling amounts to over 20,000 feet.

MINE NO. 5, MARQUETTE RANGE, MICH.⁴¹

Original exploration was by diamond drilling, but the ore body is so well known that little prospecting is now required.

MINE NO. 4, MARQUETTE RANGE, MICH.⁴²

Original exploration was by diamond drilling; some underground drilling has since been done to delimit the ore bodies.

³⁷ Eaton, Lucien, Mining Soft Hematite, Mine No. 2 of the Marquette Range, Michigan: Inf. Circ. 6179, Bureau of Mines, 1929, 15 pp.

³⁸ Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Michigan: Inf. Circ. 6180, Bureau of Mines, 1929, 11 pp.

³⁹ Schaus, O. M., Method and Cost of Mining Hematite at the Eureka-Asteroid Mine of the Gogebic Range, Gogebic County, Mich.: Inf. Circ. 6348, Bureau of Mines, 1930, 13 pp.

⁴⁰ Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Inf. Circ. 6369, Bureau of Mines, 1930, 29 pp.

⁴¹ Graff, W. W., Mining Practices, Methods, and Costs at Mine No. 5 of the Marquette Range, Mich.: Inf. Circ. 6380, Bureau of Mines, 1930, 10 pp.

⁴² Graff, W. W., Mining Practices, Methods, and Costs at Mine No. 4 of the Marquette Range, Mich.: Inf. Circ. 6390, Bureau of Mines, 1930, 8 pp.

HANOVER MINE, FIERRO, N. MEX.⁴³

Contact-metamorphic magnetite deposits chiefly in altered limestone. Very irregular lenticular ore bodies average 40 feet in thickness. Diamond drilling for exploring from the surface is done on company account. Fans of holes from one set-up are more reliable than single holes. Holes are 200 to 500 feet deep.

COPPER MINES

MORENCI BRANCH, MORENCI, ARIZ.⁴⁴

Ore is chiefly chalcocite with pyrite in altered porphyry. The ore bodies lie in fracture zones, with "commercial" walls. Total copper is about 1.90 per cent. Diamond drills are used to explore and develop the Clay ore body. Practice is described by Mosier and Sherman as follows:⁴⁵

In diamond drilling 50 per cent of the core is recovered which does not truly represent the ground drilled. This has made it necessary to save all the core and a known fraction of the sludge from every sampling interval. These are combined according to weights and grades to obtain the average grade of each sample. The sludge is split, to deliver one-eighth of the water and rock from the hole, with a splitter especially constructed for the work. The resultant sludge is allowed to settle in barrels until the water is clear. The water is then decanted off and the sludge evaporated to dryness in an electric drying oven. The dried sample is ground, mixed, and assayed. Tests are made periodically of the water decanted from the sludge.

To obtain the final assay for any sampling interval the following procedure is used: First, the core is weighed and put through the crusher and rolls in the sample mill. The resultant pulp is carefully mixed and split to a 3-ounce sample which is ground to -200 mesh and sent to the assay office for electrolytic assaying for total and acid soluble copper. The reject is placed in an air-tight can for possible future reference. Second, the sludge is dried in electric dryers, weighed, and carefully mixed and split to give another 3-ounce sample for pulverizing and assaying similarly to the core sample. The remaining pulp is also filed in a air-tight can. Third, a known percentage of the water decanted from the sludge is evaporated to dryness, weighed, and assayed to check the possible loss of sludge held in suspension and carried off by the water rejected. Fourth, the average copper content of the sample is determined by combining the assays of the core and sludge in proportion to their weights. If the weight of the sample does not correspond with the calculated weight of the section of the hole drilled the results may not be given full weight. Adjustments are made in using these results as well as the results of individual samples that are unusually high in copper.

RAY MINES, RAY, ARIZ.⁴⁶

Ore is chiefly chalcocite disseminated with pyrite in schist, either as small veinlets or scattered specks. Ore bodies are irregular and large. Diamond drilling is used for some underground exploration. Recovery of core is very low, averaging about 5 per cent. Both sludge and core are saved, but only the former is considered in final assays. Results are used with churn-drill and other development data in calculating grade and tonnage.

⁴³ Kniffin, Lloyd M., *Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.*: Inf. Circ. 6361, Bureau of Mines, 1930, 20 pp.

⁴⁴ Mosier, McHenry, and Sherman, Gerald, *Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.*: Inf. Circ. 6107, Bureau of Mines, 1929, 34 pp.

⁴⁵ Mosier, McHenry, and Sherman, Gerald, work cited, pp. 5-6.

⁴⁶ Thomas, Robert W., *Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.*: Inf. Circ. 6167, Bureau of Mines, 1929, 27 pp.

INSPIRATION MINE, INSPIRATION, ARIZ.⁴⁷

Ore is chalcocite and oxidized copper minerals in fractures and crevices or disseminated as specks throughout the schist and granite. Considerable diamond drilling has been done in recent years, chiefly to determine whether ore is oxide or sulphide, and also to locate the bottom of ore bodies. Some prospecting has also been done. Diamond drilling compares very well with churn drilling from the standpoints of speed, accuracy, and cost.

PILARES MINE, SONORA, MEXICO.⁴⁸

Diamond drilling is very successful in finding blind ore shoots. Horizontal and upper holes are drilled from stopes before abandoning them.

UNITED VERDE MINE, JEROME, ARIZ.⁴⁹

A light, compact diamond-drill rig has been designed for work in stopes; it weighs 500 pounds and has a capacity of 250 feet. Five of these machines, taking cores $\frac{7}{8}$ to 2 inches in diameter, drill 25,000 to 30,000 feet per year and are useful in finding hanging pendants which do not reach the level, etc. In addition, 5,000 feet of longer, flat holes are drilled per year for geological information.

UTAH COPPER MINE, BINGHAM CANYON, UTAH⁵⁰

Diamond drills have been used in exploratory work but are unsatisfactory in porphyry due to poor recovery of core and caving. Core and sludge assays do not check with each other.

CAMPBELL MINE, WARREN, ARIZ.⁵¹

Some prospecting by underground diamond drilling is done.

LA COLORADA MINE, CANANEA, MEXICO⁵²

Diamond drilling is applied at La Colorada for a variety of purposes, all drilling being done on a contract basis. Catron gives the following information:

Diamond drilling has not been used in the past for primary surface prospecting. At the Elisa mine several holes were drilled to determine the geologic structure of an unprospected area. At the Capote, vertical and inclined down holes were drilled from the bottom level to test the extent of the primary ore. In the Colorada mine a body of low-grade ore was developed by horizontal holes laid out on regular coordinates on two levels.

The bit used in diamond drilling gives a 1½-inch hole and a $\frac{1}{8}$ -inch core. No casing has been used in the diamond-drill holes. Occasionally a hole is

⁴⁷ Stoddard, Alfred C., *Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.*; Inf. Circ. 6169, Bureau of Mines, 1929, 23 pp.

⁴⁸ Leland, Everard, *Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico*; Inf. Circ. 6307, Bureau of Mines, 1930, 34 pp.

⁴⁹ Quayle, T. W., *Mining Methods and Practices at the United Verde Copper Mine, Jerome, Ariz.*; Inf. Circ. 6440, Bureau of Mines, 1931, 31 pp.

⁵⁰ Soderberg, A., *Mining Method and Costs at the Utah Copper Co., Bingham Canyon, Utah*; Inf. Circ. 6234, Bureau of Mines, 1930, 23 pp.

⁵¹ Lavender, H. M., *Mining Methods at the Campbell Mine of the Calumet & Arizona Mining Co., Warren, Ariz.*; Inf. Circ. 6289, Bureau of Mines, 1930, 18 pp.

⁵² Catron, William, *Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico*; Inf. Circ. 6247, Bureau of Mines, 1930, 41 pp.

grouted to prevent loss of water. The sludge is run through a riffle-type sample splitter which takes a one-eighth split. The sample is put in a compressed-air sample drier or filter, which removes most of the water. This filter was adopted after trying tubs and settling launders, which, it was thought, caused errors by losses during decanting or by concentration of the copper minerals in the baffles of the launders. The filter has the advantage of being sturdy and compact, and is easily and quickly operated. Since it may not hold all the sludge and water of the split sample from a given length of hole, it may be necessary to blow out the water two or three times to get a complete sample. Heavy canvas is used for the filtering element; the water comes through practically clear. The dried samples are weighed and assayed for total copper by the permanganate method.

The core is not removed at regular intervals, but is taken out whenever the core barrel is full or the rods are pulled for any other reason. Each section of core removed is handled as a unit. The percentage of the core recovered is noted each time any is removed from the hole. The core is carefully inspected every day by a geologist, and notes are taken for permanent record. A small piece of core is taken for a permanent sample and the remainder is sent to the assay office, where it is ground, weighed, and assayed for total copper by the permanganate method. If gold and silver assays are desired, they may be run on composite samples of the entire hole or certain portions of it, or on individual cores. Chief reliance for knowledge of the value of the ground drilled is placed upon the core assays; the sludge serves as a check. If a combined value is desired, it may be calculated either according to relative weights of core and sludge or according to percentage recoveries and volumes.

Core recoveries vary greatly in the district. In the Colorado mine recoveries have ranged from 40 to 95 per cent, averaging 75 per cent; in the Elisa mine, in hard limestone, recoveries averaged more than that figure, while in the soft Capote ground they were generally under 50 per cent.

ENGELS MINE, PLUMAS COUNTY, CALIF.⁵³

Considerable underground drilling is done on contract, at \$2.45 for depths up to 400 feet and \$0.15 per foot additional for each 100 feet of depth beyond 400 feet. If inclination is more than 15° from horizontal the base rate is \$2.70. Up to April, 1930, 75,378 feet had been drilled.

BURRA-BURRA MINE, DUCKTOWN, TENN.⁵⁴

Holes are used to explore and in part to block out ore. Diamond drills are used for depths greater than 150 feet (hammer drills for less).

MATAHAMBRE MINE, PINAR DEL RIO, CUBA⁵⁵

Diamond drills are used both on surface and underground to explore.

MARY MINE, ISABELLA, TENN.⁵⁶

Due to irregularities of deposit drifting is now done only in ore. Other ground is diamond drilled. Walls are drilled 300 feet deep at regular intervals, and cores and assays are recorded.

⁵³ Nelson, W. I., *Mining Methods and Costs at the Engels Mine, Plumas County, Calif.*: Inf. Circ. 6260, Bureau of Mines, 1930, 22 pp.

⁵⁴ McNaughton, C. H., *Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.*: Inf. Circ. 6149, Bureau of Mines, 1929, 17 pp.

⁵⁵ Richert, George L., *Mining Methods at Minas de Matahambre, Matahambre, Pinar del Rio, Cuba*: Inf. Circ. 6145, Bureau of Mines, 1929, 18 pp.

⁵⁶ Kegler, V. L., *Mining Methods of the Ducktown Chemical & Iron Co., Mary Mine, Isabella, Tenn.*: Inf. Circ. 6397, Bureau of Mines, 1931, 9 pp.

BRITANNIA MINES, BRITANNIA, BRITISH COLUMBIA⁵⁷

About 80,000 feet of diamond drilling was done between 1918 and 1928, by contract until 1925. An air-driven UG-type machine having a depth capacity of 750 feet was employed. This outfit used standard E core barrels and rods and ES bits set with 6 to 25 stones, according to the ground. The practice of setting a greater number of smaller stones increased the bit setter's labor, but resulted in over-all economy. Ground averaged 5.5 in hardness but ranged up to 7 in cherty formation. Advance was 10 to 30 feet per shift, using two to four bits.

The sample interval was 10 feet, all sludge was caught, and core and sludge were assayed separately. Core in mineralized ground was split, half being filed. Drill runners made daily reports on operations, and a geologist examined all core, split it when necessary, and made out a complete log of the hole. Holes deeper than 100 feet were surveyed by the geological department, using a Maas compass with a thermos gelatin container.

In the shear zone the natural deflection of holes was such that they tended to become normal to the dip and strike of the schistosity. This characteristic was sufficiently uniform so that rather accurate results in reaching objectives could be obtained by making allowance for deviation when the hole was started. As much as 25° was sometimes added to the initial inclination of a hole for this reason. In unsheared or massive ground deflection could not be predicted, due to the erratic effects of slips or faults. In estimating reserves it was important to know the true position of the holes, as drift or deflection was often considerable. Grave mistakes would have resulted if all holes had been assumed to be straight.

Costs in 1927 averaged \$2.56 direct cost and \$2.96 total cost per foot; 87.7 per cent of the drilling in that year was in hard, siliceous ground.

LEAD AND ZINC MINES OF CENTRAL STATES

MINE NO. 8, SOUTHEAST MISSOURI DISTRICT⁵⁸

Eighty diamond-drill holes totaled 48,000 feet in depth. A core inspector kept detailed field records, took care of core and samples, etc. All cores carrying lead were assayed and compared with sludge assays, which were used only as a check. Core assays were used for estimating ore. Drilling was done on company account, the cost ranging from \$0.95 to \$1.40 per foot.

MINE IN SOUTHEAST MISSOURI DISTRICT⁵⁹

Exploration is chiefly by surface and underground diamond drilling. An experienced man inspects cores, estimates the lead content, and notes the character of the ore. Cores are filed but not assayed. Sludge is assayed, the results being used with other information.

⁵⁷ Ebbutt, Frank, *Diamond-Drilling Practice at the Britannia Mines: Eng. and Min. Jour.*, vol. 126, 1928, pp. 446-447.

⁵⁸ Poston, Roy H., *Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Inf. Circ. 6160, Bureau of Mines, 1929, 22 pp.*

⁵⁹ Jackson, Charles F., *Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Inf. Circ. 6170, Bureau of Mines, 1929, 21 pp.*

Surface drilling is now chiefly for outlining outlying ore. The diamond drill underground is used for test holes deeper than 22 feet. One company machine averages 40 to 50 feet per shift.

AMERICAN ZINC CO. MINE, MASCOT, TENN.⁶⁰

Ore occurs as seams and veinlets of blende in dolomitic limestone. Surface exploration is by churn and diamond drilling, holes being irregularly spaced. Diamond drilling underground keeps a machine constantly busy. Specimen cores are filed except when in ore, when entire core is assayed.

LEAD-SILVER MINES

PARK-UTAH MINE, PARK CITY, UTAH⁶¹

Core recovery in three 300-foot holes was satisfactory, and although no ore was found valuable geologic information was gained. Broken ground in the faulted limestone stopped the holes before they could reach ore.

TINTIC STANDARD MINE, TINTIC DISTRICT, UTAH⁶²

A very small amount of diamond drilling has been done. The ground is much faulted and broken.

PECOS MINE, SAN MIGUEL COUNTY, N. MEX.⁶³

Ore is a complex mixture of sulphides replacing schist in a shear zone. Extensions along strike are diamond drilled from the surface, while walls and bottom are drilled underground. Core recovery is about 75 per cent. Sludge samples are unreliable; if sludge assays indicate ore, core is split and assayed. Drill-hole assays are considered in estimating reserves. Diamond-drilling data, 1927 to 1929, inclusive, are as follows:

60 holes underground, air power.....	feet.....	15, 185
8 holes on surface, gas engine.....	do.....	7, 916
68 holes total.....	do.....	23, 114

Total cost contract plus company expense was \$2.99 per foot.

BLACK ROCK MINE, BUTTE, MONT.⁶⁴

Ore is blende and galena in veins in granite, averaging 6 feet in width. Diamond drilling has been abandoned, since the meager results did not justify the cost.

⁶⁰ Coy, Harley A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6239, Bureau of Mines, 1930, 11 pp.

⁶¹ Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, Bureau of Mines, 1930, 18 pp.

⁶² Wade, James W., Mining Methods and Costs at the Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, 21 pp.

⁶³ Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

⁶⁴ McGilvra, D. B., and Healy, A. J., Method of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, Bureau of Mines, 1930, 16 pp.

SILVER KING MINE, PARK CITY, UTAH⁶⁵

Due to irregularities of ore bodies, diamond-drilling results have only provided geological information.

PAGE MINE, PAGE, IDAHO⁶⁶

Diamond drilling is unsatisfactory, since there is no core recovery in the soft vein filling, and holes cave badly.

EL POTOSI MINE, SANTA EULALIA DISTRICT, CHIHUAHUA, MEXICO⁶⁷

Ore bodies are irregular lead-silver deposits in limestone. Much drilling has been done underground with machines having a capacity of 800 feet and using standard E core barrels, rods, and bits taking $\frac{1\frac{1}{2}}{8}$ -inch core. Speed in good ground averaged 50 to 100 feet per eight hours. In 1922, 3,300 feet per month was the average drilling rate. Carbon loss in one case was 5.91 carats in 12,650 feet, or 0.000469 carat per foot. In carbonate ore blank bits or bits set with pieces of file give good results. Rods are pulled every 2 or 3 feet. In sulphide ore recovery of core is high (up to 95 per cent); and assays are more reliable than in carbonate ore, due to the greater hardness. Rods are pulled at 5 or 10 foot intervals in sulphide. The maximum depth of holes is 600 feet. Many holes are drilled ("fanned") from one set-up at several different inclinations (never upward). The entire area of the mine is thoroughly drilled, so that not more than 100 feet of virgin ground intervenes between holes. Deflection is unimportant. This drilling is important in locating the tops of ore bodies, thus permitting the best method of development for underhand stoping. Drilling is done on contract by Mexicans under an American boss who sets the bits. Runners receive \$0.085 per foot, \$2.375 per shift guaranteed. Helpers get \$0.055 per foot, \$1.85 per shift guaranteed. One typical hole 600 feet deep was drilled in 20 shifts, with a carbon loss of 0.94 carat; supplies cost \$8.08 and labor \$87.50. The average speed in limestone was 45 feet per eight hours, maximum 110 feet. Holes are carefully logged and mapped; sludge and most of the core are assayed. Core assays are usually higher than sludge due to settling of heavy particles in the hole. Discovery of extensive ore bodies and development of ample reserves were largely due to the diamond drill.

GOLD MINES

TECK-HUGHES MINES, KIRKLAND LAKE, ONTARIO⁶⁸

Diamond-drill holes are used to explore the fault zone at 30 to 100 foot intervals along drifts. Results are not used in calculating reserves.

⁶⁵ Dailey, M. J., *Mining Methods and Costs of the Silver King Coalition Mines Co., Park City, Utah*: Inf. Circ. 6371, Bureau of Mines, 1930, 11 pp.

⁶⁶ Berg, J. E., *Mining Methods at the Page Mine of the Federal Mining & Smelting Co., Page, Idaho*: Inf. Circ. 6372, Bureau of Mines, 1930, 8 pp.

⁶⁷ Walker, Harlan A., *Diamond Drilling in El Potosi Mine*: Eng. and Min. Jour., vol. 114, 1922, pp. 896-902.

⁶⁸ Henry, R. J., *Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario*: Inf. Circ. 6322, Bureau of Mines, 1930, 11 pp.

SPRING HILL MINE, HELENA, MONT.⁶⁹

Underground diamond drills are used for exploratory crosscutting where it is possible to get holes through to the contact; otherwise drifting must be used. Drilling is on company account and costs \$3 per foot.

SALT DEPOSITS

In drilling salt deposits special apparatus and precautions are required. This subject is discussed in detail by Wroth.⁷⁰ Water can not be used as the core would be dissolved; therefore brine solution is employed instead of water.

COSTS

Typical costs of diamond drilling are shown in Table 5. Drilling is commonly done by contract on a footage basis, but is also conducted on company account at some properties.

⁶⁹ Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Inf. Circ. 6402, Bureau of Mines, 1931, 11 pp.

⁷⁰ Wroth, James S., Special Features of Core Drilling in Salt Beds of Western Texas and New Mexico: Inf. Circ. 6156, Bureau of Mines, 1929, 13 pp.

TABLE 5.—Examples of diamond-drilling costs

District	Mine	Material drilled	Size of bit or core, inches	Approximate depth of holes, feet	Cost per foot	Remarks
Mineville, N. Y.	Witherbee-Sherman	Magnetite ore in gneiss.	1-inch core		\$3.92	
Ducktown, Tenn.	Burra-Burra	Schist, graywacke, massive sulphides.	do.		\$2.50	
Fierro, N. Mex.	Hanover-Bessemer	Magnetite in limestone.		200 to 500.	Drilling..... \$0.967 Setting bits..... .081 Carbon..... .207 Moving..... .186 Total..... 1.441	
Plumas Co., California	Engels	Diorite.			Contract up to 400 feet, less than 15° inclination; \$2.45. Contract up to 400 feet, over 15° inclination; \$2.70.	
British Columbia	Brittania	Chlorite, quartz-sericite schist.	1 $\frac{1}{16}$ O. S. diameter; $\frac{7}{8}$ core.		Direct cost very siliceous ground; \$2.46. Total cost very siliceous ground; \$2.99. Direct cost less siliceous ground; \$1.66. Total cost less siliceous ground; \$1.77.	Underground drilling.
Southeast Missouri.	No. 8	Limestone, shale		Average 600	\$0.95 to \$1.40	
San Miguel County, N. Mex.	Pecos	Granite and diorite		(Average 250 u. g. Average 990 surf.	\$2.90	
El Potosi, Mexico		Lead-silver ore in limestone.		600.	Labor..... \$0.15 Supplies..... .01 Carbon..... .12 Total..... .28	Native labor. Power not included. One hole only.
Helena, Mont.	Spring Hill	Limestone and diorite.			\$3.	
Rossland, B. C.	Josie	Hard augite and diorite porphyry.	1 $\frac{1}{16}$ O. S.	Up to 1,000 feet.	Average \$1.92	1902 to 1919, 120,762 feet underground drilling.
Alaska	Alaska-Treadwell	Diorite, quartz, greenstone, and slate.	1 $\frac{1}{8}$ core		Labor..... \$0.798 Diamonds..... .637 Repairs..... .012 Supplies..... .083 Assaying..... .002 Power..... .211 Total..... 1.743	5 months, 1913.

Miami, Ariz. ³	Miami	Porphyry	1 $\frac{5}{16}$ core; 1 $\frac{1}{16}$ O. S.	Average 208	Labor..... \$2.68 Carbons..... .82 Miscellaneous supplies... .84 Rent and depreciation... 1.10 Total..... 5.44	May, 1912, to June, 1913, inclusive, 2,714 feet.
Sudbury, Ont. ⁴	Canadian Copper Co.	Norite, diabase, granite, greenstone.	$\frac{3}{8}$ core	Average 800	Labor..... \$0.90 Carbons..... 1.12 Coal..... .22 Supplies..... .04 Depreciation, water, etc... .25 Total..... 2.53	100,000 feet of drilling.
Edwards, N. Y.		Zinc ore in crystalline dolomite; some gneiss.		Average 338	Total..... 2.53 \$2.05	3,386 feet total in 1930; good core recovery.

¹ Couldrey, Paul S., and Sampson, E. H. S., Diamond Drilling as a Means of Intensive Development: Eng. and Min. Jour., vol. 110, Sept. 25, 1920, pp. 608-610.

² Engineering and Mining Journal., Diamond Drilling at Alaska Treadwell: Vol. 97, Jan. 17, 1914, p. 177.

³ Naething, Foster S.; Diamond Drilling at Miami: Eng. and Min. Jour., vol. 97, May 23, 1914, pp. 1039-1041.

⁴ Parsons, L. A.; Diamond Drilling at Sudbury: Eng. and Min. Jour., vol. 102, Aug. 26, 1916, pp. 381-382.

HAMMER DRILLS

Hammer machines are now used quite widely for sampling and exploration. Their application to such work can be divided into two classes—sampling and testing with ordinary drilling equipment to shallow depths of 20 feet or less and “long-hole” drilling with heavy machines and special equipment to depths up to 250 feet or more.

SHALLOW DRILLING

Shallow drilling is a simple and often very reliable method of sampling underground faces. One method commonly employed for stope-control sampling is to catch the cuttings from one or more of the holes drilled each shift for breaking ore. Shallow holes in the backs of drifts or faces of stopes frequently serve as checks on channel or pick samples. In this case holes may be drilled up to 5 or 6 feet in depth at regular intervals of about the same distance, or shallow holes may be drilled on a closer spacing. Many engineers find that the empirical rule of making the depth of these holes about equal to their spacing gives most satisfactory results.

For ore bodies where the mineralization is spotty and irregular shallow holes normal to the structure often furnish the most reliable small samples. Prescott⁷¹ has discussed the applicability of this type of sampling to irregular replacement ores in limestone. He advocates vertical holes 6 to 12 inches deep and spaced 12 to 18 inches apart for fairly uniform ore free from marked banding. If horizontal banding is present back channels would be very misleading, and wall channels might likewise be unreliable; under such conditions vertical holes 3 feet or more deep on 3-foot centers are recommended. For vertical banding, angle holes at 45° spaced so that one hole ends in the band in which the next starts gives good results. Channels would be just as satisfactory for such an exposure if the ore and gangue possessed similar hardness and friability, however.

Shallow holes are sometimes drilled into drift walls in wide veins, in order that the breast may be kept centered in the best ore.

Ordinary machines are used for shallow-hole sampling—stoppers for back holes and jack hammers or drifters for flat and down holes. Up holes are generally drilled dry, since the cuttings will drop out by gravity and the vibration of the steel in the hole, and loss due to sludge runing down the steel or along the rock face is avoided. For dry holes the best outfit for catching the cuttings is probably a canvas sack held around the collar, the steel passing through a hole near the bottom of the bag. A rubber gasket on the steel inside the bag minimizes loss of dust through the hole. Powder boxes or pans are often used; but much dust is lost, and samples may be inaccurate with this method.

For catching sludge in wet drilling several methods are employed, including the use of cans or powder boxes placed below the hole, small pans held below the collar by an assistant, kidney-shaped

⁷¹ Prescott, Basil, *Sampling and Estimating Cordilleran Lead-Silver Limestone Replacement Deposits*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 674-676.

pans held around the steel for inclined or vertical holes, small sludge boxes below launders, and sample splitters. (See "Long-hole drilling.")

EXAMPLES OF PRACTICE

A few representative examples of practice are given below, in order to illustrate the various applications of shallow-drill sampling; the method as used at a considerable number of American mines is further discussed under "Underground sampling."

PILARES MINE, SONORA, MEXICO

At Pilares,⁷² inclined 5-foot holes are drilled into the back with a dry stoper at 5-foot intervals, in workings that are in ore, and the samples are used as checks on the grab samples taken each round. Similar holes are drilled in each corner of raises at 5-foot vertical intervals; and in stopes drill holes are used to sample backs in doubtful ground or to check assays from grab samples, if necessary.

TECK-HUGHES MINES, KIRKLAND LAKE, ONTARIO

At the Teck-Hughes⁷³ property in Ontario 5-foot holes are drilled in the walls of drifts with drifter machines at 10-foot intervals to test the wall ground and to keep the face in the best ore. Sludges are assayed in two sections. Results are used in figuring reserves.

ENGELS MINE, PLUMAS COUNTY, CALIF.

At Engels, Calif.,⁷⁴ the cuttings from two or three holes in each stope are collected daily as an aid to controlling stope operations. Holes are flat, being drilled with mounted drifters to an average depth of 7.7 feet.

HANOVER MINE, FIERRO, N. MEX.

Somewhat longer holes are employed at Fierro, N. Mex.,⁷⁵ where the ore is magnetite, occurring as irregular lenses. Jack hammers are in constant use to drill 20-foot holes in advance of all headings. Many valuable lenses of ore have thus been discovered which otherwise would have been missed, and important information is gained at low cost. The grade of ore is calculated on the basis of the assays from these drill-hole samples.

SOUTHEAST MISSOURI DISTRICT

Somewhat similar use of this method is made at a mine⁷⁶ in the Southeast Missouri lead district, where jack hammers mounted on columns and equipped with pneumatic feed are utilized for testing

⁷² Leland, Everard, *Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico*: Inf. Circ. 6307, Bureau of Mines, 1930, p. 6.

⁷³ Henry, R. J., *Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario*: Inf. Circ. 6322, Bureau of Mines, 1930, pp. 2-3.

⁷⁴ Nelson, W. I., *Mining Methods and Costs at the Engels Mine, Plumas County, Calif.*: Inf. Circ. 6260, Bureau of Mines, 1930, p. 3.

⁷⁵ Kniffin, Lloyd M., *Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.*: Inf. Circ. 6361, Bureau of Mines, 1930, 20 pp.

⁷⁶ Jackson, Charles F., *Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District*: Inf. Circ. 6170, Bureau of Mines, 1929, pp. 5-6.

backs, floors, and marginal areas of underground openings. Holes are drilled to a maximum depth of 22 feet. Backs are drilled on the corners of 25-foot squares. No water is used, cuttings are caught in a pan which fits around the steel, and cuttings from each 2 feet constitute a sample.

ACME MINE, WACO DISTRICT

At the Acme mine ⁷⁷ in the Tri-State district piston drills mounted on tripods are used to sample ground below the bottom level. Holes average 17 feet in depth, and if the collar is kept well cleaned fairly reliable results are obtained.

LONG-HOLE DRILLING

“Long-hole” drilling with heavy hammer type machines and sectional steel is a comparatively recent development in underground sampling and exploration methods. The North Butte Mining Co. conducted experiments on the use of the hammer drill for prospecting purposes some years ago (1922) and was one of the earliest, if not the first, to undertake such a program. Later, ⁷⁸ in 1923, experiments at the Chief Consolidated mines in the Tintic district resulted in the development of satisfactory equipment for drilling to unprecedented depths with heavy drifter machines. An intensive program of drilling followed, and holes more than 200 feet deep were drilled. The greatest depth attained to May, 1925, was 272 feet. ⁷⁹ During 19 months, ending with April, 1925, a total of 36,262 feet was drilled in limestone at an average cost of \$0.97 per foot. Standard “long-hole” equipment is now on the market, largely as a result of the work at Chief Consolidated.

Early use of hammer machines for drilling exploratory holes in the Southeast Missouri lead district was described by Poston in 1924. ⁸⁰ Heavy Leyner-type machines were used in the work, and 75 holes averaging 35 feet in depth (78 feet, maximum) were drilled at a contract-labor cost of \$0.16 per foot.

TYPE OF EQUIPMENT

The equipment now in general use has become rather well standardized. Its successful development has hinged on the evolution of sectionalized steel, or drill rods, having couplings strong enough to transmit heavy, rapid blows from the drill to the bit and able in addition to withstand the torsional stress imposed by rotation of relatively long, heavy strings of steel, without being so large as to choke the hole and obstruct the passage of cuttings. Various arrangements, including threaded steel pipe, diamond-drill rods, and ordinary drill steel in sections with upset ends and taper-

⁷⁷ Banks, Leon M., *Mining Methods and Costs in the Waco District*: Inf. Circ. 6150, Bureau of Mines, 1929, p. 3.

⁷⁸ Drullard, H. R., *Drilling Deep Holes with Hammer Drills and Sectional Drill Rods*: Eng. and Min. Jour., vol. 117, 1924, pp. 384-386. Dobbel, Charles A., *Deep-Hole Prospecting at the Chief Consolidated Mines*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 677-689.

⁷⁹ Dobbel, Charles A., work cited, p. 685.

⁸⁰ Poston, Roy H., *Leyner Drill in Underground Prospecting*: Eng. and Min. Jour., vol. 117, 1924, pp. 856-857.

threaded male and female joints were tried⁸¹ in the early stages of long-hole drilling, but all failed to give entire satisfaction in deep holes. The joint finally developed was a left-hand sleeve coupling having double threads on a pitch of one-half inch (for 1¼-inch steel). With this sleeve joint the ends of the steel butt together and take the full force of the blow; the sleeve transmits the rotation and cranking-out motions. Rods are usually ordinary 1¼-inch hollow round steel in 3, 6, and 9 foot lengths.

Different types of machines have been tried, including several kinds of drifters, jack hammers, and even stopers. For deep holes of 100 to 150 feet or more and for steady reliable service, the heavy drifter is most satisfactory; for shallower work lighter, 1-man types enter the field.

Independent rotation is essential to satisfactory operation, since the long string of steel has an inertia out of all proportion to the load for which the usual self-rotation mechanisms are designed. Reversible rotation is generally also provided to assist in unscrewing the rods and in starting stuck steel.

In deep drilling a water swivel is attached on the rod ahead of the chuck, and the regular water and air inlets are sealed off. Various swivel designs have been evolved. The device used in the work at Chief Consolidated was adjusted to deliver water during five-sixths of a rotation of the steel and air during the remaining one-sixth when desired on down holes; this gave a pulsating air-lift effect, which was useful in cleaning sludge from down holes.⁸² On horizontal and up holes water only, at 150 pounds pressure per square inch, was used.

The swivel employed at St. Francois, Mo., was fitted with an air valve for hand operation.⁸³

The set-up most commonly used for the heavier machines consists of a cross arm clamped at the ends to two vertical columns, although in openings more than 7 or 8 feet high, tripods give satisfactory results if carefully set up and involve less delay and expense.⁸⁴ When lighter machines are employed, a single column and arm will often suffice for a careful drill runner.

INCLINATION AND DEPTH OF HOLES

Holes can usually be drilled at any inclination desired, but most satisfactory results are obtained with a pitch of 5° to 30° above the horizontal, since the weight of steel does not then present any great difficulties in handling, and conditions are most favorable for recovery of sludge. With highly inclined up holes a counterweight arrangement to balance the great weight of steel is necessary at depths greater than 75 to 100 feet and sometimes for much shallower holes. Steep or vertical down holes are feasible, although complete recovery of sludge is more difficult, and sticking of bits due to the settling of particles must be contended with unless the hole is thoroughly cleaned before steel is added.

⁸¹ Drullard, H., and Dobbel, Charles, works cited.

⁸² Dobbel, Charles, work cited.

⁸³ Poston, Roy H., work cited.

⁸⁴ Netzeband, W. F., *Prospecting with the Long-Hole Drill in the Tri-State Zinc-Lead District: Min. and Met.*, June, 1930, pp. 295-296.

The maximum depths attainable by hammer drilling vary widely under different conditions. The chief limiting factor is the hardness of the rock drilled, since this property controls the loss of gage in drilling. At some mines where the rock is extremely siliceous and hard holes 50 feet deep can rarely be drilled, while at other properties in favorable ground, such as that afforded by limestones, remarkable depths have been reached with hammer drills. The 272-foot hole already mentioned is the deepest one of which a fairly thorough search of the literature has revealed any published record, although a 300-foot hole⁸⁵ was confidently planned at the Tonopah-Belmont in 1926; no record of the success of this hole was noted in the literature.

DIFFICULTIES IN OPERATION

Some operators have been dissatisfied with deep-drilling results because loss of gage prevented the attainment of the depths desired. In many cases such results are unavoidable, but often they are due to lack of familiarity with the equipment and an imperfect conception of the results that can be achieved by the machine in the hands of a skillful runner. Loss of gage may in some instances be largely offset by following dulled bits with sharp ones of the same gage and carefully reaming the hole. Investigation of the steel sharpening and treatment may reveal the possibility of prolonging the usefulness of each bit. Stellited bits⁸⁶ for this service have been employed at Anaconda and elsewhere, with gratifying results. When the distance to a certain objective, such as a contact, is approximately known, the feet each size of bit must drill can be roughly estimated and the procedure of reaming as against continued drilling with dulling bits, etc., governed accordingly.

Permissible reduction in gage is less than in the ordinary drilling done for blasting purposes, since the hole must not only be large enough to pass the rod couplings but must in addition provide room for ejection of the cuttings. Clogging of sludge in holes of reduced diameter not only is liable to induce sticking of bits and breaking of rods but also results in a pronounced lag in the appearance of cuttings at the collar or sometimes in incomplete recovery, thus in part vitiating the results. The use of smaller-diameter steel in front of the standard size (1 inch ahead of 1¼ inch) at Chief Consolidated was not wholly satisfactory, since the lighter steel was subject to considerable vibration,⁸⁷ with resultant low effectiveness in transmitting blows to the face in long holes.

Broken or fissured ground presents obstacles, as in other methods of drilling; open fissures generally mean loss of the hole, since return of sludge is rendered impossible and mechanical difficulties of drilling are increased. Such openings can sometimes be bridged with a length of pipe by drilling a short distance into the far wall of the cavity, placing and wedging the casing, and continuing the hole with a smaller bit. Bains⁸⁸ has described the successful appli-

⁸⁵ Brown, R. K., *Exploratory Deep-Hole Drilling*: Comp. Air Mag., vol. 31, April, 1926, pp. 1593-1594.

⁸⁶ Bains, T. M., jr., *The Gasoline Engine as a Mining Power Unit*: Eng. and Min. Jour., vol. 125, 1928, pp. 1051-1052.

⁸⁷ Dobbel, Charles, work cited, p. 681.

⁸⁸ Bains, T. M., jr., work cited.

cation of this idea at the Memphis property in New Mexico. If the hole had not already been much reduced in gage when the water-course was struck, results were satisfactory.

Cementing or grouting is quite possible in this type of drilling, but generally it is more economical to abandon the hole and try again at a different location or direction than to undergo the expense of cementing, since the investment in a hole less than 150 feet deep is relatively not great, as compared to deeper and more costly bores where a considerable outlay of time and money would be justified for grouting a bad section in the hole. At the Tonopah-Belmont⁸⁹ a hole which had struck an open fissure at 103 feet was drilled a few feet farther in the hope that sludge would plug the crevice, but no water was returned. The hole was abandoned and a new one, started 5° to the left, was bottomed at 225 feet without loss of sludge.

Loss of time due to stuck bits or broken rods varies with the hardness of the ground, degree of fracturing, and other characteristics and is also in more or less direct proportion to the experience and degree of skill of the driller, as well as the quality of the equipment used. Satisfactory fishing tools have been developed and described in the technical press. No discussion of that phase of the subject will be given here.

Advance per machine shift ranges from 8 feet in excessively hard ground to 50 feet or more under the best conditions; the average advance is generally about 25 feet.

SAMPLING PRACTICE

Sampling the cuttings is of primary importance in any drilling program. It is a surprising fact that this feature of the work is often entrusted to the unsupervised drilling crew—men who are usually expert drill runners but often inexperienced in drill-hole sampling and ignorant or careless of its importance. Such men are out for footage and unless constantly watched are inclined to pay little heed to what is happening in the sludge box.

The most common method of sampling long holes consists simply in placing a powder box or carbide can below the collar of the hole where it will catch most of the sludge and replacing it with an empty box or can after the 3 feet or other distance chosen for the sample interval is drilled. Naturally a considerable loss of slimes is the corollary of such imperfect makeshifts and doubtless is often responsible for the "unreliable" assay returns reported by many operators for their long-hole drilling. The powder-box method has given accurate results in some instances,⁹⁰ but only because the fines carried about the same proportion of values as the coarse cuttings. When the ore is known to contain values in approximately uniform distribution throughout the various sizes from coarse chips to the finest slimes, a rough method which obtains enough of the material for assay may be sufficiently refined. On the other hand, in the more usual case of disproportionate values over the range of particle sizes that occur in drill sludge no sample can be accurate or reliable which

⁸⁹ Brown, R. K., *Exploratory Deep-Hole Drilling*: *Comp. Air Mag.*, vol. 31, April, 1926, pp. 1593-1594.

⁹⁰ Dobbell, Charles, work cited, pp. 682-683.

is not truly representative of the entire volume of material broken by the bit. If it is worth while to undergo the expense of long-hole drilling at all it should certainly be worth while to go to the small additional trouble and expense of rendering the information gained from that drilling as complete and as accurate as possible. Some operators have found the long-hole drill extremely useful for proving or disproving the presence of ore but useless for providing any reliable information as to grade. Although in many instances the physical nature of the ground and the ore prevent accurate sampling by hammer-drill methods, in many others the faulty assay returns can be traced to faulty technique at the collar of the hole.

For accurate sampling of long holes, two requirements must be met. First, all the material cut during the sample interval must be driven from the hole; and second, all this material or a representative portion of it must be collected and included in the sample sent to the assay office.

The first requirement is the more easily satisfied. Most holes are drilled at a low angle above the horizontal—from 5° to 30° , according to local conditions—such that the sludge is readily washed out by the return water. In down holes air usually is required to aid the water in lifting out the cutting; the manner of introducing air has been already mentioned. In many instances it is impossible, in part because of the obstructions offered by the rod couplings, to clean out down holes completely after they have attained depth and suffered more or less reduction of gage, unless the rods are pulled and a blow-pipe is used. Holes inclined steeply upward present no difficulties in the matter of discharging sludge.

The second requirement—that of catching the sludge as it comes from the hole—is the weak link in the long-hole sampling chain. The common practice of placing a box or can beneath the collar has been discussed. A better scheme is to fix a piece of sheet metal, bent into the form of a shallow spout or lip, beneath the hole to prevent sludge from running down the wall behind the can or box. Probably the most effective method of catching sludge is that employed at Tonopah⁹¹ and New Idria,⁹² among other places.

A short piece of pipe (about 3 inches in diameter and 1 to 2 feet long) is split longitudinally for all but a few inches of its length, the split portion is spread open to form a launder, and the unsplit end is inserted in a hole drilled for the purpose a few inches below the long hole and connecting with it 6 to 12 inches from the collar. This launder or spout feeds the sludge into containers and is very effective in preventing loss by running down the walls.

With holes highly inclined above the horizontal it is difficult to effect a complete recovery of sludge on account of its tendency to run down the rods. So far as we are aware no entirely satisfactory means to combat this difficulty has been devised. Pans, gaskets, and sacks have been employed with indifferent or only partial success. A short piece of casing in the collar of the hole, fitted with a stuffing box of some form, might be applicable, provided the difficulties of vibration and possible impaired drilling efficiency could be overcome.

⁹¹ Brown, R. K., work cited.

⁹² Moorehead, W. R., *Methods and Costs of Mining Quicksilver Ore at the New Idria Mine, San Benito County, Calif.*: Inf. Circ. 6462, Bureau of Mines, 1931, pp. 3-4.

Containers for catching sludge may consist simply of a powder box or single carbide can as previously mentioned, or of settling tubs, sludge boxes, or other arrangements. Many mines reduce the volume of material handled by cutting out part of the sludge in a riffle-sample splitter; if carefully done this does not seem to vitiate the results.

When one small container is used slimes are inevitably lost. To prevent such loss, sludge receptacles should provide ample opportunity for settlement of fine material. A large tub will often accomplish this purpose when a sample splitter is used to reduce the volume retained. In this case the water is decanted when clear, and the solids are cleaned out from the tub and filtered or dried over a fire.

A satisfactory method of recovering sludge—one that has proved itself at several mines—has been described as used at Tonopah.⁹³ With this set-up the sludge flows from a launder into a carbide can, overflowing thence through two successively shorter cans at lower elevations. Virtually all the cuttings settle out before the final overflow leaves the system as clear water comparatively free of solids. At the end of each sample interval the clear water in the cans is decanted; the contents of the two smaller cans are dumped into the first can and allowed to settle, when the remaining water is decanted and the sample is collected for drying and assay.

Cuttings should always receive regular attention by a geologist or other experienced man familiar with the ore and country rock. At many mines a regular file of cuttings is maintained, the material being kept in small jars or bottles or mounted on cards on which has been noted all pertinent information, such as location, direction and depth of hole, dates of drilling, position of the sample in the hole, assay (if ore), petrographic notes, and the like.

RECORDING DATA

Long holes should be logged in accurate detail in the same general way as diamond or churn drill holes. Generally a driller's log and a geologist's log are kept, the former giving such information as location, course and inclination of hole, dates started and stopped, depths drilled each shift, bits used, samples taken (if taken by operator), time lost in delays, and general remarks, such as changes in formation, color of sludge, or hardness, fractured or broken ground noted by behavior of drill, loss of return water, and so on.

The geologist's log should record the location, direction, depth of hole, and dates; feet of various formations penetrated, presence of gouge, broken ground, dikes, veins, contacts, and the like; description of rocks cut; assays where made; and similar data. It has been found possible in many instances to obtain a surprising amount of geological information from careful study of drill cuttings; by making detailed records of such information at the time the study is made the results are preserved as a source of reliable information for the future.

⁹³ Brown, R. K., work cited.

APPLICATION OF LONG-HOLE DRILLING

Long-hole drilling finds its principal application in the field of underground prospecting and exploration and is especially suitable where a relatively rapid and low-cost method for the systematic prospecting of vein walls is needed. For distances not greater than 150 to 175 feet it often competes favorably with diamond drilling. It is an efficient tool at mines where the ore occurs in the irregular, scattered bodies so typical of limestone replacement deposits and is very useful in the case of vein deposits in fractured areas where blind parallel lodes are apt to occur with no geological guides to their presence. It is worthy of note that in such instances it is just as necessary to disprove doubtful ground as to locate suspected ore bodies, and for such service a drill hole is generally as satisfactory as a crosscut and much cheaper. In general, crosscuts cost two to four times as much as diamond-drill holes and six to fifteen times as much as long holes. An average case may be taken as an example where crosscutting costs \$9 per foot and deep drilling costs \$1.25. If a large area adjacent to a drift is believed to be favorable prospecting territory it can be explored by drilling for less than one-seventh the cost of crosscutting, or conversely, can be prospected more than seven times as thoroughly for the same cost. Suppose it is desired to explore to a depth of 100 feet from a drift 1,000 feet long at 100-foot intervals. In average ground crosscuts will cost about \$9,000 and drill holes about \$1,250. If 1 drill hole out of 10 finds ore, or indications of ore, a crosscut can be driven along the line of the hole to open up the discovery; and the total cost for finding and opening the ore as well as disproving the rest of the doubtful ground along the drift will have been \$2,150, or less than one-quarter of the cost for which the same work could be done by crosscutting alone.

Long-hole drilling is useful for obtaining geological information. In regions where replacement ore occurs in favorable beds whose position is in doubt because of folding or faulting it is often the best means of locating these beds. It is sometimes used to determine the location of dikes, faults, formational contacts, and the like. Perhaps its widest application aside from prospecting for parallel ore consists in outlining the boundaries of ore bodies to facilitate planning the stoping layout at minimum expense. It is employed for this purpose at Ducktown, Tenn., on the Menominee iron range, at Cananea, Mexico, and at other places.

The value of the information gained by long-hole drilling is in direct ratio to the degree of care and skill exercised in catching the cuttings.

Judgment and experience are necessary for the best and most reliable results with long-hole machines. Proper allowance must be made, when the results are interpreted, for the possible effect of such factors as loose or caving ground, loss of water in porous or fissured areas and consequent incomplete or misleading sludge samples, salting of samples when passing through friable ore seams, such as occur at the Black Rock mine (see p. 67), erroneous impression of thickness of formations or ore bodies because of being cut at an oblique angle, and similar considerations. Familiarity with the local characteristics of ore occurrence will generally be of great assistance in placing holes to the greatest advantage.

ADVANTAGES

Summarized, the long-hole drill as used for exploring and sampling ore-bearing ground has the following advantages:

1. For depths up to a maximum of 150 to 250 feet (depending on the hardness of the ground) it generally has a very decided cost advantage over all other methods, if it can be used at all.

2. Runners can be recruited from among the more capable machine men already on the pay roll; highly specialized men of long experience are not essential, as in diamond drilling. Some operators have reported that the best drillers for this work are men without previous rock-drill experience and trained particularly for deep-hole drilling.

3. The equipment is compact and flexible and requires a minimum amount of room for operation. Frequently station cutting is unnecessary.

4. The equipment is simple and strong.

5. Loss of tools due to breakage does not involve the large financial loss that a similar misfortune entails in diamond drilling.

6. The method compares favorably with diamond drilling as regards speed.

7. The cuttings, properly handled, are usually a reliable source of information. In some instances they serve all the purposes of a solid core. (See "Disadvantages.")

8. The method is particularly adapted to drilling a large number of closely spaced holes, when the time saved in moving and setting up, the small space required, and the speed of drilling are manifest advantages. Information as to conditions in the walls at a certain place can probably be obtained more promptly by this than any other means.

9. Like the diamond drill, the machine often has considerable utility in driving long holes for drainage, ventilation, rescue work, or power conduits.

DISADVANTAGES

The principal disadvantages of the long-hole drill are:

1. Solid core is not obtained. As a result the information may be less complete than with diamond drilling; in some instances, however, study of the cuttings reveals as much as core examination, and when good core recovery is not possible this drawback disappears. (See "Advantages.")

2. Like other drill methods, this one is subject to erroneous results from dilution or salting in caving ground or when passing through streaks of brittle, friable ore. Because of the greater vibration of the steel, this source of error is apt to be more pronounced than in diamond drilling.

3. Misleading conceptions as to thickness of formations or ore bodies may result when holes cut them at flat angles.

4. In extremely hard ground progress may be slow and depth attainable very limited because of reduction in gage. Even the best steel and most skillful operation have proved ineffectual in combatting this obstacle in some instances.

5. In loose, fractured, or caving ground more or less trouble may be experienced from wedging or sticking of bits, especially with

inexperienced machine men, and efforts to free the steel may cause breakage of rods or couplings. The best remedy for such troubles is care based on experience.

EXAMPLES OF PRACTICE IN LONG-HOLE DRILLING

Long-hole drilling is now very widely practiced in metal-mining districts, serving variously as a means of random prospecting, a source of geological information, a systematic method of sampling ore zones or possible parallel ore bodies, and a means of sampling and delineating known ore at regular intervals. Summarized examples of practice at North American mines are given below; these are chiefly derived from information circulars published by the Bureau of Mines.

COPPER MINES

Burra-Burra mine, Ducktown, Tenn.—At the Burra-Burra mine in Tennessee the ore is composed of massive sulphides (pyrrhotite, pyrite, and chalcopyrite) in a silicate-quartz-calcite gangue; the wall rocks are metamorphosed schists and graywackes. McNaughton⁹⁴ discusses long-hole (deep-hole) drilling at Ducktown as follows:

Since the ore zone is practically continuous, exploration is principally by sinking, crosscutting to the ore zone, and drifting on the strike of the ore body. The ore bodies are outlined by crosscutting holes from the exploratory drifts. Long holes are diamond drilled, whereas holes less than 150 feet long are drilled with a deep-hole hammer drill. The last diamond drilling was done by the company with its own equipment and cost \$2.50 per foot. Assay records, diamond-drill cores, and deep-hole cuttings are filed; these file records are practically complete for more than 20 years.

The deep-hole drills are of the heavy drifter type with independent, reversible rotation. These drills employ 1¼-inch hollow-round steel with forged threads, made up in 3 and 6 foot lengths and with outside couplings. These drills are mounted on a crossbar secured to two columns with universal clamps. The double-taper cross bit is used, starting with 3¾-inch gage. Twenty-five gage changes are used between 3¾ and 2¼ inches. In medium ground and for holes not more than 100 feet in depth each bit is run until the hole size is down to that of the next gage. In harder ground, or for longer holes, as many as three bits of each gage are employed.

These machines do not work satisfactorily on flat or down holes. They give best results on holes about +15° from the horizontal. Their operation requires men of exceptional ability. The drills are always run wet; the water is introduced through a swivel connection in the side of the first drill rod, which is of special forged construction.

The crew consists of two men, a runner and a helper, who drill on the average about 25 feet per shift. The total cost of deep-hole drilling is 80 cents per foot, including labor, air, shop work, and steel.

Ray mines, Ray, Ariz.—At Ray, Ariz., the ore is a chalcocite-pyrite association disseminated in quartz-sericite schist, occurring as specks and small veinlets. Long-hole drilling in exploration has been described by Thomas⁹⁵ as follows:

The first prospecting of the Ray ore body was done by sinking shafts and drifting from them. The next prospecting was carried on by churn drilling, holes being drilled over the entire area at the corners of 200-foot squares whenever practicable. Further exploration has been carried on underground

⁹⁴ McNaughton, C. H., *Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.*: Inf. Circ. 6149, Bureau of Mines, 1929, pp. 3, 10-11.

⁹⁵ Thomas, Robert W., *Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.*: Inf. Circ. 6167, Bureau of Mines, 1929, pp. 5-6.

by diamond drilling, "long-hole" drilling, and through the usual procedure of drifting and raising. This last has been confined chiefly to determining extensions of the ore body outside of the limits indicated by the original churn drilling.

At present "long-hole" drilling has almost entirely replaced prospect raising. The drill used for this purpose is a Leyner-type machine with air-motor rotation and is mounted on a 42-inch cradle. It is fitted with a special water connection in front of the chuck. Two vertical columns and a cross arm are used for the set-up, which requires a station at least 5 by 7 feet in the clear. One and one-quarter-inch round-hollow steel is used in 3, 6, and 9 foot lengths, jointed by threaded sleeve couplings the shorter lengths being used only to make changes. The bits are detachable and are of the usual cross form with a 5° by 14° double taper. The largest-size bit is 3½-inches in diameter, and each reduction in gage is one-eighth inch. Holes up to 90 feet long are drilled usually on a 45° pitch. The pitch may be as great as 55°, and flat or downward holes can be drilled, though less readily, dependent upon the nature of the ground. A pump is used for forcing water through the hollow steel when a sufficient head of mine water is not available.

The average footage drilled per machine per shift over a long period has been 23 feet; this included moving and setting up.

The sample interval is 6 feet. Sludge is split at the collar of the hole by an automatic sampler, devised locally for this purpose. The sample is dried over a heater outside the mine before being sent to the assay office.

Elisa and Colorado mines, Sonora, Mexico.—At Cananea the long-hole machine has been used in the hard limestone and spotty sulphide ore of the Elisa mine, as well as in the Colorado where the ore occurs as sulphides with a quartz gangue in brecciated porphyry. Touching on this subject, Catron⁹⁶ states that:

A great many prospect holes have been drilled in the Elisa and Colorado mines with a heavy drifter-type machine. Its principal application has been the delimiting of ore bodies, serving the purpose of crosscuts through the veins, thus assisting in the laying out of development work. The holes have all been flat and have varied in length from 35 to 125 feet, averaging 85 feet.

The drill is similar to a drifter except that it has a swiveled water connection ahead of the drill chuck. A 1½-inch shank back of the swivel fits the machine chuck; the front end of the swivel receives the threaded shank of the 1¼-inch drill steel. The shank of the swivel connection is retempered after each hole. Mine water is always available under sufficient head and no pump is used. The drill is set on a cross arm between two columns. One and one-quarter inch hollow-round steel is used in 6-foot lengths with a few 3-foot lengths for making changes. The steel is connected by sleeve joints. The bits are formed on 3-foot lengths of steel; the starter bit is 3 inches in diameter and the succeeding bits decrease in size by one-sixteenth inch. The sludge is all saved, settled in tubs, and the water decanted. The drill crew consists of two men. Twelve holes, totaling 1,180 feet, were drilled in 64 shifts, including the time required to move and set up, or an average of 18½ feet per shift.

Verde Central mine, Jerome, Ariz.—In the Jerome district the copper-ore bodies occur in pre-Cambrian greenstones and quartz porphyry. In the Verde Central mine⁹⁷ lenses of ore in a mineralized shear zone are composed of quartz with disseminated pyrite and chalcopyrite. Except for some heavy ground near faults, both ore and wall rocks are generally hard. Holes up to 50 feet deep have been put into the walls with deep-drilling equipment.

⁹⁶ Catron, William, *Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico*: Inf. Circ. 6247, Bureau of Mines, 1930, p. 5.

⁹⁷ Dickson, Robert H., *Methods and Costs of Mining Copper Ore at the Verde Central Mines (Inc.), Jerome, Ariz.*: Inf. Circ. 6464, Bureau of Mines, 1931, p. 3.

SILVER-LEAD-ZINC MINES

Morning mine, Mullan, Idaho.—At the Morning mine galena and sphalerite occur with a gangue of siderite, barite, and quartz in a fissure vein in sheared or sheeted quartzite. Wethered and Coady⁹⁸ discuss drilling as follows:

The walls of the vein are prospected by long-hole drills. A 225-pound long-hole machine with independent air rotation is the type used. This machine uses a 1½-inch, round, lugged shank chuck, which is reduced by a sleeve ahead of the water swivel to 1¼ inches in order to use the standard 1¼-inch, round drill steel as rods. The gage of the bits starts with 3¼ inches and is reduced one-sixteenth inch at each change. Air consumption is 207 cubic feet per minute at 80 pounds air pressure.

Of a total of 62 holes drilled, the greatest depth obtained was 129 feet. The average depth is 46 feet. As many of the holes were not intended to go deeper than 25 or 30 feet, this average does not mean that a higher average could not have been obtained.

Drilling speed varies with the nature of the ground and depth of hole. For a deep hole more care is required from the start, and each bit is followed by one or more bits of the same gage. A hole 100 feet deep will not average more than 12 feet per 8-hour shift.

The labor cost, including two operators, mechanical labor on broken shanks, swivels, rods, etc., is 92 cents per foot. Supplies, including power but no charge for drill steel used in new rods, is 18 cents per foot. These costs are based upon 62 holes drilled with a total footage of 2,868 feet.

Page mine, Page, Idaho.—At the Page mine fine-grained galena and blende with some siderite occur in veins in highly shattered and broken quartzite. Attempts to use the long-hole machine for exploring walls have been disappointing, according to Berg,⁹⁹ who states that “the formation is so siliceous and hard away from the ore zone that the range of the machine is very limited and progress slow and costly.”

Park-Utah mine, Park City, Utah.—The silver-lead-zinc ores of the Park-Utah mine are found chiefly in fissures in limestone and quartzite. Alternating bands of barren limestone and medium hard sulphide ore occupy the space between the fissure walls. Deep drilling failed in the hard quartzite. According to Hewitt:¹

At the Park-Utah mine an attempt was made to explore laterally with large hammer drills operated by compressed air. Two holes were drilled in quartzite—one 66 feet and the other 88 feet deep. Both had to be abandoned before attaining the desired depth, because the steel failed in the hard rock. Jointed 1¼-inch hollow-round steel rods were used. The starting bit was of the rose type 3¼ inches in diameter, which was followed by bits of the usual cross form in 10 gages.

Chief Consolidated mine, Tintic district, Utah.—At the Chief Consolidated and other mines of the Tintic district, where the ores are irregular replacement deposits in limestone, the use of the long-hole drill has been attended with very marked success. Practice at the Chief Consolidated has been already briefly discussed.

Pecos mine, New Mexico.—At the Pecos mine sphalerite, galena, chalcopyrite, and pyrite, carrying values in gold and silver, are found in irregular lenticular ore bodies replacing schistose rocks.

⁹⁸ Wethered, C. E., and Coady, Leo J., *Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho*: Inf. Circ. 6238, Bureau of Mines, 1930, 13 pp.

⁹⁹ Berg, J. E., *Mining Methods of the Page Mine of the Federal Mining & Smelting Co., Page, Idaho*: Inf. Circ. 6372, Bureau of Mines, 1930, p. 3.

¹ Hewitt, E. A., *Mining Methods and Costs at the Park-Utah Mine, Park City, Utah*: Inf. Circ. 6290, Bureau of Mines, 1930, p. 3.

Talc, hornblende, mica, and chlorite are abundant in the gangue and wall rocks. In regard to prospect drilling Matson and Hoag² summarize practice as follows:

The shear zone is crosscut or drilled at frequent intervals. The walls and downward extensions are explored by diamond drillings from underground, and extensions along the strike are explored by diamond drillings from the surface. An E-S bit is used for this work. The walls are also tested for parallel ore shoots by deep-hole drilling with Leyner machines and sectional steel. These holes are usually between 50 and 100 feet in length. The steel used is 1¼-inch, hollow round, in 3, 6, and 9 foot lengths. Only two bit gages are used, as this was found more satisfactory than the several that were used originally.

The cost of 4,020 feet of deep-hole drilling was: Labor, \$1.26 per foot, supplies, \$0.72 per foot; total, 1.98 per foot.

Black Rock mine, Butte district, Montana.—At the Black Rock mine the ore ranges from soft, crumbly sphalerite to a hard flinty mixture of sphalerite and quartz; the vein is bounded by talcose seams on both walls, and lies within a granite country rock. Deep-hole drilling has been tried without much success. According to McGilvra and Healy³—

The chief objection to long-hole drilling in the Black Rock mine is that the information gained is entirely negative. A 1-inch stringer of soft ore would salt the samples from the hole for the next 10 or even 20 feet of drilling. The only results that could be depended upon were in holes where no mineralization was encountered.

Hecla and Star mines, Burke, Idaho.—At the Hecla and Star mines, where the silver-lead-zinc sulphide ores occur in veins in steep shear zones in quartzite, often with heavy gouge walls and irregular ore boundaries, deep-hole drilling has not been productive of positive results. Foreman⁴ states that:

When drifting and stoping holes are drilled into the walls at frequent intervals to prospect for parallel stringers. Prospecting by long-hole drills has not been satisfactory, except for negative results obtained. Diamond drilling has not been attempted.

LEAD AND ZINC MINES OF CENTRAL STATES

Tri-State district.—In the Tri-State lead-zinc district, where the bedded sulphides occur as “runs” or in channels in cherty dolomitic limestones, deep-hole machines have given general satisfaction. At the Acme property the drilling is done very cheaply, and results have amply justified the expense. Banks⁵ states in regard to this work:

Deep-hole drilling or prospecting with heavy hammer drills having independent rotation has been used extensively in the mines of this company. This drilling has cost \$0.644 per foot. The cuttings were found to be useful as indications of the presence of mineral but proved unreliable when assayed for grade determinations.

² Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, p. 3.

³ McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Montana: Inf. Circ. 6370, Bureau of Mines, 1930, p. 2.

⁴ Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Circ. 6232, Bureau of Mines, 1930, p. 4.

⁵ Banks, Leon M., Mining Methods and Costs in the Waco District: Inf. Circ. 6150, Bureau of Mines, 1929, p. 3.

In an excellent summary of practice in the district, Netzeband⁶ has presented data on the long-hole drilling done by seven leading companies. Tripod mounting is generally used, except in low headings where columns and crossbar can be handled easily. Costs have ranged from \$0.60 to \$1.90 per foot and have averaged \$1.28; results have been gratifying, in the main, although very loose ground and inexperienced runners have resulted in poor showings in some instances. With regard to the value of samples, the author states that:

It has been the experience of the district that no great reliance can be placed on the actual values found in the cuttings; to be of value the cuttings and the operation of the machine must be watched by a competent person. It is probably for this reason that many of the operators did not get really satisfactory service from their machines. Fairly accurate results can be obtained when the pitch of the hole is at a low angle, for practically all of the cuttings can then be saved, but when the hole is drilled at a steep angle and the collar of the hole is very far from the floor it is a rather difficult feat to catch all of the cuttings. As a result cuttings from such a hole are not accurate as to the zinc or lead content, but a careful study of the cuttings will usually tell a competent observer whether or not a mineable ore body is indicated.

Southeast Missouri.—In the Southeast Missouri lead district long-hole exploration has been attended with good results. Details of the earlier practice have been described by Poston.⁷ Some 75 holes averaged 35 feet in depth, the deepest one of 78 feet being inclined 5° above the horizontal. One man averaged 35 feet per shift, mostly with holes 30° up or down; labor cost on a contract basis was about \$0.16 per foot.

IRON MINES

Menominee range, Mich.—On the Menominee range deep holes are used to outline or determine the commercial limits of ore bodies. The ore is rather soft hematite between ferruginous slate foot and hanging walls. Interbedded ferruginous cherts are hard and abrasive. With regard to deep drilling, Eaton⁸ states as follows:

In "deep-hole" drilling a large hammer drill with a powerful independent rotation is mounted on a cross arm between two columns, and the hole is drilled at a slight angle upwards, using standard cross bits of large diameter. As the hole is deepened, hollow extension rods are screwed on the drill steel. Water under high pressure is forced through the rods and bit, and brings back the cuttings, which are caught in a sludge box. Care must be exercised to prevent too much loss of gage in the bit, as this limits the depth to which the hole can be drilled.

Hanover mine, Fierro, N. M.—Excessively hard ground was responsible for high costs at the Hanover Bessemer property, Fierro, N. Mex. Here the ore is massive magnetite and the country rock a metamorphosed limestone rich in hard silicate minerals. Kniffin⁹ summarizes results as follows:

Deep-hole drilling by the use of mounted Leyner machines with sectional drill steel was used for prospecting for a time. These holes proved more expensive than diamond drilling, and the information was not as reliable.

⁶ Netzeband, W. F., *Prospecting with the Long-Hole Drill in the Tri-State Zinc-Lead District: Min. and Met.*, June, 1930, pp. 295-296.

⁷ Poston, Roy H., *Leyner Drill in Underground Prospecting: Eng. and Min. Jour. Press*, vol. 118, 1924, pp. 856-857.

⁸ Eaton, Lucien, *Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Mich.: Inf. Circ. 6180, Bureau of Mines, 1929*, pp. 3-4.

⁹ Kniffin, Lloyd M., *Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Inf. Circ. 6361, Bureau of Mines, 1930*, p. 6.

The method is now seldom used. The cost of this work per foot drilled for 836 feet was as follows:

Cost per foot of deep-hole drilling

	Labor	Supplies	Repairs	Air	Total
Drilling.....	\$0. 877	\$0. 609	\$0. 524	\$0. 449	\$2. 459
Sharpening.....	. 161			. 008	. 169
Moving.....	. 179	. 014			. 193
Total.....	1. 217	. 623	. 524	. 457	2. 821

The couplings and other equipment were carefully made, and the high cost is attributed to the hardness of the rock and ore. The average advance per shift was only 8.84 feet.

GOLD AND SILVER MINES

Tonopah district, Nevada.—In the Tonopah district deep-hole drills have proved very effective. As described by Brown,¹⁰ the machines were employed in prospecting virgin ground adjacent to underground workings; to find faulted ore; or to sample known veins beyond existing openings. Two holes at 10° above the horizontal determine the strike of the veins, while a third inclined 30° upward permits calculation of the dip. Practice in sampling holes at this property has been discussed (p. 60). The following detailed record of the work for November, 1925, is of interest:

	Shifts	Hours	Per cent		Shifts	Hours	Per cent
Moving.....	7	6	9. 04	Cleaning hole.....	2	½	2. 40
Drilling.....	68	3	79. 73	No air.....	0	5	. 73
Fishing.....	3	4	4. 08	No water.....	2	½	2. 40
Machine maintenance and repair.....	1	3	1. 62		85	6	100. 00

Total feet.....	2, 345
Average feet per shift.....	27
Feet per drill-shift.....	34
Average feet per bit.....	6. 4

Bits	Gage	Footage	Bits	Gage	Footage	Bits	Gage	Footage
1 19	3½	55	49	2⅞	278	9	2⅞	57
83	3¼	643	33	2¾	223			
65	3⅛	419	28	2⅝	236	365		2, 345
58	3	293	21	2½	141			

¹ Rose.

Holes are usually drilled at plus 10° inclination, since at that angle they clean themselves of sludge.

Spring Hill mine, Helena, Mont.—At the Spring Hill mine the ore is an extremely hard mixture of pyrrhotite, pyrite, marcasite, and arsenopyrite, carrying native and combined gold, with lime silicate minerals abundant in the gangue. The walls, marbled limestone,

¹⁰ Brown, R. K., Exploratory Deep-Hole Drilling: Comp. Air Mag., vol. 31, April, 1926, pp. 1593-1594.

diorite, and massive pyrrhotite, are likewise hard. Pierce¹¹ describes exploration of the ore zone as follows:

Prospecting and exploration are carried on by a combination of drifting, crosscutting, raising, and diamond drilling. Where it is possible to get crosscut holes to the contact, diamond drilling is used; otherwise drifting is resorted to. Long-hole drills have been used but have not been very successful because in the hard rock the steel loses gage so rapidly that it is almost impossible to get a hole deeper than about 35 feet. Diamond drilling is done by the company at a cost of \$3 per foot.

OTHER ORES

New Idria mine, San Benito County, Calif.—Deep-hole drilling has been very helpful in locating ore at a minimum cost at the New Idria quicksilver mine. Methods there have been described by Moorehead,¹² and his summary is well worth repetition as an example of up-to-date practice.

Prior to 1927 all underground prospecting work was done by driving drifts, raises, and crosscuts. The erratic occurrence of the cinnabar in a complex system of fissures necessitated a large amount of costly work in searching for new ore bodies with the usual mine openings.

In 1927 a deep-hole prospecting machine was used on the lower levels of the mine with satisfactory results. The total footage drilled in the mine has been 8,641 feet. The number of holes drilled was 87, and the average depth per hole was 99.3 feet. The deepest hole drilled was 228 feet. The angle of inclination of the holes ranged from 8° to 50° above the horizontal; one hole was drilled to a depth of 147 feet at an angle of 78° below the horizontal.

The type of machine used was a heavy, independently rotated drifter. The principal requirement for drilling deep holes is sufficient water pressure to free the hole from sludge. At the New Idria a water pressure of 100 pounds per square inch was provided by tapping the water column in the shaft, and by the use of an auxiliary water swivel set in guides attached to the front head of the drifter.

Drill holes were started with a 3¼-inch standard cross bit and the succeeding bits were of the same pattern with a reduction in gage of one-eighth of an inch for each change. The smallest bit was 1¾-inch gage. The drill rods were made of 1¼-inch hollow round steel in four different lengths, starting with a 3-foot section and increasing 3 feet with each succeeding one. Six-inch sleeve couplings were used to join the rods; all threads on the rods and bits were left-handed to prevent unscrewing when rotated. The drifter was usually mounted on a crossbar supported by two vertical columns. In some cases, where the crossbar and vertical columns could not be set up without difficulty, a crossbar alone was used.

As the drill hole was advanced a 3-foot section of rod was replaced by a 6-foot section, the 6-foot section by a 9-foot section, and so on. When the bit was dull the entire string was pulled out with special wrenches, a new bit screwed on, and the rods again run into the hole. A block and tackle were used on long down holes when the rods were too heavy to lift. The objective to be attained determined how much footage it was necessary to obtain with each size of bit.

To recover broken rods from a long hole, a fishing tool was used made of 1½-inch iron pipe about 2½ feet in length with a bell-shaped mouth and slightly constricted in the middle to afford a firm grip on the 1¼-inch rods. The other end was reduced to 1 inch for connecting to 10-foot lengths of 1-inch pipe. The fishing tool was attached to a sufficient number of 10-foot lengths of 1-inch pipe to reach the broken rod. When the broken rod was reached the bell-shaped end was driven on the end of the rod and after a firm grip was obtained, the 1-inch pipe was turned with a pipe wrench until a rod coupling unscrewed. The broken rod was then removed and other rods inserted in the hole to pick up those remaining. The loss in breakage of rods was small.

¹¹ Pierce, A. L., *Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.*: Inf. Circ. 6402, Bureau of Mines, 1931, p. 3.

¹² Moorehead, W. R., *Methods and Costs of Mining Quicksilver Ore at the New Idria Mine, San Benito County, Calif.*: Inf. Circ. 6462, Bureau of Mines, 1931, pp. 3-4.

Samples were taken of the sludge for each 3 feet of hole drilled. To catch the sludge a 3-inch pipe was placed in a hole drilled a short distance below the drill hole. The end of the pipe which projected from the hole was split and the split portion turned back to form a launder. The sludge from the hole flowed into the launder and into carbide cans. Assays were made by panning. In holes drilled at a high angle the sludge had a tendency to run down the steel.

The average progress per day in drilling was 30 feet in medium hard shale and sandstone.

Two men comprised a drill crew—one machine runner and one helper. The average total cost per foot of hole was 75 cents, which includes the cost of moving and setting up, sharpening steel, and drill repairs as well as the cost of equipment which amounted to approximately \$1,800.

TEST-HOLE DRILLING DATA

Test-hole drill data from a number of typical operations are summarized in Table 6.

TABLE 6.—Typical test-hole drilling data

Mine or district and State	Kind of rock	Type of drill ¹	Depth of holes, feet	Feet per drill-shift	Purpose of drilling	Results reported	Cost per foot	
							Labor	Total
Burra-Burra, Tennessee	Schists, graywacke, massive sulphide.	A	Up to 150	25	Outlining ore body	Satisfactory; best on inclination +15° or over.		\$0.80
Acme, Tri-State	Cherty limestone	A			Prospecting walls	Cuttings useful indicator, assays unreliable.		.64
Do	do	B	17		Testing bottoms	Reliable samples		.57
Ray, Ariz.	Schist	A	Max. 90	23	Prospecting instead of raising +45°	Satisfactory		
No. 1, Menominee range, Michigan	Hematite and iron formation.	A			Testing walls	Good in soft ore; poor in hard chert.		
Morning, Idaho	Quartzite	A	{ Ave. 46 Max. 129 }	2 12	Testing vein walls	Satisfactory	\$0.92	1.10
Mascot, Tenn.	Dolomitic limestone	D			Sampling stope backs	do		
Cananea, Mexico	{ Porphyry and hard limestone.	A	{ 35 to 125 Ave. 85 }	18½	Flat holes to delimit ore bodies.	do		
Engels, California	Diorite	D	7.7		Cuttings from 2 or 3 stope holes daily for samples.	do		
Park-Utah, Utah	Quartzite	A	Max. 88		Prospecting 2 holes only	Unsatisfactory; steel failed		
Pilares, Mexico	Brecciated volcanics	E		5	Sampling	Satisfactory		
Teck-Hughes, Ontario	Silicified syenite and porphyry, hard.	D	5		Testing vein walls every 10 feet.	Satisfactory; assays not used in reserve estimates.		
Fierro, N. Mex.	Hard magnetite and limestone.	F	20		Prospecting ahead. Assays used in grade estimates.	Very satisfactory		
Do	do	A		8.84	Prospecting	More costly, less accurate than diamond drilling.	1.217	2.821
Pecos, N. Mex.	Schist and diorite	A	50 to 100		Prospecting for parallel ore bodies.	Satisfactory	1.26	1.98
Black Rock, Butte, Mont.	Granite	A			Prospecting. Not now used	Holes salted by soft-ore streaks. Not reliable for grade of ore.		
Page, Idaho	Quartzite	A			Prospecting walls. Not now used.	Limited range. Slow and costly in hard rock.		
Spring Hill, Mont.	Hard contact met. ore and rocks.	A	Max. 35		Exploring to contact	Unsatisfactory in hard rock		
Eagle - Picher lead, Oklahoma ³	Chert and limestone	A	Max. 148		Exploration	Good ore found. Valuable for negative information.		1.69
Evans-Wallowa lead, Oklahoma ³	do	A			do	Eliminated much ground thought ore bearing.		1.00
Federal M. & S. Co., Kansas ³	do	A	Max. 147		do	Located many new ore bodies.		1.90
Canam Metals, Oklahoma ³	do	A			do	Several ore bodies found. Cheaper than other methods.		1.70

Missouri-Kansas Zinc Corporation.	Chert and limestone	A	Max. 152		Exploration often at steep plus angles.	Very satisfactory		60- 70
New Idria, California	Shale, sandstone, serpentine	A	{ Ave. 99	} 30	Exploration	do		. 75
Chief Consolidated, Utah ⁴	Limestone and ore	A	Max. 272		23	do	do	⁵ . 44
Southeast Missouri ⁶	Limestone	A	35 to 78	35	do	75 holes drilled; satisfactory	⁷ . 16	
Do	do	C	Max. 22		Testing backs, floors, and walls.	Satisfactory		
Tonopah Belmont, Nevada, ⁸		A	Max. 256	34	Exploration, sampling walls	Good	(⁹)	
Jarbidge, Nevada ¹⁰	Volcanics		50 to 60		Exploration	As good as diamond drills and cheaper.		
Edwards, N. Y.	Zinc ore in dolomite	A	Ave. 43		Testing walls	25 per cent deducted from assays.		. 99

¹A, Heavy drifter with special independent rotation; B, piston machine on tripod; C, jack hammer with pneumatic feed; D, standard medium weight drifter; E, stopers; F, jack hammer.

²For holes up to 100 feet deep.

³Netzeband, W. F., Prospecting, with the Long-Hole Drill in the Tri-State Zinc-Lead District: Min. and Met., June, 1930, pp. 295-296.

⁴Dobbel, Chas. A., Deep-Hole Prospecting at the Chief Consolidated Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 677-689.

⁵Year 1924.

⁶Poston, Roy H., Leyner Drill in Underground Prospecting: Eng. and Min. Jour., vol. 118, Nov. 29, 1924, pp. 856-857.

⁷Approximate.

⁸Brown, R. K., Exploratory Deep-Hole Drilling: Comp. Air Mag., April, 1926, pp. 1593-1594.

⁹Time studies made. See footnote 8.

¹⁰Park, John Furness, Mining Methods in Jarbidge District: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 518-523.

Part 2.—UNDERGROUND SAMPLING

SAMPLING METHODS

In sampling outcrops, test pits, shafts, trench openings, drifts, and tunnels in prospect work the same methods are employed and the same precautions should be observed as in regular mine sampling, examples of which are given in the following pages.

CLASSIFICATION OF UNDERGROUND SAMPLING METHODS

Methods of underground sampling may be classified as follows:

1. Drill sampling:
 - (a) Churn-drill samples.
 - (b) Diamond-drill samples.
 - (c) Rock-drill samples (with piston or hammer drills).
2. Face sampling:
 - (a) Regular channel or groove samples.
 - (b) Pick samples.
3. Grab sampling:
 - (a) Muck-pile samples.
 - (b) Car and chute-lip samples.
4. Bulk sampling.

Drill sampling.—Drill sampling has been previously discussed. Methods employed are similar to those in surface drilling, although the equipment usually differs in detail.

Face sampling.—The term “face sampling,” as here used, covers the sampling of exposed faces of ore and waste whether they be faces in stopes or faces, backs, and ribs in development headings. It includes what is often termed “belt” or “ring” samples. Face samples are taken by (a) cutting grooves or channels of uniform depth and width across the face, or (b) by picking off small pieces all over the face more or less at random. The former is more methodical, and, as will be seen later, may or may not, depending upon conditions, be more accurate than the latter, although ordinarily it would be the method selected where the utmost accuracy possible is desired. In either case the face should be cleaned of dirt and loose rock by washing or with a pick before the sample is taken.

Channel sampling is generally believed to furnish the most scientific, accurate small samples, and often they are very reliable. In some cases, however, other methods of taking small samples have proved to be more accurate. In contrast to the usual grab samples they are generally taken by samplers who are carefully instructed and trained in their work and whose sole duty it is to cut samples.

This method of sampling is employed perhaps the most widely of any. It is extensively used for sampling outcrops and test pits in exploration work; for sampling partly developed and developed mines by examining and appraisal engineers; for guidance in devel-

opment work and as a basis for making ore-reserve estimates; and for control of stoping operations.

Unless the mineralization is distributed very uniformly the high degree of accuracy often obtained with estimates based on channel sampling is due more to the large number of samples taken with reasonable care and judgment than to individual samples accurately representing the material immediately adjacent thereto, errors being compensating.

Grab sampling.—Grab sampling consists of taking samples of broken material from muck piles, chutes, cars, or skips. It is commonly done rather unsystematically, as indicated by the term "grab," by taking random handfuls from various scattered points on the surface of the muck pile, or by taking a handful, scoopful or shovelful of material from each car or skip, as the case may be. As commonly practiced the method is often very inaccurate, due to three principal causes: (1) Ordinarily the sample will be composed almost entirely of fines, and the lumps will not be represented; (2) the human tendency is to select the richer material in an amount out of proportion to actual prominence in the ore; (3) it is difficult to get the type of men to whom this work usually is intrusted—often they are trammers—to follow any systematic rules that would make the sampling roughly automatic. Often the trammer will fill his sample box from the first few cars and then will take no more samples; or he will wait until the end of the shift and then fill the box from the last car. If directed to take a fixed proportion of the ore from a muck pile by putting 1 shovelful out of every 10 or whatever number is prescribed into the sample box, the shoveler is prone to fill up the box at any time during the shift that the thought occurs to him.

Although "grab" samples have been much criticized, the writers are inclined to believe that this is because of the unscientific manner in which they are usually taken, due largely to the above causes. Where the information to be gained from accurate grab samples has sufficient value to warrant the cost of getting them, it is believed that grab sampling can be done scientifically so as to give as accurate samples as other methods, and under certain conditions it will give more accurate results. Such methods will be discussed later.

Bulk sampling.—Bulk sampling consists in breaking down large samples of several tons or several hundreds or thousands of tons and either milling the entire tonnage, or, if the sample is relatively small, crushing the entire sample and cutting it down progressively in stages to obtain a representative sample. Such samples are sometimes taken to check small samples and to obtain a factor to be applied to the assay values derived from them.

EXAMPLES OF UNDERGROUND SAMPLING PRACTICE

Details of technique in underground sampling are best described by giving examples taken from actual practice.

For convenience in grouping as well as because of similarity of characteristics affecting sampling methods, examples are given under the headings of gold and silver ores, copper ores, lead ores, zinc ores, mixed or complex ores, and iron ores.

Since several methods of sampling are often employed at a mine and since the reasons for adopting certain methods in preference to others can best be brought out in this manner, the different methods (face sampling, channel and pick sampling, grab sampling, and other methods used as checks) are included together.

GOLD AND SILVER MINES

ARGONAUT MINE, AMADOR COUNTY, CALIF.¹

The gold occurs in quartz filling in a fault fissure, and the average width mined is 20 feet. Although the gold is not distributed uniformly in the ore shoot close sampling is not necessary. A good estimate of the grade of ore can be made by visual inspection. Where sulphides are present in the quartz the ore is invariably of good grade; the amount of sulphides, however, is not indicative of the richness of the ore. Square-set stoping is practiced.

Stope samples are taken by the trammer when the ore is drawn from the chutes, a handful of material being taken from each car. At the end of the shift all the chute samples are combined into one sample for assay. Development samples are taken in the same manner.

SPRING HILL MINE, HELENA, MONT.²

At the Spring Hill mine the ore body is a large, irregular, contact metamorphic deposit in which gold is the only valuable mineral. The associated minerals are pyrrhotite, pyrite, arsenopyrite, and marcasite in a fine-grained aggregate of lime-silicate minerals. Ankerite is found in large quantities. The average grade of the ore is \$6 per ton; the gold occurs native and in intimate association with arsenopyrite and pyrite.

In all development headings each round is sampled from the drill cuttings, and in addition a grab sample is taken. Hand samples of faces are not considered reliable because of the extremely spotty nature of the mineralization. Grab samples of the broken ore, which is very coarse, are not to be entirely relied upon. The appearance of the rock has considerable weight in distinguishing between ore and waste. Grab samples are taken from all cars loaded from a given chute in a 24-hour period. These samples are not accurate, but they are indicative of the ore being mined into each chute. The mill heads sample is the most accurate obtained and forms a basis for checking the appearance and assay value of ore taken from the chutes.

PORCUPINE DISTRICT, ONTARIO

In this district the ore occurs in lenses in sheared zones in Keewatin basaltic schists. In one large mine most of the ore occurred in brecciated zones in sediments, chiefly graywacke. The gold occurs free and intimately associated with sulphides, chiefly pyrite, and the quartz in the form of stringers and irregular bunches in the schist. The average grade of ore milled is about \$8 per ton. Spectacular

¹ Vanderburg, William A., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Inf. Circ. 6311, Bureau of Mines, 1930, 14 pp.

² Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Inf. Circ. 6402, Bureau of Mines, 1931, 11 pp.

bunches of high-grade are occasionally encountered in which the gold is very coarse; on the whole mineralization is very erratic.

Channel samples are relied on chiefly as a guide to development and stoping and for estimating ore reserves. The sampling is done by special sample crews in charge of a head sampler or the mine engineer.

Samples are cut with hammer and moil and are caught on a square of canvas. The channels are cut about 4 inches wide and one-fourth-inch deep, giving about 1 pound per linear foot of sample.

It is customary to sample each face in both development headings and in stopes at least once a day and at some mines after each round. The full width of the face is sampled taking in ore and waste, each band or change in material being sampled separately. The cuts are taken at right angles to the banding. The length of each such sample is measured and recorded in inches. (See fig. 17.)

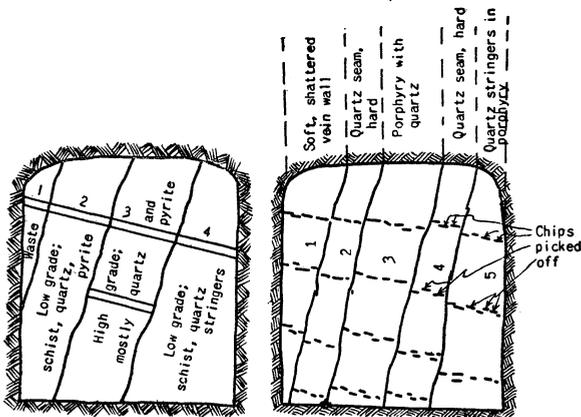


FIGURE 17.—Moiled channel samples in banded ore. Numbers indicate separate samples

FIGURE 18.—Pick sampling in banded ore at Wright-Hargreaves mine. Numbers indicate bands sampled separately

It is important to make the dividing line between samples correspond with changes in hardness of the material, and as these changes are usually quite sharp and abrupt this can be readily done. The quartz is very hard, while the schist is much softer.

It is virtually impossible to cut a uniform groove across a face in which there are bands of decidedly different hardness, more material being obtained from the soft sections, which may differ very materially in grade from the hard bands. By sampling bands of different hardness separately each sample can be cut of fairly uniform depth and width. Before the sample is cut the face is dressed of loose and projecting pieces and made fairly smooth and clean.

At one large mine several types of samples are taken, as follows:

1. Routine channel samples, which are channel samples taken after each blast as described above.

2. Special samples.

- (a) Check samples where the routine samples have shown the ore to be marginal grade (about \$4±). These are channel samples cut in the same manner as the routine samples, but taken across the backs of development headings. Back samples are usually taken at 5-foot intervals along the drift.

2. Special samples.—Continued.

(b) Rib samples, which are channel samples cut where the ore appears to be making into the wall of the drift or stope and are used as a guide in development and stoping. These samples are not used for ore-estimating purposes.

(c) Slash samples, similar to the regular face samples but they are taken across faces cut in widening or "silling out" the drifts.

3. Routine test-hole samples are taken by collecting the cuttings from holes drilled about 8 feet deep into the walls of drifts, raises, and stopes, cuttings from each 2 feet of hole usually being sacked as one sample. In drifting one test hole is usually drilled with every round, drilling to the right with one round and to the left with the next, thus alternating the samples on opposite sides. Where the ore has been lost, test holes are sometimes drilled up to 50 feet for exploration. Deeper holes are drilled with a diamond drill.

In raises and stopes the spacing of test holes depends upon conditions, such as whether it appears the ore is making into the walls, or if parallel ore shoots are suspected, etc.

4. Belt samples are taken along the sides of all crosscuts and of some drifts, individual samples being not over 4 feet long. These are channels cut with hammer andmoil and are caught on canvas in the usual manner.

5. Chute samples (grab samples) are taken as a rule only where a stope has been pulled empty and the walls are being "scaled" or trimmed down, to control the scaling work.

The total cost of channel sampling at this mine for labor and material is about 14 cents per linear foot of sample cut.

Accuracy of sampling.—The grade of ore as estimated from channel samples from development work is about 12 per cent to as much as 20 per cent higher than the average of the ore stoped, due to dilution with wall rock in stoping and to the fact that drifts are usually carried in the best ore, or an attempt is made to do so. Over the life of the mine, however, during which millions of tons have been mined, the channel sampling in the stopes has checked mill recovery plus tails within 2 cents per ton. These samples are taken across each face after blasting and include not only the ore but low-grade material and waste as broken, thus automatically adjusting the average value for overbreak of the vein where it occurs. This remarkably close check should not be taken to indicate that individual samples accurately measure the grade of ore broken by the corresponding round, but that the average value indicated by a large number of carefully cut samples is accurate even in spotty ore. Obviously where values jump from a few cents or dollars to hundreds of dollars per ton between adjacent samples, single samples are only of value as a guide to mining and only when taken in combination with many other samples are they useful for estimating ore reserves.

At a second large mine in this district the samples are of the following types.

1. Routine samples:

(a) Channel samples—

(1) Of each face after each round in both development and in stopes.

(2) Check samples of drift backs, generally at 10-foot intervals after drift has been slashed out to the full width of the ore body.

(b) Box or grab samples from muck pile after each round in stopes and development headings.

2. Special samples:

(a) Channel samples of faces where ore makes into the walls as a guide to development and stoping operations.

(b) Test holes into walls usually drilled 11 feet deep with standard rock drills.

Before channel samples are cut the face is cleaned, and the cut is marked off with a lamp flame. Samplers are instructed that where visible gold shows in the face it is not to be included in the sample, but that the sample is to be cut 6 inches above or below the visible gold. Samples are cut at right angles to the banding with hammer and moil and are caught on canvas. (See fig. 17.) Each band is sampled separately and its length measured and recorded in inches. In a drift 7 feet wide about 5 pounds of samples are obtained from one cut across the face. Samplers are paid \$4.80 per day and cut about 40 feet of channel, giving a labor cost of 12 cents per foot of sample cut. Sample assays from each stope, both channel and box samples, are recorded separately, and the channel samples frequently are checked against the box samples. Although there are usually rather wide discrepancies between box and channel samples from individual rounds or over short periods, box-sample assays average about 15 per cent higher than the channel samples. The channel samples are used as the basis for computing ore reserves and for this purpose are combined, after certain allowances are made for erratic high assays, as discussed later, on a dollar-foot basis. Over a year the ore-reserve estimate based on channel sampling is usually about 3 per cent low in grade as compared with actual mill recovery plus tailings loss. This close check is obtained by making certain experience allowances for erratic high assays, which are based to a considerable extent upon the judgment of the estimator in individual instances.

At another mine in this district ore in place is sampled (1) as a guide for current operations and (2) as a basis for making ore-reserve estimates.³

Only a very small proportion of the gold is visible, by far the greater proportion being finely disseminated throughout the mass and favoring the pyrite, the quartz, and the schist in the order named.

Diamond-drill samples.—Sludge samples are taken and assays made for each 10 feet of hole. These samples are usually not of much value but their cost is small in comparison with the cost of the drilling itself.

Diamond-drill cores are carefully examined and logged and are split for assay where at all promising in appearance. A single sample never represents more than 5 feet of core, and, if changes in the material traversed are apparent, samples may represent as little as 6 inches of core.

Development headings.—Crosscuts are rib sampled where any break or ore indication is traversed. Daily face samples are taken from all drift faces as they progress.

As drifting progresses the preliminary sampling is followed up by channel sampling of the backs at 5-foot intervals to delimit any commercial ore shoots, as well as to determine any favorable locations for exploratory raises.

Stops.—As stoping progresses, careful sampling of the walls is done when mining to an "assay wall." When the material is obviously of commercial grade enough breast samples are taken to serve as a rough check against the grade assigned to the ore block based on the back sampling of the drift.

Broken ore to mill.—As the ore is loaded at each working place (stope or drift), a grab sample consisting of a handful of fines is taken from each car and deposited in a box, and at the end of the shift is taken up for assay. While the method of taking these samples is very crude and day-to-day samples are subject to considerable variation, the results obtained are of much value in regulating the mill feed and indicating the grade of ore being currently obtained from various working places.

³ Dye, Robert E., *Mining Practice and Costs at the Vipond Mine, Timmins, Ontario, Canada*: Inf. Circ. 6525, Bureau of Mines, 1931, pp. 3-4.

The value of the ore milled in the past year (1930) as calculated from the muck sample assays was 4.8 per cent below actual mill recovery plus the tailings loss.

KIRKLAND LAKE DISTRICT, ONTARIO

Here the ore occurs in faulted zones and consists of sheared country rock in which the cracks have been filled with quartz, carbonates, and ore minerals and in which the country rock has been partly replaced. The metallic minerals occur in minute particles and are chiefly free gold; gold, silver, and lead tellurides; and small amounts of sulphides. The country rock is a complex of syenite, basic syenite, or lamprophyre and porphyry.

Gold is the valuable metal, and at depth the ore averages about 1 ounce of gold per ton, although after dilution in mining the material hoisted as ore assays \$10 to \$18 per ton, averaging about \$12.

At the Teck-Hughes mines⁴ development faces, drift backs, cross-cut walls and backs, and stope breasts are channel sampled, using hammer and moil. Samples are taken across the drift backs at regular 5-foot intervals, and these samples are used for estimating purposes. High-grade streaks and bands of different hardness are sampled separately, and the length of each sample is measured and recorded in inches. Bands of waste are sampled separately. Breasts in stopes are channel sampled in the same manner after each round is blasted for control purposes in stoping. In cutting down samples the samplers are instructed to discard any visible gold noticed. High erratic assays are rare.

Test holes are drilled to test the walls for parallel stringers of ore, and the sludges are collected and assayed in 2-foot sections.

In usual routine assaying the results are 3.5 per cent low. These are rapid assays and are low because of furnace losses due to volatilization and absorption and because of the assayer usually reporting the lower value of two check assays of the same sample.

In addition to the channel sampling every tenth car of ore drawn from the stopes or from development work in ore is grab sampled.

Over a large area the sample values are reported to average about 10 per cent lower than the actual value and to check the recovery of bullion closely.

At the Lake Shore mine all faces are channel sampled after each round is blasted, using hammer and moil, the samples being cut at right angles to the banding and each band of ore or waste being sampled separately. Usually the drifts are driven on a high-grade band with lower-grade ore on each side.

At the Wright-Hargreaves mine all faces are sampled after each round is blasted. A sample pick is used for sampling, picking off bits of ore across each band, the material from each band constituting a separate sample. (See fig. 18.) The width of each band is measured at right angles to its dip, and these widths are used for weighting the values when combining them in estimating the average grade at each face.

Development drifts are test holed to a depth of 6 feet by drilling one hole in the wall with each round, drilling alternately into the

⁴Henry, R. J., *Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.)*, Kirkland Lake, Ontario: Inf. Circ. 6322, Bureau of Mines, July, 1930, 10 pp.

right and left walls, and catching the cuttings in a clean sample sack. These samples are not used for estimating ore reserves but merely as a guide to development work. In stopes the walls are also test holed every few days to avoid missing any ore in the walls.

The method of combining assay values and the treatment of erratic high assays are discussed later. At this mine grab samples from the muck piles or cars run 30 to 40 per cent high. The method of pick sampling used has been found to give results that check very closely with results from regular channel samples.

At the Sylvanite mine each face is channel sampled after each blast. The ore bands are usually very narrow. Each band is sampled separately, two cuts being taken across each band, one near the top of the face and the other near the bottom. Later the backs are channel sampled at 5-foot intervals, and these samples are used for ore-reserve estimating purposes. Where the face exposes wall rocks on either or both sides of the ore seam the waste usually is not sampled; but its width is measured, and it is given a value of 50 cents, which has been proved to be the average value of the wall rock immediately adjacent to the ore seam.

ALASKA-JUNEAU MINE, JUNEAU, ALASKA

At this property the gold occurs chiefly, if not wholly, in quartz stringers and gash veins in slate and metagabbro. According to Bradley,⁵ the quartz is irregular in form and distribution, following the strike of the slate cleavage in a general way, but it has a slightly steeper angle of dip. Bradley states further, as follows:

Stringer lodes usually near the slate and gabbro contacts are found in a zone from 1,000 to 2,000 feet wide. These lodes are made up of a network of quartz veinlets and isolated lenses varying in width from less than 1 inch to 3 or 4 feet. The higher-grade ore bands are not over 300 feet wide, while the lower-grade material between them varies from 25 to 100 feet.

Clean quartz will average \$6 per ton within the areas of commercial ore while the rock outside of the quartz stringers is practically worthless. Outside the commercial ore zone there is an abundance of quartz carrying little or no gold.

The gold itself is erratically distributed and of wide variation in size. The size of gold ranges from nuggets with a maximum dimension of 0.75 inch, to the finest dust.

Associated minerals are galena, sphalerite, pyrrhotite, and pyrite. Galena and sphalerite are usually highly auriferous; the pyrrhotite will assay about \$6 per ton and the pyrite less. The run-of-mine ore averages about 90 cents a ton in gold.

Preliminary knowledge of the character, extent, and value of the ore deposits was gained by the early operations in the district, the gradual expansion of which had a most important bearing on present-day operations. This is true to the extent that each successive step in the progress was the result of the preceding step and the final operations of the Alaska-Juneau Gold Mining Co. were based on an accumulated fund of information rather than on a systematically developed tonnage, of which the assay value had been determined by any of the hand-sampling practices commonly in use.

Theoretically, it should be possible by hand sampling to determine the average assay value of any gold ore, but on the Juneau gold belt the gold is unusually coarse for lode gold, its distribution in the quartz is erratic, and the quartz itself is irregularly disposed throughout the ore bodies. Therefore, it is a difficult problem to determine in advance, by any system of hand sampling, the average gold content of these ores. Furthermore, the development

⁵ Bradley, P. R., *Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alaska: Inf. Circ. 6186, Bureau of Mines, 1929, 18 pp.*

work done in any one of the mines on the belt, in advance of milling operations, was not sufficient to permit a reliable determination of average assay values by hand-sampling methods exclusively.

The average gold assay value of the ore milled during the history of the Alaska-Juneau mine is 89 cents per ton, but many samples taken from within stoping areas exceed this average by several thousand per cent. Any practical method of hand sampling this ore will produce such a great variation of values.

Systematic channel sampling in the Alaska-Juneau mine is considered an unnecessary expense and is not now practiced. The value of ground not already known through actual mining is gained by grab samples taken from the muck during the progress of development work. The assay results of such samples are interpreted in light of experience and knowledge of the ground. The chief purpose of sampling in the Alaska-Juneau mines at all is to determine the grade of what is known to the miner's eye to be ore, and to make a permanent record of the information. The results obtained by this method of sampling, or any other method determine the high or low value of what the experienced eye already knows to be ore.

Final knowledge as to the value of the ore in the Alaska-Juneau mine has been derived from muck samples, moil samples, and mill returns; in addition to the mill returns from normal operations many tests have been made, not only on ore from various parts of the mine but also from bands intervening between ore bands and from waste. The conclusions arrived at have been supported by actual returns.

Bradley discussed sampling and estimation of ore at Alaska-Juneau in more detail in an earlier paper⁶ and includes some interesting data on recoveries as compared to average assay results. On page 106 he presents a table showing the weighted averages of moil and muck-pile samples as compared to mill recovery plus tails from two sections; the weighted averages taken from this table are repeated below.

Crosscut	Length of section, feet	Average assay of moil samples	Average assay of muck samples	Mill returns free gold plus tails
No. 1.....	571	\$0.499	\$1.657	\$1.012
No. 2.....	784	.943	1.577	.997

On page 104 of the same paper Bradley summarizes the experience with grab samples over a period of years at the Douglas Island mines, as follows:

	Treadwell mine (17 years)	700 mine (14 years)	Mexican mine (16 years)	Ready Bullion mine (15 years)
Average assay muck samples.....	\$2.37	\$2.20	\$2.88	\$2.72
Recovery plus tailings.....	2.47	2.33	2.83	2.24

MOTHER LODE, CALIFORNIA

Sampling methods in this district have been described by Arnot.⁷ Quoting from him—

Sampling in connection with mine operation is carried on at most of the mines in a desultory manner. Vein walls are usually clearly defined and the gold

⁶ Bradley, P. R., Mining Methods in Alaska-Juneau Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 103-109.

⁷ Arnot, Stanley L., Mining Methods in the Mother Lode District of California; Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 288-304.

content uniformly distributed through the ore body, hence there is little necessity for close sampling in the stopes as mining progresses. At a few mines sampling has been given serious consideration. All development work is sampled closely and the results recorded on assay plans. As the veins are often made up of several layers of quartz having different characteristics, samples of these layers are cut separately, and later averaged for the full vein width. At two of the mines all samples taken are referred to the foot wall which is called zero, as for example:

Feet	Character of material	Value per ton	Feet	Character of material	Value per ton
0-2	Ribbon quartz containing pyrite.....	\$20.00	8-10	Sugar quartz and gouge.....	\$1.00
2-4	White quartz containing little pyrite.....	6.00		Average, 0 to 10 feet.....	6.60
4-8	Slate with quartz stringers.....	3.00			

Satisfactory samples can be taken with a hand pick, as the quartz is seldom so hard as to require moiling. The maximum width for any one sample is 5 feet; the weight varies from 5 to 20 pounds. No quartering is attempted underground with the usual daily samples. At most of the mines the foreman takes the samples. Two mines employ samplers who keep geological data up to date in addition to performing their other duties.

HOMESTAKE MINE, LEAD, S. DAK.⁸

The ore is hard and tough. A sampling crew takes channel samples where development work is in ledge matter. Grab samples are taken from ore cars drawn from caved areas where there is a question as to whether the contents are ore or waste.

ZARUMA DISTRICT, ECUADOR

According to Emmel⁹ the veins are developed along faults, with filling of quartz, calcite, and subordinate amounts of iron and copper sulphides, sphalerite, and galena.

Close sampling is the practice wherever the vein is exposed. Channel samples are milled from the face and back one meter from each other, about 15 pounds being cut per meter. The samples may represent the whole width of the vein, or may be split as the character of the vein changes.

In the stopes the faces are frequently sampled under the direction of the mine superintendent for the purpose of directing daily work. Once every six months the entire stope surface is sampled and mapped just as the drifts are sampled, and tonnage estimates are made, using 2.9 tons per cubic meter. Because of the indefinite walls at many places the tonnage recovered is usually larger and the values lower than estimated from moil sampling.

LUCKY TIGER MINE, SONORA, MEXICO

The veins occur along fractures in volcanic rock. The ore is narrow (1.1 to 1.9 feet average) and the stoping averages 3.4 feet in width. The ore is a high-grade silver ore with some gold associated with sphalerite, galena, pyrite, chalcocopyrite, tetrahedrite, and stromeyerite in a gangue of kaolinized or silicified rhyolite. The av-

⁸ Ross, A. J. M., and Wayland, R. S., Mining Methods at the Homestake: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 424-445.

⁹ Emmel, Rudolph, Mining Methods in Zaruma District, Ecuador: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 447-464.

erage ore delivered to the concentrator carries 40 ounces of silver per ton, whereas the narrow ore seams, averaging 1.7 feet in width, usually carry 73 ounces per ton.

The mine and methods of sampling and estimating have been described in detail by Mishler and Budrow.¹⁰

The following notes on sampling are excerpts from this article:

In sampling development work, dilution in mining is accepted as unavoidable. Samples are cut over the full stoping width (3½ or 3 feet); if the vein is over stoping width the full vein is sampled.

Each sample across the back is divided into as many samples as there are varieties of ore and waste, and the exact width of each is noted. Where the ore is rich and easily recognized underground, it is believed that this method gives a more dependable average assay than is possible when a single sample is cut across both ore and waste.

All drifts, raises, and winzes are sampled at 5-foot intervals. The samples are cut from the backs of the drifts and from alternate sides of the raises and winzes. The backs of stopes are sampled in the same manner wherever the chute samples show that the ore is lower than milling grade.

In shrinkage stopes weekly samples are taken across the broken ore at 10-foot intervals. The width and distance from the end of the stope are recorded for each sample. The samples corresponding to the same distances are combined for the entire month and assayed. Grab samples, consisting of two double handfuls, are taken from each car of ore as it is loaded at the chute. Composites from important chutes are assayed daily. From chutes producing small tonnages and from development faces the samples are combined for periods of two or three days. The assays of chute samples are closely watched in order to maintain the grade of ore sent to the concentrator.

The following tabulation illustrates the accuracy of sampling. Odd-numbered samples were taken on one side of the drift and the even numbered on the other side. Samples were cut at 10-foot intervals.

Averages of alternate assays along Tigre vein

Level	Number of assays	Average of assays, ounces silver per ton			Error, per cent
		Odd numbers	Even numbers	Odd and even numbers	
7.....	428	35.4	41.4	38.4	7.8
8.....	368	34.4	31.9	33.1	3.6
9.....	288	36.0	42.3	39.2	8.0
10.....	232	32.0	39.7	35.9	10.6
Total.....	1,316	34.7	38.6	36.6	5.3

Grab sampling of cars shows an error of 9 per cent, the grab samples being high.

CRIPPLE CREEK DISTRICT, COLORADO

Jones¹¹ states that samples are taken by shift bosses. Streaks of ore are usually sampled together with a couple of grabs from the muck pile. Ore drawn from chutes is sampled by taking a handful from each car trammed. These grab samples will run 20 per cent higher in value than the settlement value of the ore.

¹⁰ Mishler, R. T., and Budrow, L. R., *Methods of Mining and Ore Estimation at Lucky Tiger Mine*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 468-483.

¹¹ Jones, Fred, *Mining Methods of the Cripple Creek District*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 512-517.

JARBIDGE DISTRICT, NEVADA

Park¹² has written of the sampling practice in this district as follows:

The wall rock is investigated by means of crosscuts and by holes drilled by ordinary drifting machines using extension bits. These holes may be quickly and cheaply drilled to depths of 50 or 60 feet and the sludge samples obtained show the presence of any values equally as well as the more expensive diamond-drill hole.

Drifts driven on the vein are sampled at intervals of from 3 to 5 feet. The samples are taken with a pick and average 10 or more pounds in weight. The face is sampled in sections, usually three or more samples being taken in one cut across the width of the drift, one sample representing the main vein material and the other stringers or wall rock included in the opening. A further sample is taken of the muck removed from each round; this serves as a rough check on samples cut from the face.

In this district the gold and silver are found mainly in the native state in quartz veins in volcanic flows. The veins are generally well defined. The chief gangue minerals are quartz and adularia. The veins are generally soft and often contain streaks of clay or gouge up to 6 or 8 inches thick.

MOGOLLON DISTRICT, NEW MEXICO

The gold-silver ore occurs in veins in igneous rocks in a gangue of quartz, calcite, wall-rock fragments, and some fluorite. Argentite and auriferous pyrite are the principal valuable minerals. In high-grade ore, native silver and free gold are often observed and frequently cerargyrite and bromyrite. According to Kidder¹³—

Underground samples are taken by trammers on each shift from every development face. Later, for purposes of preparing assay plans and estimating reserves, moil samples are cut, usually at intervals of 10 feet, and in spotted or high-grade ore, at intervals of 5 feet. In winzes and raises the samples are ordinarily cut at a difference of elevation of 5 feet, but on alternate sides of the raise or winze. Backs of stopes are cut at intervals of 10 feet, but on each successive slice the samples are cut halfway between those on the previous slice.

In addition to the sampling of development faces for estimating the mineral contents of ore bodies chute samples are taken, and the tonnage drawn from each chute is estimated by the number of cars trammed. The average assay in ounces of gold and silver is calculated for each chute at the end of the month. As the ore is trammed to the mill the cars are weighed, and a grab sample is taken from the top of each car and a composite sample made for each shift. At the end of the month the calculated average of the scale samples is checked against the chute samples taken underground and against the average mill heads as determined from the ounces of gold and silver in the bullion, concentrates, and tailings (production plus tails). On account of the slightly higher assay of the fines in the ore the average mine sample and the scale sample are usually slightly higher than the mill-head sample. The yearly average, however, seldom shows a variation of more than 10 per cent compared with the mill heads and generally agrees within 3 or 4 per cent. On spotty and high-grade ore the underground car samples are very unreliable. To be used at all, lots of ore after being sampled underground should be crushed and checked in a Snider, Vezin, or other automatic sampler.

¹² Park, John Furness, *Mining Methods of Jarbidge District*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 518-528.

¹³ Kidder, S. J., *Mining Methods in Mogollon District, New Mexico*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 529-549.

Kidder gives the following comparison between mine and mill sampling on two lots of ore.

	I	II
Weight of ore.....tons..	51,862	43,993
Mill average.....	\$11.084	\$10.254
Scale sample.....	\$11.218	\$11.193
Per cent above or below mill sample.....	1.20	9.17

TELLURIDE DISTRICT, COLORADO

Bell¹⁴ has written as follows regarding this district.

The exploration work is done by crosscutting to the veins and drifting on them. Diamond drilling is used at the Smuggler and Humboldt mines to determine if veins exist, but it gives no reliable data as to the value of the vein.

The ore has quartz gangue carrying varying quantities of sulphides, mostly pyrite, galena, and sphalerite. The precious metals are free gold, native silver, freibergite, polybasite, proustite, pyrargyrite, and stephanite. There are also chalcopryrite, sphalerite, mispickel, magnetite, and stibnite. The gangue minerals, besides quartz, include sericite, biotite, chlorite, amphibolite, apatite, garnet, orthoclase, kaolinite, calcite, siderite, rhodochrosite, dolomite, fluorite, and barite.

Cut samples are taken systematically and averaged geometrically. * * * Chute samples are taken if mill heads fall below a predetermined point. If the veins are narrow, the assays are reduced to a mineable width. * * * At one mine it was found that all assays over \$20 should be eliminated in estimating the ore, at another when mining an ore containing coarse gold it is necessary to include all the high assays, and even then the recovery is more than estimated. When mining fine-grained gold ores and silver ores, all high samples should be eliminated unless supported by additional adjacent high samples. The sampling interval is usually 10 feet.

COPPER MINES

HUMBOLDT MINE, MORENCI, ARIZ.

The ore is of the disseminated type in monzonite porphyry. The ore body now being mined by block caving is in zones of fracturing and has no definite walls, the limit of mineable material being determined by assay. The only ore mineral of importance is chalcocite. According to Mosier and Sherman¹⁵ underground openings such as drifts and raises are sampled by cutting channels in the ore with hand hammer andmoil. Care is taken to cut channels perpendicular to the structure. As the porphyry is cut by numerous tiny veins the channels have no regular direction relative to the workings. Some parallel the floor about waist high; others are cut diagonally across the back. Each sample represents 10 feet of drift or raise. Diamond drilling is also employed and has been described in the chapter on diamond drilling.

RAY MINES, RAY, ARIZ.

The ore body is of the disseminated type and occurs mainly in quartz-sericite schist and to a lesser extent in granite porphyry. The

¹⁴ Bell, Charles N., Mining Methods of the Telluride District: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 550-564.

¹⁵ Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Inf. Circ. 6107, Bureau of Mines, 1929, 34 pp.

principal ore mineral is chalcocite. Pyrite, sericite, and quartz are abundant gangue minerals.

Long-hole drilling has been extensively employed for underground prospecting. The sample interval is 6 feet, the sludge from each 6 feet of hole being split at the collar of the hole by an automatic sampler. Channel or cut samples are taken from development headings, and grab samples are taken in the cut-off shrinkage stopes and in the development headings. Thomas¹⁶ has described the practice as follows:

In developing a block of ground for mining, as will later be explained in detail, cut-off shrinkage stopes are carried up to the capping on four sides of the block. Grab samples are taken at 10-foot vertical intervals from the broken ore across the width of the stope and for a length of 12½ feet along the stope. The original method of sampling stopes was to cut channels, but owing to the irregularity of the stopes it was very difficult to obtain satisfactory samples in this manner, and grab sampling as described above has been adopted. The assays of samples taken from a block during drawing are adjusted to the mine head which is obtained from an automatic mechanical sampler.

In development work, drifts and raises are sampled by both grab samples and cut samples. The grab samples are taken to secure a daily record of the various headings. The cut samples are used for permanent records and are placed on assay maps and sections of a scale 50 feet to the inch. Cut samples are taken in drifts at 12½-foot intervals on alternate sides of the drifts. Every care is exercised to obtain as uniform a sample as possible. In cases where the planes of schistosity are well defined the samples are taken at right angles to these planes.

MIAMI MINE, MIAMI, ARIZ.

MacLennan¹⁷ summarizes the sampling methods and the relative accuracy of each for the Miami copper mine, as follows:

With an ore calculated to yield approximately 12 pounds net copper per ton, a variation of over 0.1 per cent in grade was important, and sampling of the development openings became a major problem. The original churn-drill sampling was checked by (1) the usual channel samples which were cut across the direction of the major seams every 5 feet of drifts; (2) cuttings from dry-stoper drill holes drilled across the major seams at 2.5 feet intervals; (3) samples of broken ore taken as the cars were loaded when driving the drifts; (4) a few check samples of 6 to 8 tons of ore shot down from the back of the drifts over a length of 25 feet and carefully quartered down; (5) one 1,500-ton sample mined from a narrow shrinkage stope and put through the automatic sampler at the mill. This stope was sampled at the same time by the other methods. The conclusion reached was that the churn-drill samples were accurate, the stoper-drill samples the most accurate small sample for drifts, and the channel sampling averaged 13 per cent too high.

In an earlier paper Hensley¹⁸ says of the sampling:

Shafts, raises, and winzes are sampled by taking the cuttings from horizontal shallow channels 1 to 2 inches in depth by 4 to 5 inches in width cut at intervals of 5 feet. Drifts are sampled by horizontal channels on each side approximately 4 feet above the floor; the cuttings from 10 feet of channel, 5 feet on each side of the drift are combined and represent the samples for the 5 feet of drift. No great care is taken in cutting these sample channels, as experimental sampling indicates that no greater accuracy is obtained and considerably more time is required. No consideration is taken of the structure of the ground or the presence of visible enriched veinlets.

¹⁶ Thomas, Robert W., *Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.*: Inf. Circ. 6167, Bureau of Mines, 1929, 27 pp.

¹⁷ MacLennan, F. W., *Miami Copper Co. Method of Mining Low-Grade Ore Body*: Trans. Am. Inst. Min. and Met. Eng., Year Book, 1930, pp. 39-80.

¹⁸ Hensley, J. H., Jr., *Mining Methods of the Miami Copper Co.*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 78-99.

Hensley states further that in working up estimates the value as calculated from drift samples is reduced 10 per cent, this reduction being based on results from channel drift samples of approximately 600,000 tons of ore, which were extracted without dilution with waste rock by the top-slicing system of mining.

It is probable that channel samples are enriched due to the brittleness of chalcocite, the principal ore mineral, which causes it to break away too freely in proportion to the gangue.

LA COLORADA MINE, CANANEA, MEXICO

La Colorada mine ore body is in the form of an irregular pipe of large dimensions in a brecciated zone in quartz porphyry.¹⁹ The mineralization follows well-defined zones of brecciation which have approximately circular or elliptical outlines. Stope boundaries are always determined by sampling, as the copper minerals extend outward from the main ore body as a network of small veinlets. The ore minerals are chalcocite, bornite, chalcopyrite, tennantite, and tetrahedrite. Catron describes the underground sampling methods as follows:

Drifts and crosscuts are pick sampled at 5-foot intervals when in waste. When there is doubt whether it is waste or low-grade ore channel samples are taken. These are cut horizontally, sometimes with a moil and hammer, but usually with a compressed-air chipping hammer. Grab or pick samples are taken from the face only for current information. Pick samples for permanent records are taken from the raises. Samples for control of stoping are taken by a crew which works on a special shift, overlapping both day and night shifts. Assay results from these samples are painted on the timber or the rock wall of the stope on the day following the taking of the sample. This prevents mining waste or leaving good ore, as it is often impossible, especially in the Colorada, to distinguish between ore and waste. When stopes are finished or before fill is run into a mined-out floor, another crew takes pick samples along the walls for permanent record on the assay maps.

Mined ore is sampled before being dumped into pockets or bins at the shaft. At the Capote, for example, a grab sample is taken from each car as it reaches the shaft station. At the Colorada motor cars of ore are sampled before being dumped into the pockets at the shaft. The 1-ton cars hoisted in cages to the 450 level are sampled there. Small pieces of wood are marked with the stope numbers and placed on the cars before they are caged to show the source of the ore. The car samples, as well as the pick and channel samples from stopes and development work, are sent each day to the sample mill at the Capote mine. There they are crushed, dried, ground, and split, and the resulting small samples are sent to the assay office at Ronquillo. All mine samples are assayed by the permanganate method. By means of the methods of sampling in use, the number of cars and the tonnage and grade produced by each stope and by the mine is known for each day. These figures check the returns from the mill and smelter sample plants closely enough for purposes of control of production.

CAMPBELL MINE, WARREN, ARIZ.

The ore is sulphide ore consisting of chalcopyrite and bornite as the chief copper minerals, mixed with pyrite, silica, and limestone as the gangue. Along the limestone contact with porphyry and in fracture zones are found native copper, cuprite, and chalcocite with other

¹⁹ Catron, William, *Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico*: Inf. Circ. 6247, Bureau of Mines, 1930, 41 pp.

secondary minerals. Lavender²⁰ has briefly discussed the mine sampling as follows:

Samples are taken from all faces of both drifts and raises in mineralized ground by shift bosses after each round. These samples may be obtained by picking from the face, by catching the drillings from a drill hole, or by taking grabs from the muck pile or cars. In the event ore is encountered the same procedure is followed, except that the motorman or trammer handling the cars of ore samples each car from a working place and consolidates the samples. The samples so obtained are sent to the assay office daily and from the returns made the material is classified.

The same procedure is followed for stope samples. In doubtful areas an effort is made to obtain sufficient face and drilling samples to determine the character of the material before blasting. In case of uncertainty as to the results channel samples are taken under the supervision of the engineering department. The interval chosen is usually 5 feet, due care being taken to have the samples represent sections at approximately right angles to the bedding of the material in question.

PILARES MINE, SONORA, MEXICO

The Pilares ore body²¹ is in a large pipelike mass of brecciated igneous rock having nearly vertical walls and a roughly elliptical horizontal cross section. The only primary ore mineral of importance is chalcopyrite; enriched chalcocite ores were found on the upper levels. The more continuous ore bodies occur around the rim of the brecciated zone; but irregular, disconnected bodies occur in the core of the zone. The boundaries of the former are fairly sharp, but the latter are more apt to have "assay" boundaries.

In exploration and development drifts grab samples are taken from the broken ore after each round is blasted where mineralization is shown. Check sampling is done with dry-stoper drills by drilling inclined holes at 5-foot intervals along each side of the drift in the ore area as determined first by the grab samples. The cuttings from each drill hole are caught and assayed separately. In raises inclined holes are similarly drilled from each corner of the raise.

In stoping, control of grade is very important. Daily grab samples are taken from the broken ore piles, but if the assays appear doubtful or are low in grade check samples are taken with a dry stoper. Doubtful areas in the back of a stope are sampled in the same manner. The average grade of ore blocks is determined by averaging the dry-stoper assays. These averages have, for the past few years, checked very closely with the grade of ore delivered to the concentrator.

UNITED VERDE MINE, JEROME, ARIZ.

Methods of sampling and estimating ore at this property and the degree of accuracy obtained, have been discussed in considerable detail by Quayle.²² His description is quoted below.

Sampling.—Five samplers and a head sampler are employed underground. Faces are usually sampled either by chipping with a sample pick or, to a less extent, by cutting channels. The chips are taken in four lines across

²⁰ Lavender, H. M., *Mining Methods at the Campbell Mine of the Calumet & Arizona Mining Co., Warren, Ariz.*: Inf. Circ. 6289, Bureau of Mines, 1930, 18 pp.

²¹ Leland, Everard, *Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico*: Inf. Circ. 6307, Bureau of Mines, 1930, 34 pp.

²² Quayle, T. W., *Mining Methods and Practices at the United Verde Copper Mines, Jerome, Ariz.*: Inf. Circ. 6440, Bureau of Mines, 1931, 31 pp.

the face for the width of the sample, beginning at the back, each line being successively lower. The average size of a sample is about 4 pounds. Individual pieces of rock or ore are limited to 1 inch in diameter. A sample covers a width of 5 feet, this width being regularly adhered to both in drifts and stopes. Usually the height of the wall or face sampled is 7 feet. The average length of a stope round is 6 feet. In the stopes, samples are taken after each round, so that a sample represents a horizontal area of about 30 square feet or about 20 tons.

Sampling errors vary with the type of ore. In the massive sulphides, the chalcopyrite or other ore minerals are very uniformly distributed and the copper content is gradational from the center of the sulphide ore area or the schist contact to the margin of the stopes, where the ore becomes noncommercial. Consequently, in sulphide, the sampling error is very low—an average of 2 per cent above actual content.

In the schist and porphyry ore areas, the occurrence of the ore minerals is extremely erratic; and, though in large areas of schist and porphyry a gradation of copper content may be noted, more often than not it is very rapid and sometimes abrupt. Oftentimes, too, in vertical extent ore and waste may occur on alternate floors for several floors in succession. Naturally then, the sampling error is high and can be reduced only by taking more samples. Possibly, as the ore minerals stand out in the matrix of porphyry and black schist, the personal element enters into the sampling of this class of ore. This source of error, however, is constantly guarded against. The sampling error has varied from 8 per cent to 20 per cent in this class of ore.

Samples are also taken from the various ore bins on the 1,000 level or Hopewell haulage tunnel.

The stope samples are checked against the ore-bin samples, and the latter are checked against the smelter assays. In all cases when a discrepancy exists the stope samples are higher than the ore-bin samples, which in turn are higher than the smelter samples. For the year 1928 the average error for all classes of ore between the stopes and the smelter was 5 per cent.

More specifically the methods and requirements of, and the duties of a sampler underground and in the office are as follows: On entering a stope or heading the sampler determines the number of samples necessary and records the location of the advance (measured with a tape from known points) in his notebook; he then marks the sample numbers plainly with heavy white chalk on the working face. The marks are made heavy enough to remain on the walls until the sampler on the night shift returns. The sample tags are then filled in, showing the location of the sample with regard to a known point, after which the sacks are distributed, and the samples taken.

If slips, dikes, or schistosity are parallel to the face, the sample is taken across the back of the floor below, covering the same area. When the sampler can not get under the back, any dikes or slips occurring on the face are broken into by plugging to permit sampling of the ore. Where it is impossible to obtain a satisfactory sample by the above methods, a series of drill holes, 5 to 10 feet deep, is drilled in the face and the sludges caught and used as the sample.

The samplers are held responsible for ore missed because of areas not being properly marked. The faces or walls must be marked with very heavy chalk either "O. K." or "N. G." The marks are heavy enough to last several days under ordinary circumstances; if the ore is not mined in that time, the faces are re-marked.

The sampler is required to make trips to the shaft with samples at regular intervals throughout the shift. This permits the assay office to make determinations and to prepare the daily assay sheet before the night shift goes underground. Sample sheets with the daily stope-sample returns are supplied to the foreman and shift bosses of both the day and night shifts. This enables the night-shift bosses to mine according to the results of sampling on the preceding day shift and prevents the mining of much low-grade or questionable material.

Cooperation with the mining department is essential, and is secured very successfully. The samplers keep in close touch with the foremen and shift bosses whose ideas regarding the value of doubtful ore are very acceptable, and are acted upon immediately by the samplers; the sets or areas in question are carefully checked and rechecked, if necessary. Close contact with the geologists and their stope files enables the samplers to anticipate the general characteristics, peculiarities, and irregularities of the ore bodies.

The samplers keep two sets of assay maps, one in the general office for use by the superintendent and geologist, and the other in the foreman's office for the use of the division foreman and shift bosses. These books are carefully kept and show projected ore outlines, pillar and fence lines, chutes and raises, and geology. They are posted every day in order to be up to date at all times. Thus the progress of any stope may be followed by watching these maps.

Development work is similarly plotted and posted after every round.

Some stopes are mined by means of vertical holes, and large areas are shot down at one time. In such cases, back samples are taken ahead of the cut, and mining is governed accordingly.

Occasionally samples of broken ore in the stopes, car samples, or chute samples are necessary. It has been determined that for such sampling the proper size to choose, as far as possible, is a piece $3\frac{1}{2}$ inches in all dimensions. Several cars of ore of approximately 1 ton each were chosen from various localities and sorted by hand into different sizes; Minus 1 inch, plus 1 minus 3 inches, plus 3 minus 5 inches, plus 5 minus 7 inches, plus 7 minus 10 inches. Each size was weighed and assayed, and the average assay for each car determined. From the assays it was found that the plus 3 minus 5 inch size was nearest the average for the whole car, and the plus 1 minus 3 inch size second best. From this it was deduced that the proper size to choose must be slightly over 3 inches in diameter.

ENGELS MINE, PLUMAS COUNTY, CALIF.

This property has been described by Nelson.²³

The ore occurs in shear zones in diorite and altered diorite, the principal copper minerals being bornite and chalcopyrite with some secondary chalcocite in the upper levels.

In development work a pick sample is taken from the face and also a grab sample from the broken rock after each round is blasted. In stoping, the drill cuttings from two or three holes each day are caught in a powder box and sent to the assay office. The assays furnish a guide to the values in the rock being broken. A small grab sample is taken from each car when drawing ore from the chutes; this serves as a check on the grade of the ore from the stopes. The car samples usually run higher than the mill heads. No systematic sampling of ore reserves is done.

EIGHTY-FIVE MINES, VALEDON, N. MEX.²⁴

The methods of sampling have been described as follows:

Grab samples are taken from all development faces in mineralized ground under the direction of the shift bosses. They are taken either by the muckers from the pile at the face, or by the "grizzly man" at the shaft station. Stope samples also are taken by the latter from each car as it is dumped into the pocket at the shaft.

In addition to the sampling mentioned an engineer helper samples all development faces. If the ore encountered is marginal in value it is sampled by cutting channels across the back of the drift or raise with moil and hammer at 5-foot intervals. If the ore is high grade the channels are spaced at 10-foot intervals. The engineer helper also takes 50-pound grab samples of the muck pile whenever possible. Stope faces are sampled only when the values are marginal or to determine grades for the yearly ore estimate. Mine samples are generally 10 to 15 per cent higher than the smelter samples.

The ore occurs in mineralized fault fissures in granodiorite. The vein filling is massive quartz, country rock replaced by sulphides, and silicified altered quartz-seamed rock containing stringers of chalcopyrite and pyrite. The distribution of values in the primary ores is very uniform.

²³ Nelson, W. I., *Mining Methods and Costs at the Engels Mine, Plumas County, Calif.*: Inf. Circ. 6260, Bureau of Mines, 1930, 22 pp.

²⁴ Youtz, Ralph B., *Mining Methods at the Eighty-Five Mines, Calumet & Arizona Mining Co., Valedon, N. Mex.*: Inf. Circ. 6413, Bureau of Mines, 1931, p. 5.

BURRA-BURRA MINE, DUCKTOWN, TENN.

At Ducktown, Tenn., the ore occurs in the form of a dipping, massive sulphide ore body replacing a bed in a series of highly metamorphosed schists and graywacke. The ore is an ore of sulphur and copper, the principal ore minerals being pyrrhotite, pyrite, and chalcocopyrite in a gangue composed chiefly of lime silicates, quartz, and calcite, with which the sulphide minerals are intergrown.

According to McNaughton,²⁵ grab samples are taken twice a week in stopes and daily in development headings, and in basing the ore estimates on the results of assays therefrom together with those from diamond-drill samples estimates accurate enough for all practical purposes are obtained. It should be noted that the ore has to be graded quite closely for use in acid manufacture, and thus this method of sampling seems to be quite accurate as far as sulphur content is concerned at any rate.

MARY MINE, ISABELLA, TENN.

At the Mary mine, in the same district, ring samples are taken in all drifts and raises at 5-foot intervals along the course of the drift or raise.²⁶ These ring samples are taken on the ribs and backs of the workings by chipping small pieces every few inches around the circumference of the drift or raise and have been found to be as accurate as samples taken by channeling. A 15-pound sample is taken. A small grab sample is taken of each mine car loaded and this is deposited in an iron box or car at each tramming place. These samples are collected twice a week, and the tonnages and values of the ore extracted from each stope are calculated. A grab sample is taken once each week of all working faces as a guide or check in breaking ore.

MAGMA MINE, SUPERIOR, ARIZ.

The ore occurs in a fault fissure and is of two general types, one in which the predominating gangue is altered, partly silicified diabase or porphyry and one in which the predominating gangue is quartz. In the oxidized zone the important copper minerals are malachite, chrysocolla, chalcocite, and some bornite associated with pyrite. In the primary ore zone the principal ore minerals are bornite, chalcocopyrite, and pyrite.

Snow²⁷ describes the sampling methods as follows:

Groove samples are cut with hammer and moil in all crosscuts and drifts in the vein. Crosscuts are sampled waist-high and parallel to the floor. The length of each sample cut depends upon the ore but is in no case longer than 5 feet. The Magma vein has a decided banded structure consisting of alternating bands of high-grade ore, milling ore, and sometimes waste. Samples are cut to correspond to the different bands; one sample is taken to each variation of the banding. Drifts are sampled across the back at regular 5-foot intervals. If conditions warrant, such as a change of formation, samples are taken at shorter intervals.

²⁵ McNaughton, C. H., *Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.*: Inf. Circ. 6149, Bureau of Mines, 1929, 17 pp.

²⁶ Kegler, V. L., *Mining Methods of the Ducktown Chemical & Iron Co., Mary Mine, Isabella, Tenn.*: Inf. Circ. 6397, Bureau of Mines, 1931, 10 pp.

²⁷ Snow, Fred W., *Mining Methods and Costs at the Magma Mine, Superior, Ariz.*: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.

MICHIGAN COPPER DISTRICT

In this district, where the copper occurs in the native state in amygdaloid and conglomerate beds, it has never been possible to sample ore in place in a manner to give reliable results. The individual pieces of copper range in size all the way from small specks to large irregular sheets and masses weighing several tons. Attempts to sample underground have been made repeatedly, with results so misleading as to be almost a menace in some instances. Because the copper is in the metallic state samples can not be accurately reduced in the usual manner by alternately crushing finer and quartering. The copper particles will not be reduced in size with the gangue, and in quartering a piece of copper in the original sample will either go into the sample or be rejected in its entirety.

What is or is not ore is determined by visual inspection. The foreman and bosses make daily inspections of the faces and record their condition as to copper, its continuity, and degree of concentration. These notes, together with daily reports on the assay value of the mill products, serve as a guide to mining.

Long familiarity with the ore in a given lode makes it possible to judge closely the grade of ore in any face.

BUTTE DISTRICT, MONTANA

The method of taking and averaging samples is about the same throughout the district except in the case of one company.²⁸ Following are excerpts from this paper, which was written by a number of mine officials in the district. The mine sampling is usually done by men employed solely for this purpose under the supervision of the geological or the engineering department.

The deposits to be sampled present a wide variety of conditions. Some exposures present a hard mineralization of quartz and sulphides uniformly distributed from wall to wall, but many contain bands of altered and crushed granite carrying only scattered particles of metallic sulphide. Again a portion of the ore bodies consists of veinlets of chalcocite and enargite traversing granite in such close succession as to make the whole mass valuable as ore.

The samplers make daily visits to all drifts and raises and also take any samples in the stopes that the mine foreman may require. The ideal cut or channel sample across the face of the drift is impractical from an operating point of view. Too much time would be lost if the sampler laid out and chiseled a neat groove across the vein. The same principles apply, but the result is accomplished by the use of a sample pick. The sample is caught in a sack, which is held open by a wire ring with a handle fastened to it.

OLD DOMINION MINE, GLOBE, ARIZ.

According to Shoemaker,²⁹ sampling is done under direction of the geological department. All development faces are sampled with a prospector's pick at 5-foot intervals. Where the ore is sufficiently high grade as to leave no doubt that it can be profitably extracted this pick sampling is taken as final. If the material is marginal the original sampling is checked either by channel or drill samples.

²⁸ Transactions of the American Institute of Mining and Metallurgical Engineers, Mining Methods in the Butte District: Vol. 72, 1925, pp. 240-244.

²⁹ Shoemaker, A. H., Mining Methods at the Old Dominion Mine, Globe, Ariz.: Inf. Circ. 6237, Bureau of Mines, 1930, 21 pp.

LEAD MINES

SOUTHEAST MISSOURI DISTRICT

The lead ores of this district are of the disseminated type, occurring in flat dolomitic limestone beds. The ore mineral is galena, which is found in vugs and cavities, filling or lining the walls of joints and crevices, and as aggregates of cubes in channels and joints.

At No. 8 mine³⁰ no mine or stope sampling is done, but the ore is sampled and assayed at the concentrator separately from that of the other mines of the company. Extensive diamond drilling is done from surface, however, and estimates of tonnage and grade are based on the drill samples.

At the other mines similar practice is followed. Extensive sampling of stope backs and walls is conducted, however, by test-hole drilling with jack-hammer drills.³¹ For this purpose the drills are operated dry, and the cuttings are caught in a pan which fits around the drill. The cuttings from each 2 feet of hole constitute individual samples. Samples showing lead are assayed and the values recorded.

COEUR D'ALENE DISTRICT, IDAHO

HECLA AND STAR MINES³²

The main Hecla ore body occurs along a vertical shear zone in quartzite, and other ore bodies are parallel thereto or intersecting it. The walls are usually quite well defined. The Star ore bodies are similar in occurrence but generally have no definite walls. The principal ore mineral is galena, with some sphalerite, and carries values in silver. Frequent horizons of waste occur between ore stringers. All development work is watched; and when valuable mineral is encountered cut samples are taken at intervals, depending upon the grade and continuity of the ore. The position and width over which each sample is cut are recorded. Enough samples are taken in a face to show the value of the different classes or grades of material encountered in the face.

MORNING MINE³³

At the Morning mine development samples are taken every 10 feet across the vein. Assays are made for lead, zinc, and silver, and together with widths sampled are posted on sample maps. Experience has shown that between levels there is little change in the grade of the ore and that stope sampling would be an unnecessary expense.

TINTIC STANDARD MINE, TINTIC DISTRICT, UTAH

The principal ore bodies are in limestone on or near the contact with faulted faces of hard quartzite. The principal revenue of the

³⁰ Poston, Roy H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Inf. Circ. 6160, Bureau of Mines, 1929, 22 pp.

³¹ Jackson, C. F., Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Inf. Circ. 6170, Bureau of Mines, 1929, 21 pp.

³² Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Circ. 6232, Bureau of Mines, 1930, 21 pp.

³³ Wade, James W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Federal Mining & Smelting Co., Mullan, Idaho: Inf. Circ. 6238, Bureau of Mines, 1930, 13 pp.

mine is from lead-silver ores, having a quartz-barite gangue. According to Wade,³⁴ the following is a typical analysis of the lead-ore shipments:

Lead-----per cent--	25.04	Gold-----ounce--	0.039
Copper-----do----	.31	Silica-----per cent--	49.03
Silver-----ounces--	30.29	Iron-----do----	7.18

Occurring with the lead ore is a siliceous ore of the following average analysis:

Lead-----per cent--	4.85	Gold-----ounce--	0.037
Copper-----do----	.36	Silica-----per cent--	66.96
Silver-----ounces--	17.24	Iron-----do----	11.50

Practically all stopes produce three different types of ore besides waste, and each type is broken separately if possible; otherwise sorting must be done.

Wade discusses the sampling practice at this mine as follows:

All the material hoisted, both ore and waste, is sampled three different times; waste which is used for filling is sampled twice.

The first samples taken are face samples. These are taken by especially trained men and under direct supervision of the engineering department. A portion of the mine is assigned to a sampler; it is his duty to sample daily and to record the assays of every operating face in that portion of the mine. Samples are taken in drifts and raises and every square-set face in the stopes opposite both the cap and the girt. These samples are grooves made with a hand pick at right angles to the dip of the vein and weigh approximately 5 pounds. Where a face is made up of two or more streaks of different hardness or mineral content each streak is measured and sampled separately. These samples are sent to the assay office by 10 o'clock a. m., and the determination is completed by noon. Each shift boss is furnished a complete assay sheet, showing all the samples taken, and these sample sheets are used by him in determining the breaking and classifying of ore in his section of the mine. In addition, the assays are copied on small cardboard tags and nailed on the cap over the face from which they were taken. This enables the miner himself to have full knowledge of the ores in which he is working; a most important item when sorting is to be done. Assay maps of each floor in the stope, raise, or winze and continuations of drifts and crosscuts are brought up to date twice a week by the samplers. These assay maps contain the dates of mining and details of lacing and filling for each set along with the face assays. On this sheet colors are used to indicate the month in which ore was mined. Average values so obtained are checked against actual smelter returns. These records are used in the estimation of ore reserves and by the management for the control of metal shipments.

The second sampling is a hand-grab sample taken by the trammers as they draw the chutes. These are referred to as box samples, one of which is taken from each section of the stope and from each drift. Box samples act as a check on the face samples, and in this way each round of waste broken is resampled before a gob is entered.

The third sampling consists of taking a "composite" grab sample from the cars on the surface by the top carmen of each grade of ore and of the waste hoisted as a final check before the ore enters the freight cars.

The grab samples taken by the shift bosses and top carmen are used to determine the value of the ore produced that day. Reports made by the shift bosses of the number of cars from each section, which is checked by the top carmen's report of cars of each grade of ore, are used to estimate the tonnage for the day. These figures are used in calculating the daily gross earnings.

Ore reserves are clearly defined by structural limits and computed by weighted assays taken from the assay maps.

³⁴ Wade, James W., *Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah*: Inf. Circ. 6360, Bureau of Mines, 1930, 21 pp.

ZINC MINES

AMERICAN ZINC CO. MINE, MASCOT, TENN.

At Mascot the ore occurs as veinlets and seams of sphalerite with secondary dolomite in dolomitic limestones.³⁵ Explorator in advance of mining is by surface drilling, both churn and diamond drills being used. In churn drilling all the bailings from a 3-foot advance after ore is reached are sampled and assayed.

In drifts and raises in ore grab samples are taken after each round has been shot. An underground diamond drill is in use constantly. Underground sampling is confined to grab samples from development as above; to core from the diamond drill; and to cuttings from test holes by machine drills in roof of stopes or mill holes. Grades of working faces are estimated.

TRI-STATE DISTRICT

Both zinc and lead are mined in this district, but usually zinc predominates, and this district is therefore included under "Zinc ores."

In this district sampling is confined to samples obtained from churn-drill holes and cuttings from rock-drill holes, as previously described. Visual examination of the stope faces is used to control the stoping.

MIXED OR COMPLEX ORE MINES

PARK-UTAH MINE, PARK CITY, UTAH

With the exception of a small amount of bedded ore in limestone, all the ore found at the Park-Utah occurs in fissures.³⁶ The fissures in which the ore occurs show a series of parallel bands of sulphides with alternating bands of altered limestone across the width of the veins. Two classes of ore are mined: Siliceous silver ore and lead-zinc-silver ore. These two classes are not found in the same stopes. The former is largely oxidized and occurs in the upper levels. Hewitt writes as follows concerning the sampling:

At the Park-Utah mine the grade of the ore in any particular fissure is very uniform, so that sampling is a simple problem. Grab samples, which consist of several handfuls of ore from each car, are taken at the chutes and are combined into one general sample for each chute on each shift.

Before being dumped into the railroad cars for shipment to the smelter the ore is sampled automatically. The automatic sampler is described in the section on transportation. The samples thus taken are combined into one general sample for each daily shipment.

Very little sampling is done in the stopes because the physical characteristics of the ore are such that with a little practice it is not difficult to distinguish between the ore and waste by inspection.

BLACK ROCK MINE, BUTTE DISTRICT, MONTANA

The following statements are taken from McGilvra and Healy.³⁷ The ore occurs in veins in granite which is intensely altered in prox-

³⁵ Coy, Harley A., *Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.*: Inf. Circ. 6239, Bureau of Mines, 1930, 11 pp.

³⁶ Hewitt, E. A., *Mining Methods and Costs at the Park-Utah Mine, Park City, Utah*: Inf. Circ. 6290, Bureau of Mines, 1930, 18 pp.

³⁷ McGilvra, D. B., and Healy, A. J., *Methods of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Montana*: Inf. Circ. 6370, Bureau of Mines, 1930, 16 pp.

imity to the vein. The principal ore minerals are sphalerite and galena, in a gangue of altered granite, quartz, and pyrite. The vein structure presents a banded arrangement of ore and waste parallel to the wall.

All development headings are sampled after each round is blasted. Stope breasts are sampled at each alternate square set. Samples are taken with a hand pick. Structural irregularities of each breast sampled are considered in taking the samples. Each band of ore is sampled separately and the widths of the bands of ore and waste are measured.

PAGE MINE, PAGE, IDAHO ³⁸

At this mine the ore is a zinc-lead-silver ore occurring in fissures in quartzite. The ore minerals are galena and sphalerite associated with some siderite.

Where ore is encountered in development drifts and raises, cut samples are taken at 10-foot intervals, and assays are made for lead and zinc with occasional assays of composites for silver. The sample widths and assays are posted on sample maps. No regular sampling is done in stoping, as familiarity with the ore requires only inspection and careful watching by bosses.

Mine run of ore averages 8.5 per cent lead, 2.8 per cent zinc, and 2.6 ounces silver per ton.

GROUND HOG MINE, VANADIUM, N. MEX.

Richard ³⁹ writes of this property as follows:

The average grade of the mill ore is 6.5 ounces of silver per ton, 8 per cent lead, 3.5 per cent copper, and 15 per cent zinc. * * * Below the 400-foot level the mineralization is practically all primary, consisting of galena, chalcopryite, and sphalerite, with quartz and pyrite as gangue minerals. * * * Samples are cut by hammer andmoil in all drifts and crosscuts. Channels 3 inches wide by about 1 inch deep are cut across the vein at 5-foot intervals. In crosscuts the channels are cut about 3 feet above the floor; at every 5 feet a sample is taken. Raises are not sampled. * * * Close sampling is not necessary for operating purposes because the grade is easily estimated by eye by experienced men. Samples therefore are taken only from doubtful ore as it is encountered in drifting or stoping. Production samples are obtained daily. These are taken at the collar of the shaft and consist of a small shovelful from each car hoisted combined into one sample for each shift.

PECOS MINE, PECOS, N. MEX.

Matson and Hoag ⁴⁰ are authority for the following:

From that time (January, 1927) to January, 1930, the mine has produced 584,158 tons of ore averaging 16.06 per cent zinc, 3.73 per cent lead, 1.02 per cent copper, 3.39 ounces of silver, and 0.109 ounce gold per ton. * * * The ore bodies are found in the shear zone, replacing the schist. * * * The mineralization consists of a mixture of sphalerite, galena, chalcopryite, and pyrite, carrying appreciable values in gold and silver, associated with the products of metamorphism, such as talc, hornblende, mica, and chlorite schist.

³⁸ Berg, J. E., Mining Methods at the Page Mine of the Federal Mining & Smelting Co., Page, Idaho: Inf. Circ. 6372, Bureau of Mines, 1930, 8 pp.

³⁹ Richard, F. W., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.: Inf. Circ. 6377, Bureau of Mines, 1930, 13 pp.

⁴⁰ Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

All underground openings showing mineralization are sampled by cutting channels. This is usually done with a hammer and moil, although in hard ore a compressed-air hitch cutter may be used. Channels are cut normal to the schistosity where possible, and the length of channel included in individual samples is made to correspond with bands of high, medium, or low grade ore or bands of waste.

The grade of ore is generally easily estimated by inspection, and sampling is unnecessary for stoping purposes. Sampling is usually done along the boundary lines of each floor of a stope previous to filling. The tonnage produced by each stope is determined by survey and checked by the car tally, which first is corrected to agree with the tons milled. No record is kept of the grade of ore produced by individual stopes. This could be done only by taking grab samples of cars, which would probably be inaccurate with ore of this character; it is believed that all other purposes are better served by the samples taken from ore in place in the stopes.

IRON MINES

In sampling iron ores it is desired to obtain the percentages of a number of elements which will affect the behavior, use, and treatment of the ores in the blast furnace, and the quality and composition of the pig iron produced. Most samples are assayed for iron, phosphorus, silica, manganese, and alumina. Separate samples are usually taken for determining the moisture content. Other determinations frequently made are of magnesia, sulphur, and loss by ignition.

The methods of sampling and analysis of iron ores have been described in the third edition of *Methods of the Chemists of the United States Steel Corporation for the Sampling and Analysis of Iron and Manganese Ores*, published by the Carnegie Steel Co., bureau of instruction, Pittsburgh, Pa. Extracts from this work have been quoted in *The Iron Ores of Lake Superior*.⁴¹

The subject is discussed under the following captions:

Importance of Sampling, Conditions of Sampling, The General Principles of Sampling, Diamond-Drill Samples, Churn-Drill Samples, Test-Pit Samples, Sampling in Open-Pit Mines, Sampling in Underground Mines, Sampling Drifts and Crosscuts, Sampling from Tram Cars at Underground Mines, Sampling from Railroad Cars at the Mines, Soft Ores, Hard Ores, Collecting of Samples for Moisture at the Mines, Standard Method of Sampling Cargoes, Sampling for Moisture in the Cargoes, Sampling from Railroad Cars at the Furnace, General Procedure in Sampling Trainloads of Ore, and Samples for Moisture.

This report is concerned with sampling ore at the mines and more particularly with ore in place. Underground two types of samples are commonly taken: (1) Channel or groove samples in soft ores and pick samples in hard ore; and (2) grab samples.

In the iron ore mines of the Lake Superior district it is usual to cut samples at intervals of 5 to 10 feet in all development headings. In soft ore a groove $1\frac{1}{2}$ inches wide and 1 inch deep is cut across the ore face at right angles to the direction of the formation. It is customary to combine the samples from 25 feet of heading or raise into one sample for assay unless the formation has changed from ore to waste, or vice versa, or the grade of ore is very erratic within that distance. In hard ores the cutting of uniform grooves is expensive; and in such ore it is customary, instead of cutting regular grooves, to break off small pieces roughly equal in size with a pick or hammer from equally spaced points over the face.

⁴¹ Crowell and Murray, *The Iron Ores of Lake Superior*: The Penton Press Co., Cleveland, Ohio, 7th ed., 1930, pp. 44-54.

MONTREAL MINE, MONTREAL, WIS.

The methods employed at the Montreal mine⁴² have been described briefly and illustrated in detail by O. M. Schaus, who states that "standard methods of sampling and preparing samples tending to eliminate the judgment of the sampler are used on the property." Schaus further states as follows:

In the development of an ore body the ore is blocked out so that enough trench samples may be taken across the formation to allow accurate estimate of the analyses of the ore in place. Tonnage in dike ore bodies is calculated by regular prism formula.

The straight iron ores are graded on phosphorus and silica; the manganiferous ores on manganese and silica. The iron is usually acceptable as to grade in all ores if the silica is controlled.

The underground sampling for the control of phosphorus and manganese is done on the ore in place, with grab samples as a check.

The underground sampling for the control of silica is done by grab samples of the broken dirt in the working place.

Control samples are checked by tram-car samples.

Ore is stocked in winter according to underground control samples, but check samples of the stock pile are taken and located to assist in the grading in shipping season.

The ore is shipped to the lake dock in 50-ton cars. The tonnage of lake cargoes ranges to a maximum of 14,000 tons.

Grading is governed in shipping direct, and from stock pile by samples of railroad cars in 5-car lots.

Due to the variation in analyses of ore that must enter one grade, provision is made at the shipping dock to average the analyses. This is done by selection of the last railroad car to complete a 6-car loading pocket.

Samples are prepared and analyzed on the property. A 10,000-ton cargo is represented by 40 samples, weighing 2,400 pounds, taken at 4,800 points on railroad cars.

Figures 19 to 25 are reproduced from Schaus's paper. Figure 19, A, shows the standard scoop used for obtaining equal portions from each point in grab sampling and 19, B, a standard sampling line for measuring off and locating sample points on steel ore cars. This line is employed for spotting sample points on standard ore cars, as shown in Figure 20. A scoopful of ore is taken from the top of the car at each point indicated by the small circles. If the point falls on a lump a piece half the size of the scoop is broken off, or enough small pieces to fill the scoop. Figure 21 shows the points *S* at which

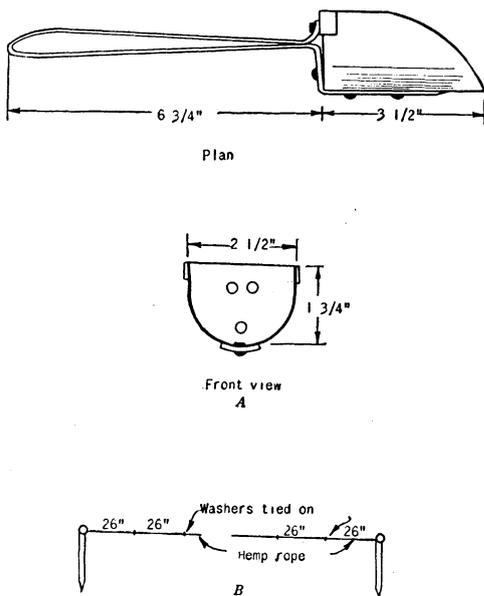


FIGURE 19.—A, Standard sampling scoop; B, standard sampling line for steel ore cars

⁴² Schaus, O. M., *Mining Methods and Costs at the Montreal Mine, Montreal, Wis.*: Inf. Circ. 6369, Bureau of Mines, 1930, 29 pp.

scoopfuls are taken for the sample on underground cars. Figure 22 shows the method of cutting groove samples on the side of the drift and in the breasts. Figure 23 shows the distribution of points for sampling muck piles in headings; Figure 24, sample points in stock-pile sampling; and Figure 25, the sample-crushing plant at the Montreal mine.

The use of a scoop and knotted or tagged rope for measuring off sample points on a pile of ore, whether in a car or muck pile, eliminates much of the usual "personal error" in taking grab samples without these accessories and makes for greater sampling accuracy.

MINE NO. 4, MARQUETTE RANGE, MICH.

At mine No. 4, Marquette range, standard methods of sampling as given by Crowell and Murray in the volume previously mentioned

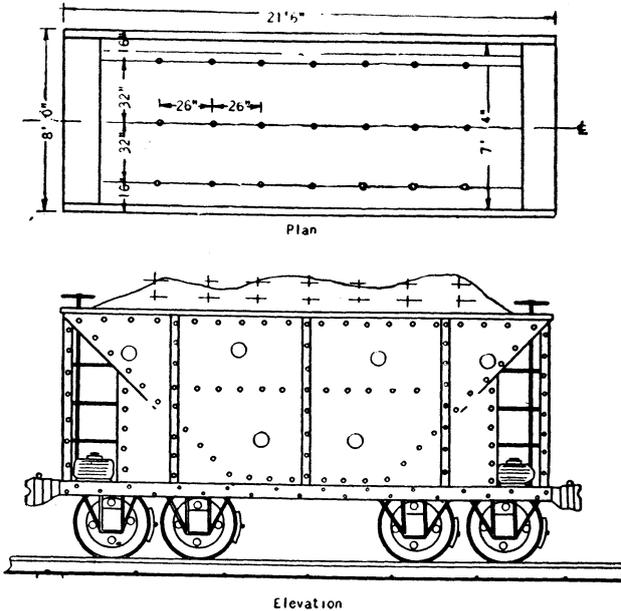


FIGURE 20.—Standard ore-car sampling

are used.⁴³ A sample from each skip hoisted determines the grade of the daily output. Underground samples are taken at the shaft from the tops of the cars, one scoop for the stope sample and one for the level sample. Where the analysis of a stope remains uniform the stope samples are only taken occasionally for a check, and only the level sample is taken daily. A standard scoop $1\frac{1}{2}$ by $2\frac{1}{4}$ by $3\frac{3}{4}$ inches is used and is filled from two places on the top of each 4-ton car.

⁴³ Graff, W. W., Mining Practices, Methods, and Costs at Mine No. 4 of the Marquette Range, Mich.: Inf. Circ. 6390, Bureau of Mines, 1930, 8 pp.

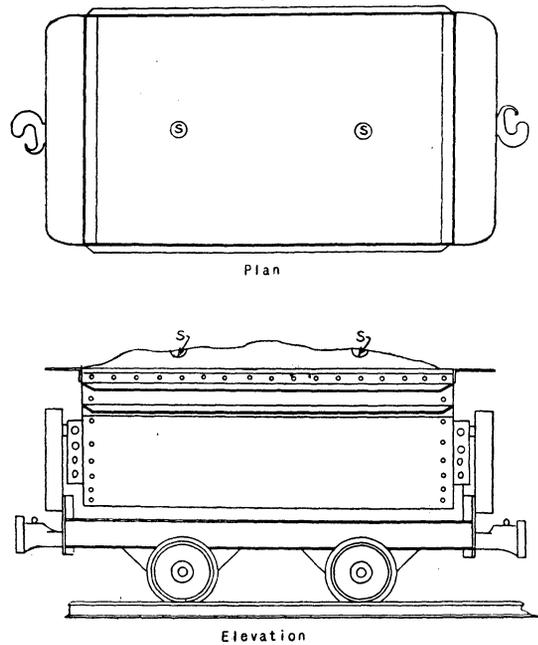


FIGURE 21.—Underground car sampling

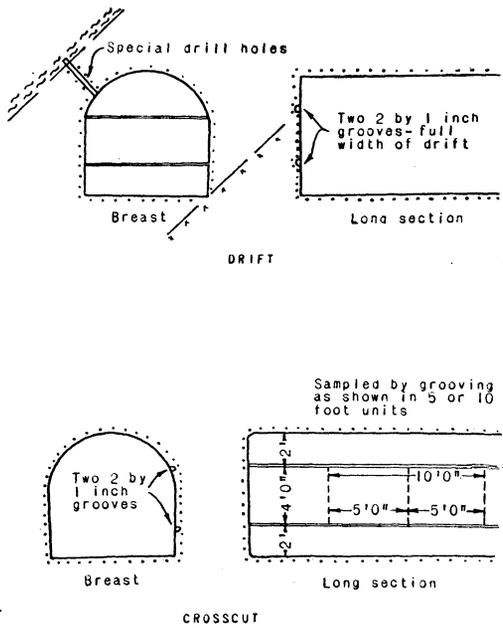


FIGURE 22.—Special underground sampling

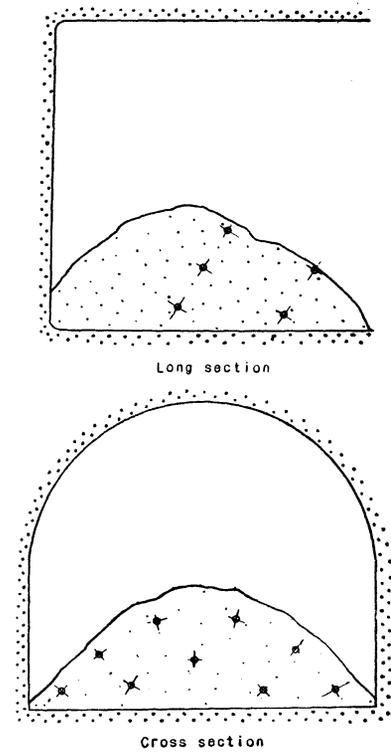


FIGURE 23.—Underground muck-pile sampling

MINE NO. 2, MARQUETTE RANGE, MICH.

At mine No. 2, Marquette range, prospecting and development drifts and raises are surveyed geologically, the formations are mapped, and when in ore are sampled at 5-foot intervals.⁴⁴

EUREKA-ASTEROID MINE, GOGEBIC RANGE, MICH.

At the Eureka-Asteroid mine the ore is distinct from the waste, and its iron content is very uniform throughout a given ore body.⁴⁵ The phosphorus content, however, which determines the grade of the ore, is sometimes quite variable; and when both Bessemer and non-Bessemer grades are being produced, the services of a good sampler are necessary. In development work the breast of each heading is sampled by the shift boss after each blast.

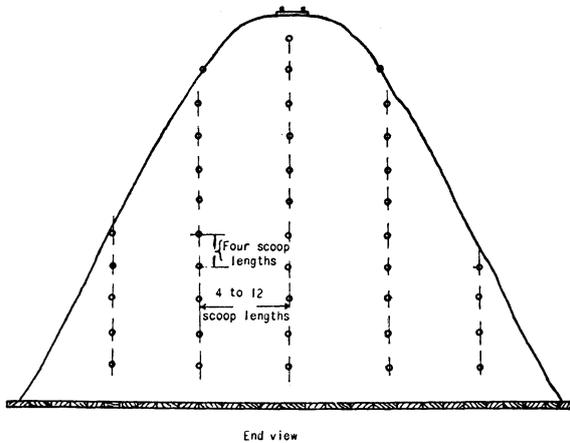


FIGURE 24.—Stock-pile sampling

HANOVER MINE, FIERRO, N. MEX.

The sampling methods at Fierro, N. Mex., have been described by Kniffin.⁴⁶

Kniffin states in part:

The accurate control of so many chemical elements—iron, sulphur, silica, phosphorus, copper, and manganese—requires careful sampling methods. The principle employed is to take the samples at points where a large quantity of the material is broken to as fine a size as possible.

Development samples are taken as the material is loaded into the cars. It is found that blasting in drifts and raises breaks the ore quite fine and gives a quite thorough mixture. This sample, representing the full cross section of the heading, is more valuable than the small area of a channel sample. When the face is made up of two or more classes of material it is necessary to sample each separately by channeling. The ore in most cases is very hard, and jack-hammer drills are found to be useful in cutting these channels. A sampler is employed in the mine on each shift to see that the samples are properly taken.

⁴⁴ Eaton, Lucien, Mining Soft Hematite at Mine No. 2 of the Marquette Range, Mich.: Inf. Circ. 6179, Bureau of Mines, 1929.

⁴⁵ Schaus, O. M., Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.: Inf. Circ. 6348, Bureau of Mines, 1930, 13 pp.

⁴⁶ Kniffin Lloyd M., Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Inf. Circ. 6361, 1930, 20 pp.

Stope samples are in general taken from a conveyor belt in the crusher plant after the ore has been broken to minus 5-inch size. Special samples are taken in the stopes as required. These are either chipped or obtained by drilling into the walls.

Finished products are sampled by automatic equipment in the concentrating plant.

Development samples are assayed and plotted on tracings of the mine plans and vertical sections, the latter being usually at 100-foot intervals though sometimes at 50-foot intervals.

GRAB SAMPLES AND BULK SAMPLES

GRAB SAMPLES

Grab samples are commonly considered very unscientific and inaccurate. As previously pointed out this is perhaps due largely to the manner in which grab samples are often taken. The same criticism might be applied with equal force to channel sampling where that method is employed carelessly or unsystematically.

The writers believe that with grab sampling as accurate results can be obtained as by channel sampling, provided equally as systematic and careful work is done. At the same time it is admitted that

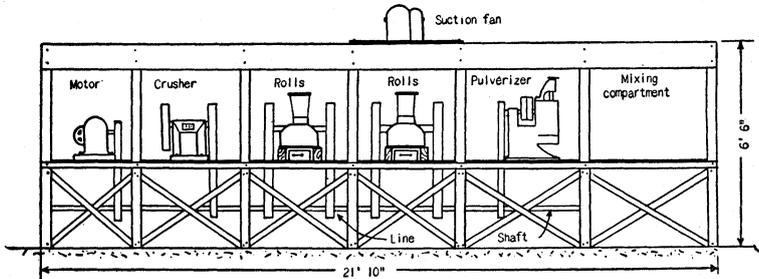


FIGURE 25.—Crushing plant

in many instances the expense and trouble required to obtain reliable grab samples are not warranted by the end to be gained thereby.

Channel sampling lends itself more readily to system. Thus, after a round is blasted the face usually remains in condition for sampling for at least a shift while drilling, timbering, etc., are going on and may be sampled by channeling or picking by a trained sampler without interfering with mining operations. Also channels may be cut from backs or sides of headings some distance back of the miners.

Obviously it would be impracticable to delay mucking at each place throughout a large mine until a sampler had visited it and grab sampled the top of the pile or to provide a sampler who would remain at each pile to take properly distributed samples throughout the entire mucking shift. In some instances grab samples from tops of cars may be accurate if carefully taken. In other instances the values might be concentrated in the fines, which usually shake down to the bottom of the car, to some extent at least, during haulage, if the distance is long. In order to obtain an accurate sample from the tops of the cars under these conditions they should be sampled at the loading chute. If these samples were taken by special samplers it would often require a large number of them to sample all cars

properly. Thus, car sampling is usually left to the loader, the trammer, or a man at the shaft who has other duties. It should be possible to train muckers, loaders, trammers, and men of that type to follow instructions for sampling to the letter, although every mine operator knows the difficulties to be overcome in enforcing strict adherence to regulations, especially where small details must be observed and the wrong way is decidedly easier than the right way.

Assuming that the end to be gained by accurate grab sampling is worth the cost and no cheaper methods of attaining that end are available most ores can be sampled as accurately by this method as by other methods.

To do this it is either necessary to have trained samplers who realize the importance of their work or rigid discipline of the regular workmen who take the samples.

The exact methods to be employed in any given instance will depend upon the degree of dissemination and size of the mineral particles and other characteristics of the ore and the methods of loading and handling. If the object is to get a daily check on the grade of ore broken the method may depend upon the mining system employed. Thus, in a shrinkage stope sampling of the ore as drawn from the chutes will not give the grade of ore broken on the preceding shift except when the stope is just being started. In this case grab sampling of the surface of the pile of broken ore would be required. On the other hand, if it were desired merely to ascertain the grade of ore produced by each stope during a period of, say, one year, the cars might be grab sampled as drawn from the chutes. In cut-and-fill stoping the grade of the daily break can usually be found in the same manner.

In any event, to take reliable grab samples it is almost always necessary—except for low-grade, uniformly mineralized ores—to be very systematic and to follow a set procedure which eliminates the human equation as much as possible. This may be done (1) by using a scoop or shovel with which practically the same amount of material is taken at each point from which material is removed for the sample and (2) by mechanically selecting the location of sample points by standard measurements or by taking material in equal amounts as loaded from one out of every so many shovelfuls loaded.

For taking equal amounts of material from each point a scoop such as is shown in Figure 19, *A*, may be employed. For mechanically selecting points of sampling, whether on a car or the surface of a muck pile, a knotted or tagged rope (fig. 19, *B*) may be used, as shown in Figure 20 (for car sampling) or as shown in Figure 23 (for muck-pile sampling). Figure 24 shows a method for mechanically marking off points for sampling a stock pile, using the scoop as a measuring stick. When these methods are used one scoopful of material is taken from each sample point if the material is fine. If the measured point falls on a chunk, enough pieces of the chunk are chipped off with a hammer at the point where the marker falls to fill the scoop. This is not always easy to do in practice, but with many ores can be done accurately enough for all practical purposes without undue trouble, and by so doing there is no discrimination by the sampler as between fine and lump ore.

The purposes of grab sampling have been stated in preceding pages under the descriptions of underground sampling methods at individual mines. Grab samples are often taken daily to determine the grade of ore from individual stopes and levels or from the entire mine, as well as the grade of ore in development headings. In other instances they are taken merely to obtain a rough check on the grade of ore from stopes, often where they are admittedly very inaccurate. Under such conditions one may question why they are taken at all. It would seem that where they are of value it would pay to use somewhat more care in taking them, using a mechanical method of marking off sample points on the cars and a scoop, as previously described.

Grab samples are sometimes taken as a check against moil and pick samples. Frequently they are taken from muck piles or cars where it is questionable whether the material is ore or waste, smelting or concentrating ore, for the purpose of quickly determining the disposition of the material on surface or underground. As a rule grab samples are not used as a basis for figuring ore reserves, although occasionally they are used in this manner.

Obviously, in examining an inactive mine or prospect, grab samples could seldom be taken, as there would ordinarily be no broken ore to sample; however, grab sampling might be used occasionally, as where broken ore has been left in shrinkage stopes or unshipped stock piles. Therefore, the examining engineer is usually forced to rely solely on cut or pick samples of ore in place.

The expense and care warranted in taking grab samples will depend upon the purpose for which the samples are being taken; as previously stated, it can usually be done so as to give reliable results, the question being whether the results are worth the means employed in specific instances.

BULK SAMPLES

Bulk samples are useful in checking the reliability of other types of samples and sometimes may be taken to determine a correction factor for use in estimates based on samples of other types. Such samples may be taken by blasting down drift backs or a section in a stope or otherwise obtaining a sample of several tons to several hundreds or even thousands of tons; the entire lot may then be milled separately and its mineral content determined thus or be reduced to a state of fineness such that it may be mechanically cut down and sampled to give a small sample that will accurately represent the entire tonnage.

Maclennan⁴⁷ has given an example of the use of bulk samples for checking the accuracy of small samples, which has been quoted in preceding pages.

Bulk samples are often taken to obtain material upon which to make experimental milling and metallurgical tests, and this is perhaps the most common use of samples of this type.

⁴⁷ Maclennan, F. W., work cited.

HANDLING AND TREATMENT OF SAMPLES

The handling and treatment of samples from drill holes have been discussed in previous sections, where numerous references were made to articles describing methods employed at various mines and under a variety of conditions.

Channel, pick, grab, and bulk samples from underground ore faces are usually sent to the assay office for crushing, mixing, and cutting down to the proper size for assaying. However, in sampling prospects or idle mines crushing and other assay office equipment may not be available close at hand. In this event it may not be convenient to ship out a large number of bulky samples, or the expense of doing so may be great so that it may be desirable for the sampler or examining engineer to reduce the size of the samples on the job. Therefore it is within the scope of this paper to discuss briefly the handling and cutting down of samples under conditions where hand work must be employed.

CUTTING DOWN SAMPLES

Cutting down samples by hand may mean considerable arduous work if accuracy is to be achieved because of the necessity of crushing and grinding the ore to a degree of fineness depending upon (*a*) the size of the sample; (*b*) the distribution, size, and number of the valuable mineral particles in the sample; (*c*) the difference in value of the high and low grade particles of mineral; and (*d*) the percentage of total weight of the sample comprised in valuable mineral.

Generally speaking, the larger the sample the more evenly distributed, smaller, and greater the number of mineral particles, and the less the difference in value between the high and low grade particles in the sample, the less grinding will have to be done. This refers not only to the original sample but to samples resulting from cutting down of the original at various stages of the operation.

QUARTERING

Large samples usually are first cut down by "quartering." Quartering consists of shoveling the sample into a conical pile, dropping each shovelful on the tip of the cone as it builds up. The fines will tend to remain at the tip or slide down the sides, while the lumps will roll down and accumulate around the bottom of the cone. Thus the fines will tend to become more or less segregated from the lumps, which is not conducive to thorough mixing and accurate splitting of the sample. In practice, the tip of the cone will tend to move in one direction or the other so that when the pile is completed the tip will be off center with reference to the center of the base. If this occurs there will be greater concentration of lumps on one side, and when quartering is done as described below the lumps and fines will not be proportionately divided between the sample and the reject. Since the value of the fines frequently differs considerably from that of the lumps an incorrect final sample will result. Some samplers employ an iron rod to mark the center of the cone, and this practice aids in reducing the error due to "drawing" the tip

of the cone off center. Others employ a wooden or steel cross having arms of equal length, which is laid on the floor where the ore is to be coned. Each shovelful is then dropped over the intersection of the cross arms.

For quartering, the cone is first flattened into a disk by dragging it out spirally with a shovel, beginning at the tip of the pile, taking care to spread the material uniformly in all directions. The finished disk is usually about one-tenth its diameter in thickness. The disk is divided into quadrants with a straightedge or a steel or wooden right-angle cross, and two opposite quadrants are shoveled and swept out clean and rejected. The other two quadrants are retained for a sample. If it is desired to reduce the size of the sample further, the operation of coning and quartering is repeated. Objections to this widely practiced method have been mentioned above.

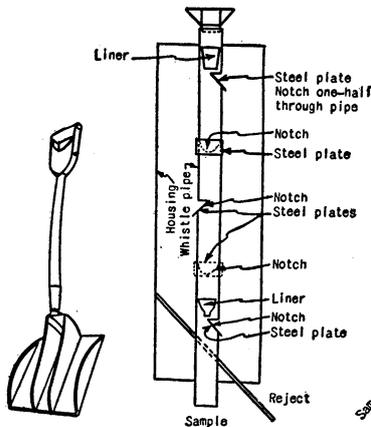


FIGURE 26.—
Brunton
quarter
shovel

FIGURE 27.—Whistle-
pipe sampler

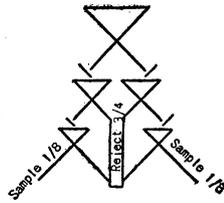


FIGURE 28.—Diagram
of bank or combina-
tion riffle sampler

BENCH OR COB SYSTEM

Another similar method sometimes used and termed the “bench” or “cob” system reduces the liability of segregation of fines; it consists in coning only a small portion of the sample, flattening this out into a thin disk, building another small cone on top of this, flattening, and repeating the operation until the entire sample has been coned and flattened. It is then quartered as usual.

MECHANICAL SPLITTING

Although coning and quartering are extensively employed for cutting down samples mechanical devices are much to be preferred. The simplest of these is the Brunton quarter shovel, Figure 26. For splitting the sample this shovel is forced into the pile until the compartments are filled. The shovel is then tilted back, emptying the reject and retaining the sample portion in the shovel. This is then emptied into the sample box and the operation repeated until the entire pile has been handled.

A somewhat similar device is the split shovel. A split shovel is constructed with alternate openings and prongs with turned-up edges to form long, narrow boxes such that the area of the boxes equals that of the openings between. This shovel is held over a box or other receptacle, and another workman empties a shovelful of the sample over the split shovel. Half the material is supposed to fall through into the receptacle, and the other half is caught by the prongs and discarded. The entire sample is put through the split shovel in this manner.

Another simple device is the "whistle pipe," Figure 27. It has the disadvantage that the entire sample must be crushed quite fine, as the final sample is only a small proportion of the original (one-thirty-second in the illustrated pipe).

A bank or combination riffle sampler like that illustrated diagrammatically in Figure 28 might be constructed at a remote mine and gives an automatically cut sample. Figure 29 shows a Jones splitter.

Moving automatic samplers are laboratory types and not within the scope of this paper.

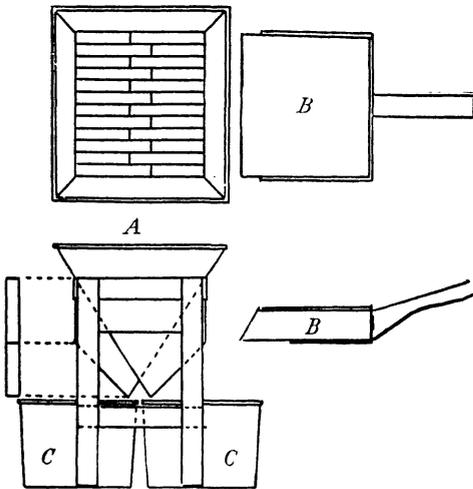


FIGURE 29.—Jones splitter: A, Splitter; B, shovel for spreading ore over riffles; C, C, pans for receiving split sample and reject

ROLLING AND QUARTERING

Small samples may be mixed by rolling on a sheet, spreading out into a disk, and then quartering.

Mixing by rolling is done by placing the sample on a square of smooth canvas or rubber, drawing one corner over toward the opposite corner, laying it back again, and then drawing the opposite one over in the same manner; the operation is repeated with the other pair of corners and so on until the sample is well mixed. The corners should be drawn over in such a way as to roll the sample, tumbling it over on itself rather than sliding it from one corner to the other, in which case the heavy and fine particles would tend to become segregated rather than mixed with the rest of the sample.

INACCURACY IN CUTTING DOWN

Whatever system of cutting down the sample is employed, the danger of inaccuracy due to inclusion of disproportionate amounts of valuable mineral particles, either in the reject or in the sample, increases as the sample becomes smaller. For this reason it is often necessary after each quartering to crush the sample finer before requartering. An exaggerated case would be where there is only

one particle of valuable mineral, which might either go into the discard, giving an assay value of zero to the sample, or into the sample, giving it a value twice, four, eight, or more times its correct one. The crushing or grinding should be fine enough at each stage so that the inclusion of a few more ore particles than the correct proportion in either the sample or reject will not affect the accuracy of the result appreciably.

This principle of relative fineness applies to all methods of cutting down samples, but takes on added importance where the particles of mineral are coarse or irregularly distributed in the gangue, differ greatly from each other in value, or make up only a very small part of the total weight of the sample.

No set rule for determining the fineness required will cover all ores, but safety lies on the side of overgrinding rather than undergrinding before splitting.

This subject has been discussed by Woodbridge,⁴⁸ Brunton,⁴⁹ Rickard,⁵⁰ and others. Peele's *Mining Engineers' Handbook*⁵¹ contains a discussion of the subject.

ACCURACY OF SAMPLING METHODS

The accuracy of samples—that is, the degree to which they accurately represent the ore or other material sampled—is separate and distinct from the accuracy of estimates of grade and tonnage of ore reserves based upon the samples. Samples may quite accurately represent the material sampled; but in mining, the grade of ore delivered to the mill may be considerably lower than assays of the samples indicate, due to dilution with wall rock or to the deliberate inclusion of low-grade material found in stoping or drifting, which can be handled and treated more economically as ore than as waste. Moreover, in averaging the results of the samples certain assumptions as to the continuity of ore and grade which are necessarily approximations usually have to be made, and these also introduce errors not due to errors in the samples themselves. The immediate discussion is confined to the accuracy of sampling methods. Estimates of ore reserves will be discussed later.

Descriptions of sampling methods at a large number of mines have already been given; and in some instances the results of the sample assays were compared with the assays of the mill heads, ore as shipped, or bullion produced plus tailings (in the case of gold and silver mines).

It has been pointed out that except for ores in which the valuable minerals are disseminated very uniformly individual samples do not accurately represent more than a few tons at the most and indeed often represent only a few pounds. Nevertheless, by taking a multiplicity of samples, using care and methods adapted to the type of ore, the average of such samples can be a very reliable basis for deter-

⁴⁸ Woodbridge, T. R., *Ore-Sampling Conditions in the West*: Tech. Paper 86, Bureau of Mines, 1916, pp. 16–17.

⁴⁹ Brunton, David W., *Modern Practice of Ore Sampling*: Trans. Am. Inst. Min. Eng., vol. 40, 1909, pp. 567–596.

⁵⁰ Rickard, T. A., *The Sampling and Estimation of Ore in a Mine*: Eng. and Min. Jour., 1904, pp. 23–26.

⁵¹ Peele, Robert, *Mining Engineers' Handbook*: 2d ed., John Wiley & Sons, New York, 1927, pp. 1847–1853.

mining the average grade of large tonnages of ore in place. It can not be argued from the law of averages that improper methods or carelessness will result in compensation of errors in individual samples if a large number of samples is taken, for in that event the errors will usually be cumulative in one direction or the other rather than compensating.

Even in ores in which the valuable minerals are uniformly distributed and with the most careful sampling, samples may consistently run higher or lower than the ore. One frequent cause of this is the difference in hardness or friability between the ore and the gangue minerals. At a given property experience factors can often be determined which, when applied to the sample assays, will give a correct estimate of the grade of the ore where samples run consistently high or low.

SUMMARY OF UNDERGROUND SAMPLING PRACTICE

Table 7 summarizes data on underground sampling practice by various methods under different conditions. These data are taken from previous publications in some instances and from special communications in others. Data on accuracy of sampling are not as voluminous as desired, but such figures as are available are believed worthy of tabulation.

TABLE 7.—*Summary of underground sampling practice*

District or mine and State	Character of ore	Sampling method	Indicated accuracy of sampling
GOLD AND SILVER			
Argonaut, Calif.-----	Gold in quartz, fissure vein	Chute and muck samples; visual inspection	Not reliable as to grade or ore.
Spring Hill, Mont.-----	Gold with sulphides. Contact metamorphic deposit. \$6 average value.	Drill cuttings and grab samples	
Porcupine district, Ontario:			
Mine 1.-----	Gold in quartz and sulphides in lenses in schist.	Chiefly channel samples; some grab samples; test-hole drilling.	Average of many channel samples is accurate. Drift samples 12 to 20 per cent high. All channel samples 0.3 per cent high.
Mine 2.-----	do-----	Channel samples and grab samples. Average grade \$9.	Erratic high assays are reduced in computations resulting in error of plus or minus 3 per cent.
Mine 3.-----	do-----	Channel samples and grab samples from cars. Average grade \$8.	Day-to-day grab samples are erratic. Over a year average was 4.8 per cent below mill recovery plus tailings loss.
Kirkland Lake district, Ontario:			
Teck-Hughes-----	Gold in quartz, some tellurides and sulphides in fault zone in syenite porphyry and lamprophyre.	Channel samples; test-hole drilling in walls of stope and sides of drifts.	The average of channel samples usually about 10 per cent less than mill recovery plus tailings loss.
Wright-Hargreaves-----	Similar to above veins in syenite porphyry	Pick samples; each band in face sampled separately. Grab samples.	Pick samples as accurate as channel samples and check recovery plus tailings loss plus or minus 5 per cent. Grab samples 30 to 40 per cent high.
Alaska:			
Alaska-Juneau-----	Gold in ramifying quartz stringers in slate and metagabbro. Average value ore \$0.89. Quartz sometimes high grade.	Grab samples in development headings to determine high or low value of known ore. Moil samples.	Section 1.—Moil samples \$0.499, mill heads \$1.012, error -50.1 per cent. Muck samples \$1.657, error +63.7 per cent. Section 2.—Moil samples \$0.943, mill heads \$0.997, error -5.4 per cent. Muck samples \$1.577, error +58.2 per cent.
Treadwell mine-----	Gold with sulphides and quartz in dikes	Muck samples	Average 17 years, \$2.37, mill heads \$2.47; error -4 per cent.
700 mine-----	do-----	do-----	Average 14 years, \$2.20, mill heads \$2.33; error -5.6 per cent.
Mexican mine-----	do-----	do-----	Average 16 years, \$2.88, mill heads \$2.83; error +1.8 per cent.
Ready Bullion mine-----	do-----	do-----	Average 15 years, \$2.72, mill heads \$2.24; error +21.4 per cent.
Mother lode, California-----	Gold-quartz veins in slates and greenstone	Hand pick. 5 to 20 pound samples for sampling development headings.	Grab samples generally 20 per cent high.
Homestake mine, South Dakota-----	Gold-quartz replacements in folded dolomitic beds.	Channel samples in development work. Grab samples from cars.	
Cripple Creek district, Colorado-----	Gold tellurides in veins in tuffs, breccias, and granite.	Grab samples from chutes. Pick samples for stoping control at Cresson mine.	Grab samples generally 20 per cent high.
Jarbridge district, Nevada-----	Native gold and silver; quartz veins in volcanics.	Long holes for exploration. Pick samples in development headings, and grab samples.	

TABLE 7.—*Summary of underground sampling practice—Continued*

District or mine and State	Character of ore	Sampling method	Indicated accuracy of sampling
GOLD AND SILVER—continued			
Mogollon district, New Mexico.	Gold and silver with sulphides in quartz and calcite gangue.	Moil samples; grab samples from muck piles, chutes and cars and at mill.	Mine car samples 3 to 4 per cent high. Grab samples at chutes very unreliable on high-grade ore. Grab samples at mill sometimes 10 per cent high.
Telluride district, Colorado.	Gold ore with quartz and complex sulphides.	Diamond drilling for exploration. Channel sampling.	Erratic high assays must be reduced in channel sampling.
Cortez, Nevada.	Silver-bearing quartz and sulphides in fissures, bedding planes and dikes. Some oxidized ores.	Pick samples; chute samples for stoping control.	Pick samples not accurate for estimating grade of ore.
Zaruma district, Ecuador.	Gold with sulphides in quartz-calcite veins.	Channel samples.	Tonnage recovered larger and grade lower than estimates due to dilution.
Lucky Tiger, Sonora, Mexico.	Silver, gold, and sulphides in veins in rhyolite. Average grade about 36 ounces silver per ton.	Channel samples. Grab samples in stopes. Grab samples from chutes for controlling grade in mining.	In check sampling by channels average discrepancy was 5.3 per cent. Estimates over 14 years based on channel samples, 8.4 per cent low.
COPPER ORES			
Humboldt, Ariz.	Chalcocite disseminated in porphyry.	Channel sampling. Diamond drilling.	In diamond drilling core recovery 50 per cent. Core alone not representative.
Ray, Ariz.	Chalcocite disseminated in quartz-sericite schist.	Long holes for exploration, grab samples for stope control. Channel samples for record.	Churn-drill samples accurate. Channel samples 13 per cent high. Test holes most accurate small samples.
Miami, Ariz.	Chalcocite disseminated in porphyry.	Churn drilling, channel samples, test holes, grab samples. Bulk samples to check accuracy of other samples.	Car samples check closely enough for production control.
Cananea, Sonora, Mexico.	Sulphide replacement in porphyry.	Pick samples in waste development and walls of stopes. Channel samples in doubtful ground, grab samples from cars at station.	Channel samples to check uncertain ground, where accuracy is required.
Campbell, Bisbee, Ariz.	Sulphide replacement in limestone.	Pick samples, drill cuttings or grabs from muck piles or cars in all development. Channel samples.	Samples check mill heads closely.
Pilares, Mexico.	Chalcopyrite in brecciated intrusives.	Grab samples and drill cuttings in development and for stope control.	1. Samples 2 per cent high. 2. 8 to 20 per cent error. Average error all classes of ore in 1928 was 5 per cent. Car samples high.
United Verde, Arizona.	1. Uniform massive sulphide bodies; 2. erratic sulphides in schist and porphyry.	Pick and channel samples of all faces.	Mine samples 10 to 15 per cent higher than smelter samples.
Engels, California.	Copper sulphides in shear zones in diorite.	Pick and grab samples from stopes and development faces, drill cuttings and car grab samples.	Ore estimates based on grab samples accurate enough for practical purposes.
Eighty-Five mines, New Mexico.	Sulphides in siliceous vein; uniform ore.	Grab samples from all development faces and from cars; channel samples in development and marginal ore.	Pick samples as accurate as channel samples.
Burra-Burra, Ducktown, Tenn.	Massive sulphides replacing schists, hard ore.	Grab samples biweekly; diamond drilling and long-hole drilling.	
Mary, Ducktown, Tenn.	do.	Pick samples, grab samples from cars, muck pile grab samples once a week.	

Magma, Superior, Ariz.....	Oxidized and sulphide copper ores in altered diabase or porphyry and ores with quartz.	Channel samples.....	
Michigan copper district.....	Native copper in amygdaloid and conglomerate beds.	No underground sampling. Control by visual inspection.	Has been found impossible to sample these ores accurately underground.
Old Dominion, Globe, Ariz....	Mainly sulphide ores in limestone and quartzite.	Pick samples. Marginal material checked by channel or drill samples.	
Butte district, Montana.....	Sulphide ores in veins in granite. Quartz or crushed granite gangue.	Pick samples.....	Where ore is uniform in veins of good mining width error about 0.5 per cent. In spotty ores and small widths error is high.
LEAD ORES			
Southeast Missouri district....	Galena disseminated in limestone.....	Diamond-drill samples and test-hole drilling....	Quite accurate.
Coeur d'Alene district, Idaho:			
Hecla and Star.....	Lead-silver ore in shear zone in quartzite.....	Channel samples.....	
Morning.....	Lead, zinc, and silver ore in quartzite.....	Channel samples (in drifts only).....	
Tintic Standard, Utah.....	Lead-silver ore in limestone; 3 types of ore.....	Groove samples cut with pick from all faces for stope control and reserve estimates. Grab samples from chutes and from cars on surface.	
ZINC ORES			
Tri-State district.....	Zinc and lead sulphides in flint and cherty limestone beds.	Churn drill and test-hole samples.....	Results are usually lower than actual grade.
COMPLEX ORES			
Park-Utah, Utah.....	1. Siliceous silver ore. 2. Lead-zinc-silver ore in altered limestone, uniform mineralization.	Grab samples from cars.....	
Black Rock, Montana.....	Sphalerite and galena in altered granite with quartz and pyrite.	Pick samples in development headings and stopes.	
Page, Idaho.....	Zinc-lead-silver sulphide ore in quartzite.....	Channel samples of all ore faces in development.	
Ground Hog, New Mexico.....	Galena, chalcopyrite, sphalerite with quartz and pyrite.	Channel samples in all drifts and crosscuts.	
Pecos, N. Mex.....	Zinc, lead, copper, silver and gold ore in shear zone in schist.	Grab samples from cars.	
IRON ORES			
Lake Superior district.....	Hematite and limonite ore bodies of different types.	Channel samples; churn and diamond-drill samples; grab samples.....	High degree of accuracy obtainable if sufficient care is used.
Fierro, N. Mex.....	Magnetite replacing limestone beds.....	Grab samples from cars. Channel samples where face contains more than one class of ore.	

Part 3.—ESTIMATION OF TONNAGE AND VALUE OF ORE DEPOSITS

PURPOSE OF ESTIMATES

Estimates of the tonnage and average grade of ore bodies are made for a number of different purposes. These include estimates in connection with examinations of prospects or of partly or well developed operating mines. Such examinations may be conducted on behalf of vendors or of prospective purchasers as a basis for placing a valuation on the property. Sometimes the estimates are used as a basis for computing valuations for exchanges of stock in connection with merger plans or may be employed in connection with litigation.

Operating companies usually estimate ore reserves at regular intervals. The results may be employed in the control of stoping operations and often are absolutely necessary where more than one grade or class of ore is mined in the same mine and where each class must be kept separate. Periodic estimates may also be made for accounting purposes, setting up assets, and figuring depletion, depreciation, and deferred development charges. Such estimates may also be made in connection with valuation for tax purposes.

BASIS OF ESTIMATES

Estimates of ore reserves are based upon the results of exploration and development and the samples therefrom.

Unless a deposit is fully developed, and even then to a lesser extent, certain assumptions have to be made regarding the continuity and grade of ore between exposed and sampled openings and extensions of ore beyond exposed ore faces. In making these assumptions the engineer must interpret the data available, and the accuracy of the final results will depend to a large extent upon his experience and the soundness of his judgment. Personal honesty and integrity have, of course, prime importance.

Not only must the known facts regarding drilling and underground development openings and the assay values of samples be interpreted and grouped properly but forecasts based upon geologic data must often be made. In addition, the mining method must be considered in the final calculations, since it may influence the percentage recovery of the ore body, the dilution of ore as mined, and the cost of mining and of milling, which in turn may determine the grade of material which can properly be classed as ore.

It is thus apparent that the estimation of ore reserves is not a precise and accurate procedure. In some districts where long experience has been gained regarding the character and habit of the ore bodies, certain more or less arbitrary rules may be used without introducing serious error in estimating extensions of ore beyond exposed

faces. This is exemplified by the practice in the iron districts of Michigan and Minnesota, where both the mining companies and tax commissioners accept valuations in compiling which such rules are employed.

VALUE OF EXPERIENCE AND JUDGMENT

In other instances there may be little if any previous experience in a district, and the estimator must rely solely upon his judgment and experience with similar types of deposits elsewhere. Even in estimating large, uniform ore bodies, such as those of the disseminated type and the flat, bedded type, the simplest to estimate, serious mistakes of interpretation are possible. Thus, as pointed out by Joralemon,¹ vertical drill holes may cut steeply dipping enriched streaks or fractures in low-grade ore, giving the impression of high-grade ore when in reality such grade may not extend even a few feet to either side of the hole. In the same article Joralemon illustrates how a row of drill holes may appear to indicate two flat ore bodies separated by a band of lean material and capable of being mined separately and points out the error in this assumption which, while not affecting the total tonnage, would affect the grade of ore to be mined.

To avoid overestimating ore bodies and give some value to the possibilities of undeveloped ore in the mine engineers frequently resort to the expedient of setting up tonnages under three separate headings: "Developed ore," "Probable ore," and "Possible ore."

The judgment of the engineer must be relied upon in drawing the line between these classes of ore reserves, and here again the type of deposit and known geological information must be carefully considered. Developed ore is usually ore the continuity and grade of which have been proved on four or at least on three sides, experience factors or allowances frequently being introduced to allow for loss in mining, dilution with waste, etc.

In estimating the probable ore, more latitude is allowed for speculation; but even here the engineer usually limits himself within rather definite lines, such as to ore exposed on two sides and to certain conservative distances for extensions of ore beyond exposed faces.

In estimating possible ore more latitude may be taken in interpreting geological criteria and without using empirical rules.

Since the greatest value in a property may lie in the more speculative probable and possible ore classifications, estimates of these classes are often of utmost importance. They are not calculable by mere mathematical formulas, hence their reliability depends upon the experience, knowledge, and judgment of the engineer or geologist. In any event, since the tonnages and values are more speculative than for developed ore, it seems logical to set up tonnage estimates under the three separate captions to indicate the comparative degree of uncertainty in the figures for each group.

The uncertainty as to grade is usually greater than that as to tonnage. Exceptions to this occur, however, as, for example, in estimat-

¹ Joralemon, Ira B., Sampling and Estimating Disseminated Copper Deposits: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 615-617.

ing certain iron ore deposits in the Lake Superior district, where the grade of ore is often quite uniform and may be more certain than the boundaries, before development work is completed.

METHODS OF ESTIMATING

The methods employed in estimating reserves of ore vary with the type, size, shape, and dip of the ore deposits and with the methods by which they have been explored, developed, and sampled. Numerous illustrations are given or cited in the following pages of the practices employed at different mines and for ore deposits of different types.

ESTIMATES OF VOLUME

Although the details of procedure in estimating ore bodies vary considerably, there are three general methods in common use: (1) Cross-section method, (2) average depth and area method, and (3) the method of factors based upon past production.

CROSS-SECTION METHOD

In this method cross sections of the ore body are prepared on which are plotted the intersections or projections of mine workings and drill holes. The cross sections may be vertical, horizontal, or at right angles to the dip, usually parallel each other, and often are spaced equal distances apart. Frequently they are taken on lines of drill holes where exploration has been done by drilling from the surface. Estimates are commonly made from sections normal to the strike of the ore body. In preparing such sections it may also be necessary to make longitudinal sections, working one set against the other in interpreting the structure.

If the sections are quite similar in outline the average area, in square feet, multiplied by the distance between them, in feet, will give the volume, in cubic feet, closely enough for practical purposes:

$$H\left(\frac{A_1+A_2}{2}\right)=V$$

where A_1 and A_2 are the end areas of the block,
 H is the perpendicular distance between them, and
 V is the volume of block.

If there is a series of sections spaced equidistantly, this formula becomes

$$V=\frac{1}{2}H(A_1+2A_2+2A_3+\text{---}A_n),$$

the common "end-area" formula. This formula is developed from the more general prismoidal formula by assuming that the mid area (halfway between the sections) is one half the sum of the areas of the sections. Due to the usual irregularity of ore bodies this may never be actually the case but is often a close enough approximation for all practical purposes.

If the ore body is very irregular so that adjoining sections show marked dissimilarity in shape, the volume may be approximated more closely by using the prismoidal formula :

$$V = \frac{H}{6} (A_1 + 4M + A_2),$$

where V = the volume,

H the perpendicular distance between A_1 and A_2 ,

A_1 and A_2 the areas of the two sections, and

M the area of a parallel section halfway between A_1 and A_2 .

In practice the areas are usually measured by planimeter. M is determined by plotting a mid section interpolated from the corners of sections A_1 and A_2 and measuring the area thereof. It may be necessary to construct longitudinal sections at right angles to the cross sections in order to construct the mid section.

For very irregular ore bodies it is apparent that where actual data are available for making closely spaced sections the closer these sections are together, the more accurate will be the estimate.

Figure 30 shows two adjacent sections quite dissimilar in shape. By using the end area formula, $\frac{(A_1 + A_2)}{2} H$, the volume is 17,737,500 cubic feet. Using the prismoidal formula and an interpolated mid section M , the volume $\frac{H}{6} (A_1 + 4M + A_2) = 18,162,500$ cubic feet,

a difference of 425,000 cubic feet. In this particular instance the end area formula gives the smaller volume and in practice might be a safer figure to use. Knowledge of the geological structure may be used in interpolating the mid section, however, while with the end formula the calculation is arbitrary.

If a series of closely spaced sections based on actual data can be made the prismoidal formula may be used, making each alternate section an end area and the intervening sections mid areas.

Somewhat intricate calculations involving higher mathematics have sometimes been advocated for making tonnage estimates, but since the basic data usually are not exact and often speculative it is questionable whether such refinements in calculation are warranted.

AVERAGE DEPTH AND AREA METHOD

This method is frequently applied to the estimation of broad, flat deposits which have been explored from the surface by drills or test pits. It is fairly accurate where the horizontal dimensions of the ore body are large compared to the thickness. Estimates of tabular deposits (veins or beds) from longitudinal projections, by computing the volume in blocks bounded by drifts and raises from the areas of the blocks and average thickness as determined by sampling, is another application of the average depth and area method, in which only the thickness is determined by averaging. Unless the ore is of very uniform thickness, the method is not as accurate as the cross-section method.

For determining the average depth of large ore bodies it is sometimes desirable to prepare cross sections as well as plans, particularly if drill holes or development intersections are spaced irregularly. If the holes are spaced regularly, the average depth may be taken as the

arithmetic average of the depth of ore in all the holes. If the holes are spaced irregularly, the depth of ore in each hole must be weighted by a factor depending upon the relative area, governed thereby.

Where estimates are based upon drilling, weighting of depths as well as of assay grades may be done graphically by dividing the area into triangles or polygons, with the holes at the corners of the triangles or centers of the polygons. The triangle method is accurate where the holes are equally spaced and the triangles are equilateral,

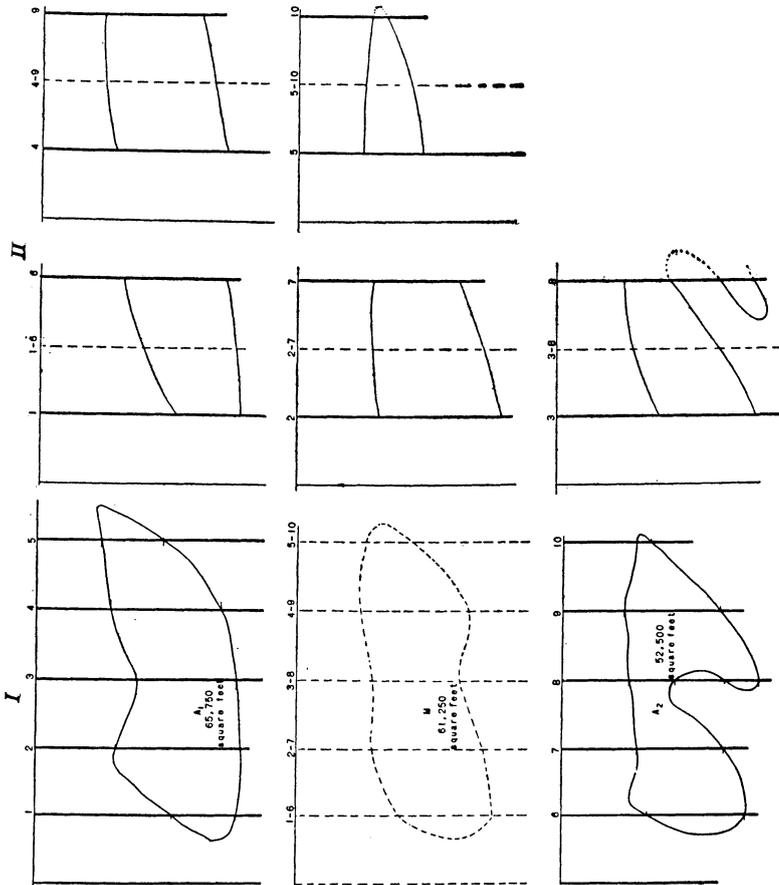


FIGURE 30.—Sections of irregular ore body used in computing volume of ore: I, Cross sections; II, longitudinal sections for interpolation

otherwise the polygon method is more accurate. See Figure 31.

ORE EXTENSIONS BEYOND EXPOSED FACES

In estimating the extension of ore beyond drill holes or development faces the geological conditions must be carefully considered. In some instances it may be known that the ore stops abruptly against a fault, dike, contact between formations, or other limiting geological structure, the position of which is known, and the volume between

actual ore exposures and these limits may quite confidently be calculated as ore. In other instances ore limits may be gradational, and it is not safe to project ore much beyond actual exposures. In yet other instances, even though limits of ore are not defined clearly by structural features, certain assumptions may be conservatively made on the basis of previous experience with the ore bodies in the same mine or district, and depending upon whether the ore was widening out or narrowing down in the direction of the latest development.

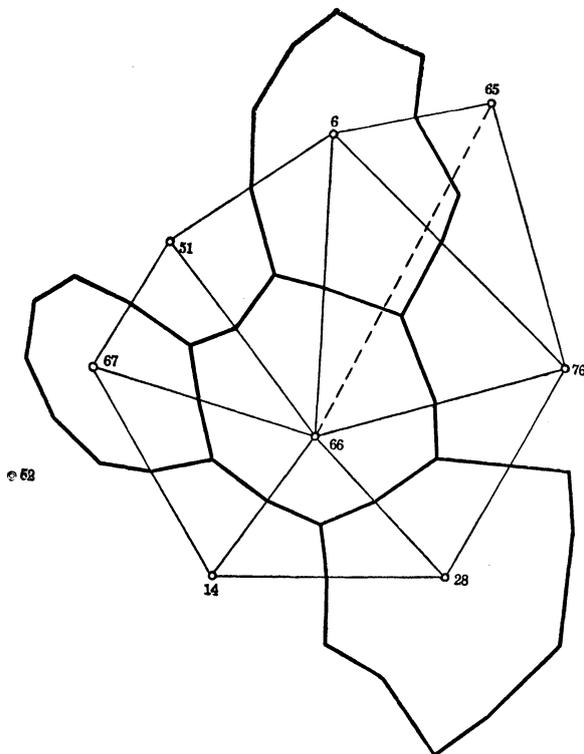


FIGURE 31.—Method of forming polygonal areas around surface diamond-drill holes

ESTIMATES OF TONNAGE

If the volume of ore within the explored area, of the probable and possible marginal ore, and of the ore below the lowest ore developed has been estimated, the next step is to figure the tonnage thereof.

The tonnage is derived from the volume by applying a cubic-feet-per-ton factor, which varies for different ores, depending upon the specific gravity and proportion of each of the constituent ore and gangue minerals, and the density (or conversely, the porosity) of the mineral complex.

TONNAGE FACTOR

The factor may be determined by calculation, if the specific gravity and percentage of each of the constituent minerals and the porosity

of the ore are known; by making direct specific-gravity tests of samples from various parts of the ore body; by actually weighing the ore extracted from an accurately measured excavation in the ore; or by applying an experience factor derived from engineers' measurements and actual shipments over a considerable period of time.

Since the specific gravity often changes appreciably from one section of the ore body to another often the factor will not be exact but can usually be determined within the degree of accuracy of the other calculations involved in the estimate. Variations in grade of ore, porosity, and composition of the gangue may cause variations in the weight per cubic foot. Curves or tables (figs. 32 and 33)

are sometimes employed for use in determining the cubic-foot-per-ton factors for different grades of ore.

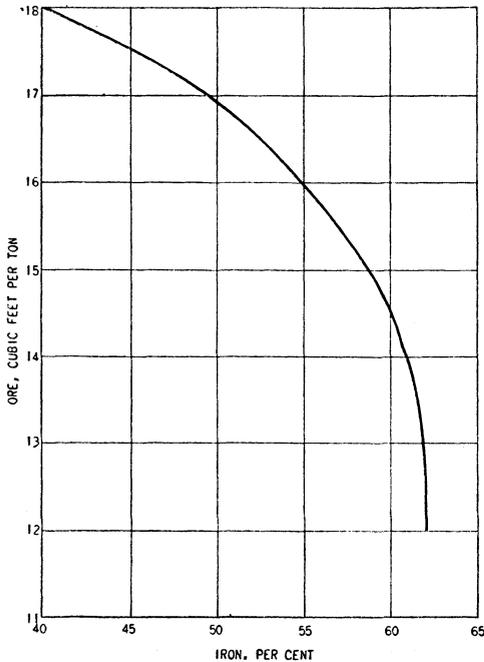


FIGURE 32.—Diagram for estimating tonnage of iron ore, applicable to certain ores of the Lake Superior district

MINING FACTOR

Finally the tonnage estimate may also require that a mining factor be applied to take care of losses in mining, dilution of ore from the walls of the deposit, or waste inclusions within the ore body. Thus the character of the wall rocks and their probable action under the mining method to be employed and the probable loss of ore in mining by that method must be considered in calculating the final figures. In some of the older districts enough mining experience has been gained for a reliable factor to be applied, allowing for these items. Thus in one district

an allowance of 10 per cent for mining loss is usually made with a factor of 10 per cent for waste inclusions. In another district it is necessary to allow 20 per cent for dilution by walls and capping, which increases the tonnage and reduces the grade accordingly.

CHECKING ESTIMATES

In attempting to check tonnage estimates against actual tonnage mined from a given block there is usually an uncertainty not only in the estimate but in the figures of actual production. Variations in the specific gravity of the ore and irregularities in the thickness of the ore body, together with the dilution factor in actual stoping operations, all affect the accuracy of the estimate. Tonnage mined from a given block usually is based either upon the number of cars

drawn from a stope or upon engineers' estimates based on measurements of the excavation. In only a few ore mines are the mine cars weighed underground. Usually a fairly close approximation of the weight of ore per car is made and is used in all calculations. More often than not a safety factor is applied, so that actual weights are slightly more than the figure used in estimating. Where tons mined are figured on engineers' measurements irregularities in the outlines of the ore and excavations which can not actually be measured, due to the mining method (particularly with shrinkage stoping), and inexactness of the cubic-feet-per-ton factor all tend to make the results estimates rather than exact figures.

Likewise in attempting to check estimated grade against actual grade mined, as determined by mill recovery plus tailings loss, inaccuracies in the estimates of actual tonnage mined obscure the actual grade of ore mined.

COMPUTATION OF GRADE OF ORE

Having computed the volume and tonnage, the next step is to estimate the grade of ore. Here, too, the estimator is confronted with the necessity of making certain assumptions as to continuity or regularity of grade between exposed faces and continuance thereof beyond these exposures.

Where the mineralization and grade are erratic individual samples may vary greatly from each other and often do not indicate the true grade of ore for distances of a few feet, or even inches, on either side as has been previously pointed out. By taking a large number of samples and using methods suited to the type of ore, errors are usually compensating, and the average grade estimated may closely approximate actual grade.

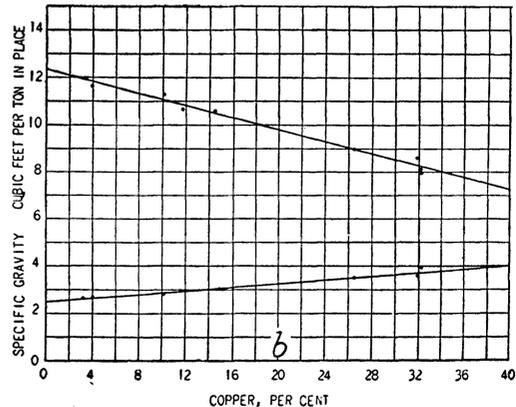
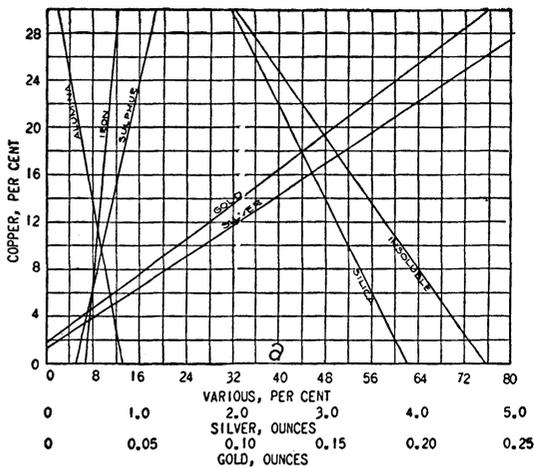


FIGURE 33.—a, Variation of values and analysis of La Colorado ore with copper content; b, La Colorado ore: Specific gravity and cubic feet per ton in place with varying per cent of copper

VEIN DEPOSITS

In developing vein deposits wider than drift width it is common practice to endeavor to drift and raise in the best portion of the vein, provided the character of ore and walls is such that the levels can be easily maintained in this position in the ore bodies. Where this is the practice it is evident that drift samples will give assays higher than the average grade of ore to be mined. Thus, in some gold mines with which the writer is familiar the regular drift samples are employed only for guidance in development and for assistance in keeping development in the best portion of the vein, and the assays employed in tonnage estimates are those of samples taken across the back of the level after it has been silled out to the full width of the vein.

On the other hand, where the vein is narrower than the drift or than stoping width assays generally must be reduced to cover the inclusion of enough wall rock to provide a working width. A possible exception might be where ore and wall rock are easily distinguishable from each other and can be sorted in the stope or where resuing or stripping can be practiced. Even then, however, allowance would have to be made for either loss of ore in the fines or dilution with waste, depending upon the mining method used.

LARGE LOW-GRADE DEPOSITS

In large, uniformly mineralized low-grade deposits of the disseminated type there is less variation between individual samples than in the vein or replacement type of deposit, and on the whole it may be said that estimates are less liable to error. In deposits of this type, however, the margin of profit per ton of ore is usually small under normal market conditions, and failure to estimate to within a small fraction of a per cent may have serious consequences. Thus in any event the estimation of grade of ore is important and requires care and the best of judgment.

GRADE BEYOND EXPOSED FACES

It has been pointed out that the reliable estimation of tonnages of ore beyond actual exposures is a matter of judgment based upon experience and is attended by risk.

For many types of ore deposits it is even more risky to attempt to estimate the grade of ore beyond actual exposures. On the Gogebic range in Michigan the ore is found in pitching troughs formed by the intersection of dikes with footwall quartzites or with impervious slate members in the iron formation. Here it may usually be assumed that the ore above the actual intersection will continue downward to the intersection, and the grade of the ore may be predicted with some confidence. In most other types of deposits, however, it is one thing to say that ore will continue a certain distance beyond an exposed face and quite another to predict its grade.

GROUPING OF ASSAYS

In estimating the grade of developed ore it is usual to group the assay values in blocks, or by sections, weighting each assay in the block or section according to the length of the sample and the distance between it and adjacent samples to get the average grade. The method of grouping the assays will depend upon the type of deposit, method of mining, and method of sampling. Numerous examples of practice in this respect are given in the following pages, illustrating methods used for different types of deposits and methods of mining and sampling. These examples are not cited as being mathematically correct in all instances; but as most of them are taken from operating mines, it is assumed that the methods employed give results which satisfy practical requirements.

The formula for arriving at an average value for a series of samples is

$$G = \frac{G_1L_1 + G_2L_2 + G_3L_3 + \dots + G_nL_n}{L_1 + L_2 + L_3 + \dots + L_n}$$

where

G = average grade,

$G_1, G_2, G_3, \dots, G_n$ = value or grade of each sample, and

$L_1, L_2, L_3, \dots, L_n$ = length of the corresponding sample.

This simple formula applies to the averaging of a number of samples before the introduction of weighting by area or volume, where the samples are spaced equidistantly. The feet-per cent (or dollar-feet for gold and ounce-feet for silver) of individual samples are added and divided by the total feet of samples to give the average grade.

WEIGHTING OF ASSAYS

If samples or drill holes are spaced at irregular intervals care must be taken to give the proper weight to each, particularly where values are spotty and irregular and there is an appreciable difference in grade between the samples. Each sample should be given a weight corresponding to the length, area, or volume it represents.

Even where samples are spaced at regular intervals serious error may result from incorrect combining of assays if the values of individual samples or sections vary widely. An example of this is given below, and illustrated by Figure 34. *A* and *B* are drifts 300 feet long and *C* and *D* are raises 100 feet long connecting the drifts at the end of the block, the average grade of which it is desired to estimate. The figures show the length and value in dollars of the samples around the block, the first figure in each instance being the length (equals the width of the ore). The question arises as to what weight should be given to the raise samples, samples in both drifts and raises being taken at regular 25-foot intervals.

If equal weight is given to all the samples (including the raise samples) the tonnage based on 12 cubic feet per ton will be 17,812, the average grade \$7.98, and the total value of the block \$142,140.

In this case the average grade of the raise samples is higher than that of the drift samples; and their influence, when given the same weight as the drift samples, is out of proportion to the volume of ore which they actually govern.

If the block is divided into four smaller blocks by lines *f-f-f-f* and *g* as shown and each block be computed separately the end or raise values will then be averaged more in accordance with their influence.

The calculation, then, is as follows:

Block A $\frac{300+200}{2} \times \frac{50 \times 6.67}{12} = 6,948$ tons at \$4.82.

Block B $\frac{300+200}{2} \times \frac{50 \times 8.0}{12} = 8,333$ tons at \$7.41.

Block C $\frac{100 \times 50}{2} \times \frac{7.00}{12} = 1,458$ tons at \$21.64.

Block D $\frac{100 \times 50}{2} \times \frac{6.00}{12} = 1,250$ tons at \$4.83.

6,948 tons at \$4.82	-----	\$33,489
8,333 tons at 7.41	-----	61,748
1,458 tons at 21.64	-----	31,551
1,250 tons at 4.83	-----	6,037
<u>17,989 tons at 7.38</u>	-----	<u>132,825</u>

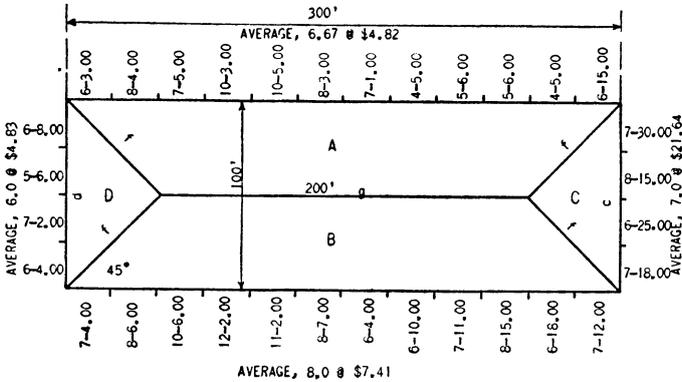


FIGURE 34.—Showing division of block for estimating average grade

By this method the average grade is \$7.38, and the total value of the the block is \$132,825 as compared to an average grade of \$7.98 and total value of \$142,140 by the less precise method. In this particular case the percentage of error in the calculations is probably no greater than the sampling error. Had the values in raise *C* been greater and the other values remained the same the error would have been greater by the first method of calculating.

ERRATIC HIGH ASSAYS

The treatment of erratic high assays is a subject that has elicited much discussion. Practices in this respect are given in some of the examples of estimating methods cited in following pages. Erratic high assays are most common in ores of the precious metals, and it is necessary to consider their treatment carefully in taking them into calculations of average grade.

Obviously, one very high assay among a few only will greatly affect their average value, and as the proportionate number of low or normal assays increases the effect of the abnormal high one will decrease.

It is often argued that there is no mathematical reason for any particular high assay being treated differently from other assays, since if high-grade spots exist and a sufficient number of samples is properly taken such spots will be represented by samples in proportion to their actual occurrence in the ore body; they should therefore be inserted in the calculations at their full value whenever they occur.

Although this is correct from a purely mathematical viewpoint it is not necessarily so in practice, since usually, in the type of ore under consideration, the values as determined by assay are not true representations of the value of the face sampled and may even fail to represent the material included in the sample. A small particle of free gold included in the assay pulp and not proportionately represented in the reject may give an assay several times the actual grade of the sample. Furthermore, in high-grade material it is often impossible, if the gold is coarse, even to get two assays of the same pulp to check within reasonable limits.

It therefore seems justifiable to reduce high assays in some instances by double averaging or by empirical rules based on experience, especially since in numerous instances actual results from mining and milling have proved this practice necessary. In making these reductions the experience and judgment of the estimator must be called upon.

A single erratic high assay from among a series of uniform lows in a stope or drift usually demands, first, a re-assay of the sample; if this checks the original, a re-sampling on or close to the line of the original sample may be advisable. If the high assay value is repeated on this sample it is reasonable to suppose that a high-grade spot does exist. The assay may then be inserted in the calculations at its full value or at a reduced value depending upon experience with the ore and the judgment of the engineer.

If, however, the erratic high assay occurs among a series of high assays the full value would usually be given to it. The various considerations involved and wide range of conditions met make the question so complex that for brevity the reader is referred to the examples of practice in the following pages without further discussion here.

EXAMPLES OF ESTIMATING PRACTICE

In the following pages examples of estimating practice at a number of typical mines are given, covering a wide range of conditions and types of ore bodies. The methods of sampling at most of these mines have been described in a previous chapter, with brief notes on the types of ore bodies and character of the mineralization in each instance.

GOLD AND SILVER MINES

ARGONAUT MINE, AMADOR COUNTY, CALIF.²

The areas are plotted monthly on a longitudinal section and a record is also kept of the tonnage and grade of ore taken from each stope. From these records factors can be derived for estimating the grade and tonnage of any block which has been exposed on two or more sides. The ore body is fairly regular in its characteristics and the factors, derived from actual mining results, take account of dilution effects by waste rock. Consequently the grade factor and the tonnage factor (about 12.5 cubic feet per ton for solid ore and 18 cubic feet per ton for broken ore) both check very closely.

SPRING HILL MINE, HELENA, MONT.³

Estimates of tonnage taken from the mine are based on a factor of 1.8 tons per car. These are checked accurately in the mill by weighed samples. Estimates of ore reserves are made by roughly measuring blocks of approximately uniform value and figuring the tonnage on a factor of 10 cubic feet in place per ton of ore. The value of the block is arrived at by observation of the ore and from experience based on chute samples, hand samples, drill cuttings, and mill heads. It is wasted effort to try to measure reserves more accurately, as the ore is about three-fourths mined out before the outlines are fully known, and it is entirely mined out before the correct value is known; this is because of the irregular ore outline and the spotty distribution of the gold.

PORCUPINE DISTRICT, ONTARIO

Special assay plans and sections are kept on which are marked the assay values and lengths of all samples. Estimates are usually made from longitudinal sections of the veins. At some mines the development and stope outlines are plotted on drawing paper and the assay values on tracing cloth pinned over the paper. Tonnage estimates are made from these sections by multiplying the area of each block by the average thickness of the ore as indicated by the channel samples and using a factor of about 12 cubic feet per ton for solid ore. Broken ore is figured at 20 cubic feet per ton.

At one mine carefully determined factors of 11.4 cubic feet per ton for solid ore and 19 cubic feet per ton for broken ore are employed. At this property each vein is treated differently in making the estimates, depending upon conditions and previous experience with the ore therefrom. Sometimes the original routine channel samples are used. In other instances the samples across the drift backs taken after the level has been silled out and the backs taken down are employed. In wide veins these are the samples used. All assays are figured to a minimum stoping width of 5 feet. In stopes known to be high grade, erratic high assays are arbitrarily cut down to \$100 before they are entered in the calculations. In stopes of average grade these assays are cut to \$50. In calculating average grades the assays are weighted by the usual length times dollars method.

Where diamond-drill hole assays are used in estimates of reserves high assays are arbitrarily cut to \$10. Where ore is shown in diamond-drill holes the sample assays may be used in the calculations if the ore cut is between faces of developed ore, in which cases ore

² Vanderburg, William A., work cited.

³ Pierce, A. L., work cited.

is figured to extend for 50 feet around the hole. Isolated diamond-drill ore showings are not included in estimates of ore reserves. In estimating the grade of broken ore reserves in shrinkage stopes the weighted average of all regular breast and "slash" samples from the stope to date is used.

At another mine in this district the basis for estimating proved ore depends upon geological conditions and the nature of the vein, but figures never include ore exposed only on one side for more than 62½ feet above or below the level nor more than 50 feet ahead of a development face. A factor of 12 cubic feet per ton is used at this mine. In estimating solid ore between the back of a stope and the level above the last or upper channeled breast samples in the stope are averaged with the samples from the drift above. The distance between samples is ignored in all reserve estimates (they are usually at regular intervals in development openings), but the length times value formula is employed in all calculations.

Box samples taken from the stope chutes are checked against the channel samples from each stope and over a long period have checked within 3 per cent of the ore estimates based on channel samples; the box samples are high.

In stopes where unusually large amounts of coarse, visible gold occur the average grade of samples usually runs below mill recovery plus tailings. Visible gold is always avoided in cutting samples. Where stope samples are consistently high the exact assay values of all samples are used in the calculations. Where an erratic high assay occurs among average lows it is arbitrarily cut to \$50.

The estimates of ore reserves are all based upon the channel sampling in development openings and stopes. The drift samples used may be only the regular breast samples; or the back samples may be averaged with them, depending upon conditions.

At another mine in this district regular channel sampling of development and stope breasts is practiced. In addition, box samples (grab samples) of the fines are taken from each car of ore. For the year 1929 these samples checked the mill recovery plus tailings within 4.8 per cent, the box samples being low. The ore averages \$7 to \$8 per ton.

VIPOND MINE, TIMMINS, ONTARIO ⁴

The cross-sectional value of an ore lens at any level is determined by averaging the back samples (channel samples cut from the back and at right angles to the vein) taken after the drift has been slashed out to the full width of the ore body, the samples being weighted according to their spacing and to the width of ore represented by each sample. This average value is calculated, after obviously high samples have been averaged with the samples on either side and the resulting value substituted in making the calculation. In arriving at the tonnage and value of the block, the area is assumed to extend 50 feet above and below the level, or is joined up with a corresponding exposure at the adjacent level if such an exposure has been developed.

⁴ Dye, Robert E., *Mining Methods and Costs of the Vipond Mine, Timmins, Ontario*: Inf. Circ. 6525, Bureau of Mines, 1931, pp. 3-4.

This method of arriving at an estimate of ore reserves without the usual raises from level to level is adopted, not because of the regularity of the deposits but rather because of the fact that they are extremely irregular in cross section and in value. While the method often does not result in an accurate estimate of the ore contained in any given block, experience has shown that losses in one quarter are offset by gains in another, so that the method at any given time results in a reasonably accurate picture of the mine as a whole.

Ore reserves are calculated annually, at which time any new blocks are included on the basis outlined above. Partly mined blocks are included for the calculated tonnage remaining unmined and at the value originally assigned to the block, unless the mining accomplished has shown this figure to be obviously incorrect, in which case a new calculation for grade is made. Broken ore is included at a value 10 per cent below the calculated value of the ore in place, unless the "box samples" (grab muck samples) taken in the course of extraction have shown this figure to be obviously incorrect, in which case an adjustment is made giving due consideration to the value indicated by the muck samples.

KIRKLAND LAKE DISTRICT, ONTARIO

Sampling methods in this district were described in a previous section.

At the Teck-Hughes mine drift back samples and test holes are plotted on assay plans, together with all geological information, to a scale of 1 inch equals 10 feet. Raise and stope samples are plotted on ore-extraction charts or longitudinal vertical projections (scale, 1 inch equals 10 feet). Estimates of ore reserves are computed from these plans, using 12 cubic feet per ton for solid ore and 20 cubic feet per ton for broken ore. No ore reserves are computed on the strength of diamond-drill results.

Samplers are instructed to throw out visible gold when cutting samples and only occasional high erratic assays occur. When they do occur they are arbitrarily cut down to twice the average of the moderately high assays before they are entered in the ore-reserve calculations. Although this procedure is admittedly not mathematically correct, it is done to provide a factor of safety.

Over a considerable period of time and a large area of vein mined, it is reported that the estimates have checked the actual grade within 10 per cent, the estimates being lower than actual grade as determined from bullion recovery plus tailings loss.

At the Wright-Hargreaves mine the faces are pick sampled as described in an earlier chapter. At this mine the usual dollar-foot method of averaging sample assays is employed in making estimates of ore reserves. The method of treating erratic high assays is of especial interest at this property, inasmuch as they are not cut to an arbitrary figure but are cut down in such a manner that their average values vary with the actual assay figures.

In estimating the grade of a given block of ore all the assays are first averaged by the dollar-foot method. A second average is then made, inserting the first average value in the place of each erratic high. Values estimated in this manner check actual bullion recovery plus tailings loss within 5 per cent. Some months the estimates are high and other months low. The ore milled averages about \$12

per ton in gold. Following is a sample calculation, using round numbers for grades and lengths for simplicity in calculating.

First calculation; actual assay value			Second calculation; with substitutions for erratic high values			First calculation; actual assay value			Second calculation; with substitutions for erratic high values		
Feet	Value	Dollar-foot	Feet	Value	Dollar-foot	Feet	Value	Dollar-foot	Feet	Value	Dollar-foot
5	\$6	30	5	\$6	30	5	8	40	5	8	40
4	9	36	4	9	36	4	10	40	4	10	40
6	10	60	6	10	60	3	350	1,050	3	40	120
8	16	128	8	16	128	4	16	64	4	16	64
10	5	50	10	5	50	5	8	40	5	8	40
8	15	120	8	15	120	6	5	30	6	5	30
5	150	750	5	40	200	8	2	16	8	2	16
4	19	76	4	19	76	2	1,000	2,000	2	40	80
5	24	120	5	24	120	5	18	90	5	18	90
6	8	48	6	8	48	5	15	75	5	15	75
5	6	30	5	6	30						
5	3	15	5	3	15						
4	8	32	4	8	32	125	-----	5,000	125	-----	1,600
3	20	60	3	20	60						
								Average value \$40			Average value \$12.80

NOTE.—The value used for the block is then \$12.80.

ALASKA-JUNEAU MINE, JUNEAU, ALASKA

P. R. Bradley⁵ has discussed the methods of sampling and estimating at Alaska-Juneau and the difficulties in gaging the grade of the ore from any system of hand sampling with the type of gold occurrence at that property. His discussion brings out the necessity of a large number of samples, the correct manipulation and assaying of samples, the correct interpretation of data, and the difficulty in even checking assays on the same sample where the gold is coarse. The discussion is too long for repetition here. Bradley states, however, that the grade of ground not already known through actual mining is gained by grab samples from the muck during the progress of development work. The assay results of such samples are interpreted in the light of experience and knowledge of the ground.

MOTHER LODE, CALIFORNIA

Arnot⁶ has written as follows regarding estimating practice:

Estimating ore on the Mother lode is usually, though not always, a relatively simple matter because of the regularity of the ore bodies. Cubic contents are computed after blocking out by drifts and raises, and a weight factor applied. Mother lode ore averages close to 12.5 cubic feet per ton, in place, and this figure is commonly used. Estimates arrived at in this manner, even when high assays and irregular widths are reduced to the common level, are usually from 10 to 20 per cent high in value and low in tonnage, depending on the vein width, because of dilution from soft walls.

A method giving very satisfactory results when, in the case of an operating mine, the ore reserve is required at the end of each year, is the "proportional areas" method. From a vertical longitudinal section of the mine, the areas stoped during the year are found and the tons per square foot as well as the ton-dollars per square foot for each stope, computed from the actual tonnage

⁵ Bradley, P. R., *Mining Methods in Alaska-Juneau Mine*: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 103-109.

⁶ Arnot, Stanley L., work cited.

and value of the ore mined. These figures, obtained from stoped areas above and below an unstoped block of ore, are averaged, and the resulting factors are applied to the unstoped block.

LUCKY TIGER MINE, SONORA, MEXICO

Mishler and Budrow⁷ have described the methods of calculating ore reserves, and the following is abstracted from their paper.

The assays, figured to stoping width are plotted on longitudinal sections and plans drawn to a scale of 1 inch = 40 feet. Ore reserves are estimated as of January 1 and July 1 each year. Backs of old stopes are surveyed and plotted on the assay maps as of those dates. If the back has not been sampled, the average assays along the levels above and below are weighted inversely to their distance from the back, and the average thus obtained is taken as the assay value of the back. If all the assays surrounding a block represent stoping width, their arithmetical average is taken as the assay of the block. The tonnage is figured by multiplying the area of the block by the stoping width and dividing by a cubic-feet-per-ton factor of 11.5. When any of the assays around the block represent more than stoping width, the excess width must be considered in figuring the average assay and tonnage.

If blocks are developed on less than four sides, it is customary to figure that ore extends 30 feet from the drifts and raises, or below the lowest level.

The full assay value of each sample is employed. The modification of high assays is warranted when the estimates are based upon only a few samples, but when several thousand samples are available, abnormally high assays will be offset by abnormally low ones.

Over a period of 14 years the estimated ore reserves have averaged 34.0 ounces of silver per ton; the ore mined during the period averaged 37.1 ounces, an error of 3.1 ounces or 8.4 per cent.

JARBIDGE DISTRICT, NEVADA

According to Park,⁸ raises are put up in ore, usually about 100 feet apart. The blocks between these raises are estimated from the average assay value and cubical contents. The tonnage figure for the ore is 18 cubic feet per ton in place and 23 cubic feet broken. The estimated tonnage in a block is generally within 10 per cent of the actual amount and is always low because of dilution in mining. The estimated grade runs about 17 per cent higher than the true grade, and this factor is taken into account when reports are prepared on newly blocked-out territory.

MOGOLLON DISTRICT, NEW MEXICO

According to Kidder,⁹ stope maps on a scale of 1 inch equals 10 feet show the width and value of ore where each sample was cut as well as the tonnage and value of ore broken in the stope during each month. The average grade of ore is calculated from the foot-ounces of gold and foot-ounces of silver, allowing 13 cubic feet per ton of ore in place.

The sampling of the smaller blocks of ore generally checks closely with the tonnage and grade of ore produced, but the larger blocks are rarely sufficiently developed ahead of mining to permit more than rough estimates of their probable production. As stoping proceeds

⁷ Mishler, R. T., and Budrow, L. R., work cited.

⁸ Park, John, work cited.

⁹ Kidder, S. J., work cited.

and the width and grade are more clearly established it has been found that the monthly estimates of ore broken, when finally checked against the ore drawn, agree closely as to tonnage and grade. The ore drawn, however, commonly exceeds the estimates of tonnage, while the grade of ore drawn will be correspondingly less.

CONSOLIDATED CORTEZ MINE, CORTEZ, NEV.

At Cortez, Nev., silver ore occurs principally in fissure veins and the ore bodies are irregular in dimensions and in grade. Hezzelwood¹⁰ states that these conditions have been responsible for evolution of the following practice:

The usual methods of blocking out the ore by measuring and sampling in making estimates of ore reserves has been found unreliable at the Cortez mine. The tendency of the ore to narrow or widen and the grade to change without apparent reason makes such methods inaccurate. A ratio between the number of feet of development work and the number of tons mined has been worked out for the operations on the lower levels which were started in 1926. This ratio furnishes a basis for estimating probable ore, particularly when development work is confined to the three known zones. This method of estimating, although not accurate, is probably as safe as any method for this form of ore deposit.

COPPER MINES

HUMBOLDT MINE, MORENCI, ARIZ.

Mosier and Sherman¹¹ write briefly regarding estimating practice as follows:

For the estimation of ore reserves a full knowledge of the ore deposits must be obtained. Caving stopes have reasonably regular outlines, and selective mining is therefore not practicable by this method.

Some material of a grade that will not pay to reduce must be mined, and some good ore on the boundaries must be left because its inclusion would bring in too much waste. The side boundaries, which are vertical or nearly vertical, are drawn as compromise planes to inclose as much ore as possible without too much waste.

Except for preliminary estimates, the volume of material within the stope outlines constitutes the ore reserves which are bounded by (1) the under-cutting level, (2) the shrinkage side outlines, and (3) the leached gossan or a stope above as the case may be. Within these boundaries the grade of ore in place is calculated by combining assays in a rational manner.

RAY MINES, RAY, ARIZ.

The following is quoted from Thomas:¹²

In churn drilling, samples were obtained by the use of a split divider. A careful record was kept of the type of material being drilled through, the color of the sludge and the character of its various mineral constituents, the weight of material cut for each 5 feet of drilling, the size of bit, and the length and size of the casing in the hole. From the weight of the sample and the size of the bit it was possible to determine whether there was caving in the hole and to thus arrive at some conclusion as to the accuracy of each 5-foot sample. The samples were assayed locally by the iodide method, and the remainder of the pulp was sent away for determination of the copper by the electrolytic method.

¹⁰ Hezzelwood, George W., *Mining Methods and Costs at the Consolidated Cortez Silver Mine, Cortez, Nev.*: Inf. Circ. 6327, Bureau of Mines, 1930, p. 4.

¹¹ Mosier, McHenry, and Sherman, Gerald, work cited.

¹² Thomas, Robert W., work cited.

In the case of diamond drilling all the sludge and water from each 5 feet of drilling was retained until the sludge had completely settled, allowed decanting of the clear water. The core was also retained. The percentage of core, however, was so small, being approximately 5 per cent of the total, that it was not used in connection with the final assay results.

Every precaution was used to obtain fair samples from the diamond-drill holes. If upon lowering the rods into the hole after a 5-foot interval was drilled it was found that the bit did not return to its former position by 1 foot, the operators were required to remove the rods and case the hole to the bottom. All samples were assayed and records kept as in churn drilling.

In considering the ore body as a whole the probable tonnage and grade of the ore in place are calculated entirely from the results of churn and diamond drilling, except in the case of a few isolated ore extensions. The first step in calculating the tonnage and grade of the ore body was to make vertical sections along the north-south lines of drill holes, showing the ore outline and the assays of each hole. Next the ore area and average grade of each section were calculated. Finally the volume and average grade of ore between each pair of adjacent sections were calculated and these in turn combined to give a volume and grade for the whole ore body. A factor of $12\frac{1}{2}$ cubic feet per ton was used to calculate tonnages; this figure was determined by actual tests on the ore. The ore was not assumed to extend beyond the drill holes until shown to do so by subsequent development.

In the course of underground mining operations calculations are made of the tonnage and grade of the individual blocks being mined; the necessary information for such calculations is obtained from actual stope sampling and mine development. In developing a block of ground for mining, as will later be explained in detail, cut-off shrinkage stopes are carried up to the capping on four sides of the block. Grab samples are taken at 10-foot vertical intervals from the broken ore across the width of the stope and for a length of $12\frac{1}{2}$ feet along the stope. The original method of sampling stopes was to cut channels, but owing to the irregularity of the stopes it was very difficult to obtain satisfactory samples in this manner, and grab sampling as described above has been adopted. The assays of samples taken from a block during drawing are adjusted to the mine head which is obtained from an automatic mechanical sampler.

In determining the tonnage and assay value of a mining block all assays which are available within the block are considered. As the cut-off stopes are carried to the capping all around the block and are sampled, as previously explained, they provide a series of vertical sections from which the tonnage and copper contents of the block can be computed. Additional data are also provided by the cut sampling of development drifts which may have been previously driven to explore the country or to provide access for subsequent mining; and these results are also taken into consideration in arriving at a final estimate of probable assay value of the ore within the block.

The future development program and the sequence of subsequent mining operations are controlled by a consideration of the assay maps and sections, upon which are also shown the geological features.

MIAMI MINE, MIAMI, ARIZ.

The following is abstracted from Hensley's paper.¹³

Estimates were based almost entirely on underground drifting and raise development. Churn drilling was largely used to delimit lateral extensions, and diamond drilling to determine the depth of the ore bodies. Estimates are based upon a factor of $12\frac{1}{2}$ cubic feet per ton of ore in place.

Tonnage estimates are made from vertical sections, parallel to the direction of drawing operations (undercut, block-caving method of mining is used) taken at 25-foot intervals; the ore limits on these sections are obtained from sampling of the final drift and raise development of which there is an average of 1 foot to each 47 tons of ore in place.

The assay value of ore in place is based on the average assay values of parallel drifts only, an average of one 5-foot drift sample for every 238 tons of ore in place. The assay value of ore in place computed in this manner is reduced 10 per cent in arriving at the final figure used.

¹³ Hensley, J. H., jr., work cited.

LA COLORADA MINE, CANANEA, MEXICO

Catron ¹⁴ has discussed the estimating practice at La Colorada, as follows:

Methods of estimating reserves vary according to the data available. In the caving blocks the ore outline is laid out on the assay map of the level or the undercut floor to include all ore possible that runs above shipping grade and exclude any that is under. All assays within the ore outline are then averaged to give an average grade; the area is computed or measured. Products of areas by grades on the level in question and the one above are then combined to give an average area and grade, and multiplied by the height to give cubic feet.

Information gathered from all previous stopping or development assays was used in making estimates of ore in blocks laid out for cut-and-fill or other methods. Either vertical or horizontal end areas may be calculated. For the initial stopes on a level, drifts and crosscuts are usually the only sources of information available. Allowance must be made for the position of the drifts, if the ore is in the shape of a vein with a high-grade core and low-grade walls. High assays may or may not be scaled down, depending upon the nature of the ore. Pillars can be estimated with considerable accuracy, as there are usually assays on both sides and at every floor.

The conversion of cubic feet to tons was formerly calculated so as to allow for sorting. In the Capote, for example, 15 cubic feet per ton was at ore time the factor used. Figure 33 (p. 121) is a chart which shows the relation between grade and cubic feet per ton of Colorada ore and which is used in making conversions from cubic feet to tons.

CAMPBELL MINE, WARREN, ARIZ.

Lavender ¹⁵ has referred briefly to practice in estimating the tonnage and grade development in a large, massive lenticular ore body of variable dip as follows:

The assay returns from all working faces are recorded in the engineering office; these data are used in the estimation of tonnage. An assay correction factor, based on comparison of mine and smelter returns, is applied. A factor of 9 cubic feet per ton of sulphide ore in place and of 12 cubic feet per ton of oxide ore in place is used in calculating tonnage. These factors have been arrived at by a comparison over a long period with smelter tonnage returns and also by actual determination of the specific gravity of the different materials.

PILARES MINE, SONORA, MEXICO

Leland ¹⁶ has written as follows concerning estimating practice at the Pilares mine:

In estimating tonnage, if ore has been cut by a single drift it is estimated to extend 10 feet on each side of the drift. If it is cut by two or more drifts at right angles or closely parallel to each other a straight line is drawn to all ore limits in the drifts, and the area inclosed is considered as a single ore body.

Ore occurring along the mine wall is known to be more continuous in vertical extent than ore in the core of the oval. Therefore, if ore has been opened on two consecutive levels, along the mine wall, within relatively vertical limits, the average of the two areas is assumed to extend the entire distance between levels, provided no raise has disproved this vertical extent. If opened on one level only and no raises have determined the height or depth, it is estimated for 25 feet above and 25 feet below the level. No ore is estimated below the lowest working level.

In core ore bodies in which no vertical prospecting has been done the ore is estimated for 10 feet above and 10 feet below the level. If a raise shows the

¹⁴ Catron, William, work cited.

¹⁵ Lavender, H. M., work cited.

¹⁶ Leland, Everard, work cited.

ore to be continuous between levels the average of the ore areas on the two levels is taken for the total distance between levels. If a raise shows that the ore is not continuous the ore is pyramided between the area on the level, up and down, to the limit of the ore in the raise. When two raises within 50 feet of each other cut ore at approximately the same elevation this ore is considered continuous between the two raises. When ore between two levels has been cut by a raise it is estimated for 10 feet on all sides of the raise for the height shown.

In all tonnage estimates ore discovered by diamond drilling is given the same value as ore found in drifting and raising.

The average grade of ore blocks is determined by taking an average of the dry-stoper sample assays of each block. Grades of working stopes are averaged with the grade of ore area on the level above, and 1 ton of ore equals 12 cubic feet in place.

The proof of the accuracy of this method is demonstrated by the fact that for the past few years the grade of ore delivered to the concentrator has very closely approximated the estimated grade of ore in the mine.

UNITED VERDE MINE, JEROME, ARIZ.

Quayle¹⁷ writes of the estimating practice at the United Verde as follows:

All underground ores are divided into four classes, according to the gangue: (1) Massive sulphide (sulphide ores with less than 25 per cent silica), (2) siliceous massive sulphide (sulphide ores with more than 25 per cent silica), (3) black schist, and (4) quartz porphyry. The proportions of these various types of ore vary in different sections of the ore body and from level to level. Over the whole ore body the proportions are approximately as follows:

	Per cent
Massive sulphide-----	44
Siliceous massive sulphide-----	34
Black schist-----	11
Quartz porphyry-----	11

The classification "ore blocked out" is used only under the following conditions:

1. Ore between levels when silled both above and below.
2. Ore silled on either top or bottom, with a raise finished and with drifts or crosscuts on top or bottom.
3. Ore with drifts and crosscuts on top and bottom, and with a raise finished, and with more than one diamond-drill hole intermediate between levels.
4. In general, all ore where there is practically no risk of failure of continuity.

The classification "probable ore" includes:

1. Ore cut by drifts or crosscuts, but without intermediate work.
2. Ore silled on one level, but without workings of any sort on the level above or below, distance up and down not to exceed width of stope.
3. Ore cut by drifts or drill holes when within the ore zone may be taken up and down a distance equal to the width of the ore.
4. In general, ore where there is a risk yet a warranted justification for assuming continuity.

"Possible ore" includes ore which can not be included in either of the above classifications, nor definitely known or stated in terms of tonnage. This includes ore cut by diamond-drill holes outside the main ore zone where good information is lacking.

Tonnage factors in terms of tons per cubic foot are computed from specific gravities. Samples of each class of ore from each stope or ore area are combined into composite samples and the specific gravities of these composites are used in calculating the tonnage factors for the respective classes of ore. In some cases composites from more than one floor are made; the averages of the specific gravities are then used in determining the tonnage factors. If no com-

¹⁷ Quayle, T. W., work cited.

posites are available, the average for the mine is used. Representative tonnage factors are:

	Ton per cubic foot	Cubic feet per ton
Massive sulphide.....	0.1264	7.911
Siliceous massive sulphide.....	.1030	9.709
Black schist.....	.1101	9.083
Quartz porphyry.....	.0967	10.341

Three general cases are considered in determining volume. Where the ore continues from level to level the formula for a truncated cone or prism is used; that is,

$$\frac{\text{Area of top} + \text{area of base}}{2} \times H = \text{volume.} \quad (H = \text{vertical height.})$$

If the ore pinches out to a line, the wedge formula is used; that is, $A/2 \times H =$ volume. Where the ore pinches out to a point, the cone or pyramid formula is used, $A/3 \times H =$ volume. Wherever it is known that the ore pinches or swells between levels, the volume of the ore body is figured in two or more blocks, being split where the dip of the ore changes. In general, the volumes of definitely outlined but irregularly shaped blocks, such as those that contain pillars, are calculated as special cases, adhering to geometric principles.

A correction for dike dilution of both grade and specific gravity is used for all dikes over 1 foot wide. In fire areas, where the ore limits can not be sampled or accurately determined, but are surrounded by old stopes, the wall assays of which show ore, 50 per cent of the area is assumed to have the average value shown by the old assays and 50 per cent of the area to average 3 per cent copper. This, in a rough way, includes those samples approaching our mining limit of 3 per cent copper.

In the calculation of grade all stope samples are used, making adjustment by the use of wall assays according to the method or direction of mining. Erratic high assays are reduced to the general average of adjacent samples unless local conditions indicate otherwise.

In estimating ore reserves a detailed knowledge of the principal ore deposit of the mine and the relationship of the smaller ore bodies is requisite. The method of projecting ore outlines is empirically chosen in each case, and therefore can not be stated.

ENGELS MINE, PLUMAS COUNTY, CALIF.¹⁸

At Engels, tonnage reserves are calculated on the basis of production records from adjoining stopes, and no systematic sampling of ore reserves is done.

EIGHTY-FIVE MINES, VALEDON, N. MEX.¹⁹

All assay returns are recorded on assay sheets. The returns from all faces are plotted on a longitudinal section assay map.

An ore reserve estimate is computed yearly. As the stopes are generally at 100-foot intervals, the estimate is based on 100-foot blocks. The estimate includes two classes of ore—actual ore and probable ore. When the ore has been cut on two consecutive levels and the probable shape of the ore body has been outlined, the ore between the levels is all classed as actual ore. If the ore has been developed by a drift on one level only, the actual-ore outline is drawn 25 feet below the drift, 35 feet beyond the face, and 35 feet above the drift (or the proved height if raising or stoping has been done).

When a drift on the bottom level, or any drift that extends beyond other workings, is in ore, an additional 25 feet is classed as probable ore, below,

¹⁸ Nelson, W. I., work cited.

¹⁹ Youtz, Ralph B., work cited.

beyond, and above the drift. Where levels are 250 to 300 feet apart and each level is in ore a similar estimate of probable ore in addition to actual ore is made. Probable-ore estimates resulting from the application of these rules are often reduced because of the erratic nature of the ore bodies in certain sections of the mine, especially in the east side or where the walls are andesite.

In calculating the grade of ore reserves proper deductions are made for differences between mine and smelter assays and for dilution. A factor of 14 cubic feet per ton of ore in place and 20 cubic feet of broken ore per ton is used in calculating tonnages. The former is high enough to allow for sorting and for the occasional horses of waste encountered in stoping.

The amount of broken ore in shrinkage stopes is estimated monthly.

BURRA-BURRA MINE, DUCKTOWN, TENN.

The method of computing tonnage and grade of ore at this mine is briefly as follows, according to McNaughton.²⁰

Burra-Burra mine is laid out in 100-foot blocks measured along the strike and from track to track of haulage levels. These blocks are designated by the level, direction, and distance from Burra-Burra shaft. Tonnage is estimated by blocks after drifting through them on both levels and crosscutting the four corners with drill holes or drifts. A factor of 8.1 cubic feet per ton is used in computing tonnage. Grab samples are taken twice weekly in the stopes and daily in development headings. Assays for copper and sulphur are recorded by blocks and used to control the grade of production. Assay values of ore reserves of the entire mine are computed yearly. Blocks containing active stopes are given the grade of the last 50 assays from the stopes in these yearly estimates. New blocks are given a weighted average grade based upon development and drill-hole assays. In practice this method has proved accurate for all practical purposes.

MAGMA MINE, SUPERIOR, ARIZ.²¹

In estimating the tonnage in a block of ore between two levels, the average width of the vein on each level is first calculated by averaging the widths of ore cut by all crosscuts on each level. The averages for the two levels are then combined into an average width for the entire block. This, multiplied by the length of the block, gives the volume in cubic feet. A ton of ore is equal to 10.5 cubic feet in place. The method used to determine the grade of ore is to calculate "average value of ore" of all openings in the vein on each level, in foot units. From these the average of the mine is readily obtained.

MICHIGAN COPPER DISTRICT

According to Vivian,²² no sampling of ore in place or in muck piles is practiced in the district because of the occurrence of the copper in metallic form, irregularly distributed, and in pieces of considerable size. Experience has shown that it is impossible to secure results approaching reasonable accuracy in these mines.

The practice followed is for the section foreman to keep constantly informed as to the assay value of the mill product coming from his section through written reports to him. On his daily trips into development openings he notes the condition of the face and estimates its grade. At the end of the month he combines his notes on daily estimates and makes a composite estimate of grade for the ore developed during the month. Owing to the great linear extent and depth of the operating mine and the care with which records have

²⁰ McNaughton, C. H., work cited.

²¹ Snow, Fred W., work cited.

²² Vivian, Harry, *Deep-Mining Methods, Conglomerate Mine of the Calumet & Hecla Consolidated Copper Co.: Inf. Circ. 6526, Bureau of Mines, 1931, 20 pp.*

been kept it is possible to gage (without sampling) the grade of ore now exposed and also that to be developed in the immediate future with a degree of accuracy not to be obtained by sample assays.

Estimates of tonnage beyond the limits of complete development and the grade assigned to such extensions are carefully and conservatively computed and have been subjected to repeated check by actual mill extraction records. All geological indications are carefully considered and only such extensions of the ore lines are made as can be safely counted upon for the future on the basis of the obvious tendencies of the vein at the time.

BUTTE DISTRICT, MONTANA

The following is abstracted from a paper prepared by a group of mine officials representing a number of operating companies in the Butte district.²³

Estimates are usually made from longitudinal sections drawn to a scale of 1 inch equals 100 feet. One company uses plan maps. The geological department has charge of estimating ore reserves for one company. A certain minimum width and assay value are assigned by the operating department as the lower limit for ore. Assay-curve records are made of all drifts and raises; and the drifts and raises, the curves of which show the required grade of copper, are colored red on the section while those in waste are colored gray. Blocks are then laid out and their areas determined by planimeter.

Average widths are taken from assay records or actual stope measurements when available. Ten cubic feet per ton is the average factor for ore in place, varying between 9 and 11. The grade of each block is calculated from the assay curves of the drifts and raises or from more complete daily reports where necessary. The outlines of blocks are laid out with due consideration for the pitch of the ore shoot, the losses of waste it may contain, and other irregularities.

Differences between the estimated grade of blocks and the grade actually mined depend largely upon the width and character of the veins. When uniform ore is exposed in several drifts where the widths were suitable for convenient mining and the walls definite, the methods used are very accurate. It was found that by checking the grade of ore shipped from several drifts of this kind against the estimated average for the same drifts an error of only 0.5 per cent was found in both copper and silver. In most instances, however, the ore is not uniform in character, and the assay widths are not suitable for mining. In these cases, the assay widths are increased to fit the mining practice and the corresponding decrease figured in the grade.

MATAHAMBRE MINE, PINAR DEL RIO, CUBA

At the Matahambre mine, where the ore bodies are in the form of large pipes of lenticular cross section, visual sampling usually provides reliable information as to grade of ore. Estimation of reserves is discussed by Richert²⁴ as follows:

An ore reserve estimate is made every three months. As the ore is of comparatively constant grade, tonnage alone is considered in making this estimate. Three general rules governing the calculation of reserve ore are as follows:

1. A stope silled out, first cut taken, but having no raise run in ore connecting the stope with the level above, is considered as having no reserve.

²³ Transactions American Institute of Mining and Metallurgical Engineers, Mining Methods in the Butte District: Vol. 72, 1925, pp. 243-244.

²⁴ Richert, George L., Mining Methods at Minas de Matahambre, Matahambre, Pinar del Rio, Cuba: Inf. Circ. 6145, Bureau of Mines, 1929, p. 4.

2. A stope silled out, first cut taken, and having a raise run in ore connecting the stope with level above, but with no ore silled out above this level, is considered as having a reserve. In calculating this reserve the area of the stope is considered as the base of a cone and the height to the next level as the height of the cone. The reserve is calculated as a cone, the area of the base times one-third the height.

3. In calculating the reserve of a stope silled out and connected by a raise or raises in ore to the level above, with a stope being silled out on this level above, the average area of the two stopes and the distance between them is used.

A new plan is made of each stope after each cut is taken, and the latest plan is used in calculating the reserve of any stope. The tonnage of ore reserve is calculated on the basis that 11.62 cubic feet of ore in place is equal to 1 short ton. Possible ore and probable ore do not enter into the reserve calculations. Over a period of years the recovery of the estimated ore has been over 90 per cent. The other 10 per cent includes ore lost by caving, ore of too low grade to mine or pinching out, and overestimation. The factor, 11.62 cubic feet per ton of ore, takes into consideration this 10 per cent discrepancy and compensates for it.

INSPIRATION MINE, INSPIRATION, ARIZ.

At Inspiration disseminated copper ore occurs in large flat bodies having a total length of 8,000 feet, an average width of 800 feet and an average thickness of 200 feet.

Estimates are based chiefly on the results of churn drilling according to Stoddard,²⁵ who describes the methods employed as follows:

The tonnage estimates were made on the basis that $12\frac{1}{2}$ cubic feet of ore in place would equal 1 ton. As the drilling was done on the corners of 200-foot squares, it was usual to take 40,000 square feet as a unit. In a few instances an area of one-half a unit was used, and also in a few instances the area was calculated, due to irregular spacing of the drill holes.

The ore was classified through various gradations from low carbonate to low sulphide. The footage of each class of ore in each drill hole was taken, and the sum of these footages in the four holes of the quadrilateral was divided by four to get an average footage or thickness of ore in the quadrilateral. This average footage multiplied by the area of the block gave the cubic feet of the class of ore in the block; and this amount when divided by $12\frac{1}{2}$ gave the tonnage of each class. Summing up the block tonnages gave the total tonnage of ore of each class.

The grade of the ore in an area was computed by taking the sum of the assays for each of the four holes, considering each class of ore separately, and dividing by the number of assays to get the average value. This average value of each class of ore in each hole was then multiplied by the footage to get a result in feet per cent. The feet per cent of the four holes was summed and divided by the total footage to give an average value for the area. The tonnages and grades of the individual blocks were combined to give a total tonnage and grade for the property of each class of ore. The classes of ore were finally combined to give a total tonnage and grade for all the ores.

To check this estimate the same method of computation was used on all the ore without regard to class to arrive at a total tonnage and grade.

Subsequent to these estimates, another estimate was made from sections.

Judging from the experience of other companies under similar conditions the results of the churn drilling were considered sufficiently accurate to justify the spending of huge sums for mining development and plant construction, and no underground work was done to check the churn drilling until actual mining operations were ready to begin. However, early in the development of the Inspiration division ore body there was an opportunity to check by means of drifts and raises a considerable area that had been drilled. The check of the estimate made from the drill samples against two sets of mine samples was remarkably close.

In the light of subsequent events perhaps this checking was done under the most favorable conditions. Still, the knowledge gained from mining areas

²⁵ Stoddard, Alfred C., *Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.*: Inf. Circ. 6169, Bureau of Mines, 1929, 23 pp.

which probably would not have checked so well has now made it possible to weigh the reliability of churn-drill sampling as a method for estimating the grade and tonnage of an ore body.

The original tonnage and grade estimates were made by the engineering staffs of the Live Oak Development Co. and the Inspiration Copper Co. from time to time. The study of the churn-drill records in connection with underground mining operations has always been used to guide the planning of future development work.

UTAH COPPER MINE, BINGHAM CANYON, UTAH

At the Utah Copper mine, with its large, low-grade disseminated ore body, churn-drill holes are drilled as nearly on the corners of equilateral triangles as topography will permit. Soderberg²⁶ has described the estimating practice in considerable detail as follows:

In making the ore-tonnage estimate each level was treated as a separate mine; workings, churn-drill holes perforating that particular level, and the ultimate location of the level. The specific gravity of the ore was determined, which gave a factor of 13 cubic feet per ton in place. The ore was divided into blocks 100 feet square, with the height of the shovel bank in question taken as the depth of the block. Drill-hole assays within the segment of the hole between the top and bottom elevations of the bench were averaged, and the value was assigned to the 100-foot block perforated by the hole. The intervening blocks between drill holes were assigned assay values determined by proportioning the average of the intercepted drill-hole segments in accordance with their distances from the block in question. Where blocks were cut by the drifts and crosscuts of the old underground mine workings, the assays taken were also averaged for each block.

A detailed system of toe sampling is used at the mine to enable the operator to have a close check on the grade of ore loaded by each shovel. Samples are taken every 10 feet along the face of the level, following each shovel cut. The assay results are placed on a plan map showing the toe and edge of each level. New toe-sample maps are made up every two weeks.

In the tonnage and grade calculations the toe assays were also averaged for each 100-foot block and compared with the assigned churn-drill and underground mine assays in the same block. From this comparison a discount factor was arrived at for those levels where any decided difference was noted between toe assays and drill and underground mine assays, the toe assay being used as the basis for the discount. Differences in value were found to be confined largely to old stope areas, while the drill assays checked closely with toe assays.

For estimating tonnage and grade of the ore located below the lowest level of the present working faces, namely, 6,240 feet elevation, a plan map was made as of this elevation, showing all drill holes extending below this plane. Volumes were figured by triangular prisms bounded by drill holes, multiplying the area by the average depth of the hole and weighing the assays in accordance with the footage. Refinements of this method, using corrections for triangles that were not equilateral, were tried, but the differences did not warrant this procedure. The results obtained from the separate levels were combined with this lower calculation to give the total gross figure.

Preparatory to making detailed ore estimates the cut-off between commercial ore and waste must be determined. In other words, a grade must be determined below which the material can not be mined and meet its mining and treatment costs and show a profit. To arrive at this cut-off grade certain assumptions must of necessity be made, such as the selling price of copper, the estimated recovery in per cent of gross metal content, and the cost of producing a pound of copper, which includes all costs other than stripping. The

²⁶ Soderberg, A., *Mining Methods and Costs at the Utah Copper Co., Bingham Canyon, Utah: Inf. Circ. 6234, Bureau of Mines, 1930, 23 pp.*

stripping cost is kept separate for reasons that will develop later. Table 8 is set up to illustrate the method used to determine the point of cut-off:

TABLE 8.—*Method of determining point of cut-off between ore and waste*

Per cent copper	Gross pounds per ton	Assumed recovery per cent	Pounds recovered	Assumed selling price per pound, cents	Assumed cost per pound, less stripping, cents ¹	Partial net profit per pound, cents	Loss per pound, cents
1.10	22	91	20.0	13.5	6.25	7.25	-----
1.00	20	90	18.0	13.5	6.94	6.56	-----
.90	18	89	16.0	13.5	7.81	5.69	-----
.80	16	87	13.9	13.5	8.99	4.51	-----
.70	14	85	11.9	13.5	10.50	3.00	-----
.60	12	83	10.0	13.5	12.50	1.00	-----
.50	10	80	8.0	13.5	15.62	-----	2.12

Per cent copper	Stripping cost per cubic yard	Ratio of waste to ore ²	Proportionate stripping cost per pound this basis, cents	(6+11) Total cost per pound, cents	Profit per pound, cents	Loss per pound, cents
1.10	\$0.40	½ to 1	0.50	6.75	6.75	-----
1.00	.40	½ to 1	.56	7.50	6.00	-----
.90	.40	½ to 1	.62	8.43	5.07	-----
.80	.40	½ to 1	.72	9.71	3.79	-----
.70	.40	½ to 1	.84	11.34	2.16	-----
.60	.40	½ to 1	1.00	13.50	-----	-----
.50	.40	½ to 1	1.25	16.87	-----	3.37

¹ Based on a cost of \$1.25 per ton. These figures can be varied to cover increasing costs due to increasing copper content, such as bullion freight, refining, selling, etc.

² Or one-fourth yard waste to 1 ton of ore.

Under this set of conditions a copper content of 0.6 per cent would be the point of cut-off; any grade under this figure would be waste, and anything over should be classed as ore.

Other tables should be made with a variation of doubtful assumptions to assist the engineer in establishing a safe cut-off figure.

It is at this stage convenient to set up a table of grades showing the amount of stripping any given grade of ore will carry. This is usually worked up as follows:

If it costs 40 cents to waste a cubic yard of overburden weighing 2 tons, ½ ton will cost 10 cents. It then becomes necessary to determine what grade of ore will yield a return of 10 cents per ton under the given conditions, assuming the average recovery to be 85 per cent. The figure in this case is 0.0436 per cent copper, arrived at as follows:

0.0436 per cent copper × 2,000 pounds = 0.872 pound × 85 per cent recovery = 0.741 pound;

0.741 pound × 13.5 cents = 10 cents, the cost of moving ½ ton of stripping.

By adding this increment of grade (0.044 per cent) to the ore it will support the removal of an additional half ton of waste for each addition of the increment. One can then set up the following table:

TABLE 9.—*Tonnage of stripping, plus all other costs, carried by 1 ton of ore of various grades*

Grade of ore, per cent	Tonnage of stripping	Grade of ore, per cent	Tonnage of stripping	Grade of ore, per cent	Tonnage of stripping
0.6-----	0.5	0.688-----	1.5	0.776-----	2.5
.644-----	1	.732-----	2	.820-----	3

From the foregoing tables graphs can be made from which it can be determined at a glance whether or not a certain block of material is ore or waste when the stripping ratio has been determined.

Dividends, of course, can not be paid on a grade of ore at or near the point of cut-off; the average grade, therefore, must be well in excess of the cut-off grade, and no section of the ore body that will not pay its own way should be combined with higher grades for the purpose of increasing the reserve. Possible exceptions to this rule appear, of course, when a "horse" of waste or a small amount of low-grade material occurs that has to be removed in any case. These small quantities of waste may not be easily separated and may be milled at a loss (capacity permitting) which, however, will be smaller than the cost of removing the material as waste. In such cases the low-grade tonnage is included in the ore reserve with its grade. The engineer's judgment will guide him (after he has made a complete analysis of the ore body) in rounding out an estimate where so many variables are concerned. It is well to remember that material which at the time of the estimate is waste may come into the classification of ore by an increase in the price of copper, by an improvement in metallurgy, or by a lowering of costs with improved equipment.

OLD DOMINION MINE, GLOBE, ARIZ.

Shoemaker²⁷ has described the estimating practice at the Old Dominion mine, as follows:

A carefully worked out set of rules is used for estimating ore. The common end-area formula is used for the calculation of tonnages.

The formula is as follows:

$$V = \frac{L}{2} (A_1 + 2A_2 + 2A_3 + \dots + A_n)$$

Where V is volume, L the distance between sections, A_1 the area of first section, A_2 the area of second section, A_3 the area of third section, and A_n the area of n th section.

An average factor of 12½ cubic feet of ore in place is taken as 1 ton of ore.

Ore is divided into two classes—developed and prospective. Developed ore consists of ore that has been cut on two different levels and has been proved to be continuous between by one or more raises. If ore has been cut on a level only by a crosscut, provided there is proof of continuity to another level, the ore for a distance of 60 feet on either side of the crosscut is considered as developed ore. This is an arbitrary figure based upon past experience which has shown that 60 feet is the probable minimum lateral extent of the ore. Prospective ore is that which has been cut only on one level and must be given an assigned altitude. The altitude given varies with the section of the mine but it is never over 100 feet.

The mine is divided into 100-foot blocks. For each block on every level estimate cards are made out indicating the class, grade, and tonnage of ore. This is a great convenience, making it possible to obtain estimate figures for particular areas by simply consulting the files.

In addition to these estimates one is made of the total tonnage available in each stope, together with the estimated rate of mining. This gives the operating department data on when new sections must be ready for mining to insure a uniform production.

ESTIMATING PRACTICE AT OTHER COPPER MINES

The following papers are referred to for additional information on estimating practice at copper mines:

JORALEMON, IRA B. Sampling and Estimating Disseminated Copper Deposits: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 607–620.

DICKSON, ROBERT H. Sampling and Estimating Orebodies in the Warren District, Arizona: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 621–627.

PROUTY, R. W., AND GREEN, R. T. Methods of Sampling and Estimating Ore in Underground and Steam-Shovel Mines of Copper Queen Branch, Phelps Dodge Corporation: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 628–639.

²⁷ Shoemaker, A. H., work cited.

LEAD MINES

SOUTHEAST MISSOURI DISTRICT

Poston²⁸ has described the method of estimating ore reserves at the mines of the St. Louis Smelting & Refining Co. as follows:

No mine or stope sampling is done at the No. 8 mine, but the ore is sampled and assayed at the concentrator separately from that from the other mines. All mining plans are based upon data obtained from the surface drilling, as described in the preceding section on exploration.

No exploration drifts are driven into virgin ground because the irregularity of the occurrence of the ore and its uniformly flat dip would make this form of exploration much more expensive and less effective than the present system of surface diamond drilling.

Ore at No. 8 mine that carries less than 14 feet per cent lead content is not mined; in other words, a mine stope must be at least 7 feet high and must carry at least 2 per cent lead over the entire stope face. Where more than 7 feet are involved, a calculation is first made to determine whether the entire thickness is minable. If it is not, only that portion averaging 2 per cent lead and more than 7 feet in thickness is included as ore. Ore in place is calculated at 12½ cubic feet per ton, and broken ore weighs approximately 2,000 pounds per 20 cubic feet. Before the present mining campaign was undertaken the entire property was drilled with more or less uniform spacing of holes—not on coordinates. This made it possible to apply the following method of estimating ore reserves for comparison with similar estimates made by the conventional formula. For tonnage estimates, ore is assumed to exist in a circle 150 feet in diameter, using the diamond drill hole as its center, and the value as that of the drill-core sample.

Where drill holes are irregularly spaced and fairly close together, say, at a maximum separation of 200 feet, a system of polygonal areas is used for the calculation of ore reserve tonnages. The attached sketch (fig. 31) shows in detail how these polygons are formed. For example, it is desired to outline the polygon around drill hole No. 66. The first step is to draw lines from the selected hole (No. 66) to all near-by holes (in this case holes Nos. 6, 76, 23, 14, 67, and 51). Dotted line from No. 65 to No. 66 is incorrect, since the holes included for this purpose are determined by measurements that establish the shortest diagonals of the trapezoid which is formed by the four holes under consideration.

In each instance the two holes at the ends of the shortest diagonal of the trapezoid are used. If this rule is followed throughout, the polygons can be reproduced around each hole by anyone, with the same results, thus eliminating the personal equation. Each of the near-by holes is then connected with each other; this forms a number of triangles with hole No. 66 at the apex. The center of each of these triangles is now obtained by bisecting each side and drawing a light line from the point of bisection to the opposite apex. This is done from each angle, and the center is obtained. From this center a heavy line is drawn to the center of each of the three sides. These are permanent lines and, when all construction lines have been erased, result in making each hole the center of a polygon, the area of which is in proportion to the spacing of the surrounding holes.

The polygonal method of estimating ore reserves was applied in this particular case because the spacing of the drilling made it particularly applicable. The custom in the district is to employ whichever of the conventional methods of ore estimation, as determined by experience, may be best fitted to the particular case in hand.

Polygonal areas are measured by means of a planimeter, and the final equation used to obtain the tonnage figure is as follows:

$$\frac{\text{Surface area (in square feet)} \times \text{Estimated stop height (in feet)}}{12\frac{1}{2} \text{ cubic feet}} = \text{Estimated tons of ore.}$$

This tonnage is assigned the lead value shown by the core assays from the drill hole around which the polygon is formed. All core samples containing lead

²⁸ Poston, Roy H., work cited.

visible to the naked eye are assayed to determine the true lead content. The core record then takes the form illustrated by the following example:

	590 to 595 feet, 7.2 per cent by assay	
	596 to 597 feet, 1.2 per cent by assay	
	590 to 595 = 5 feet × 7.2 per cent = 36.0 feet per cent	
Then	596 to 597 = 1 foot × 1.2 per cent = 1.2 feet per cent	
	595 to 596 = 1 foot × 1.0 per cent = 1.0 feet per cent	
	<u>7 feet</u>	<u>38.2 feet per cent</u>
or	<u>38.2 feet per cent</u>	<u>5.46 per cent</u>
	7 feet	

The example shows that the section contains the equivalent of 7 feet of 5.46 per cent ore. It will be noted that the 12 inches in this run between 595 and 596 has been arbitrarily assigned a value of 1 per cent. This is done to compensate partly for the known loss of lead in the core sample while drilling and to make a complete calculation.

The lead is much softer than the country rock and usually shows evidence of wear where the core is broken in the lead ore run. Experience has demonstrated that the total ore mined is always somewhat in excess of the estimate made prior to mining.

Methods employed at other mines in the district are similar.

COEUR D'ALENE DISTRICT, IDAHO

HECLA AND STAR MINES

At the Hecla and Star mines the grade of all faces is calculated to a diluted mining width, as it is assumed that a certain portion of the wall rock will be mined with the ore and that a minimum width of stope will be carried, according to Foreman.²⁹

Foreman states further as follows:

The amount of dilution and the minimum width of stope will be carried. The amount of dilution and the minimum width are dependent upon the condition of the vein and country rock. The tonnage and assay of the various blocks are calculated by the prismoidal formula. The factor to be used for cubic feet per ton is dependent upon the assay of the block in question.

MORNING MINE

At the Morning mine the practice is described briefly by Wethered³⁰ as follows:

The development drifts on the vein are spaced 200 feet apart vertically. At the time these drifts are run development samples are taken every 10 feet across the vein. Assays are made for lead, zinc, and silver, and together with widths sampled are posted on sample maps. The mining widths of partly mined blocks of ore are shown on the stope maps and are posted monthly.

Ore reserves are figured yearly for the entire mine, and are based upon the average stope widths and the properly weighted assays in development work done during the year. In estimating tonnages a factor of 9 cubic feet to the ton is used.

Experience has shown that between levels there is little change in the grade of the ore and that stope sampling would be an unnecessary expense. Therefore, little sampling is done in the stopes during active mining, and those samples that are taken are not considered in estimating the grade of ore in partly mined blocks.

²⁹ Foreman, Charles H., work cited.

³⁰ Wethered, C. E., work cited.

TINTIC STANDARD MINE, TINTIC DISTRICT, UTAH

The methods of sampling employed have been described in an earlier chapter as quoted from Wade,³¹ who further states simply that "ore reserves are clearly defined by structural limits and computed by weighted assays from the assay maps."

ZINC MINES

TRI-STATE DISTRICT

Estimates of ore reserves are based upon a careful analysis of the churn-drill records. Past experience has shown that an ore body usually mills out about 10 per cent better than the estimate. In estimating tonnage, a factor of 12.5 cubic feet per ton is used for rock in place.

MIXED OR COMPLEX ORE MINES

PARK-UTAH MINE, PARK CITY, UTAH

According to Hewitt,³² the estimating of ore reserves is based upon width, length, and height of each ore shoot as determined by mine development work. The assay value of unstoped ore is calculated from samples taken in adjacent stoped areas and in drifts and raises.

BLACK ROCK MINE, BUTTE DISTRICT, MONTANA

According to McGilvra and Healy:³³—

Ore reserves are estimated by multiplying the length, height, and average width for individual blocks as outlined by development work. The tonnage factor is 10 cubic feet for each ton of ore in place. In estimating positive and probable ore, the characteristics of each section of the vein, as determined by past mining operations, are taken into consideration.

GROUND HOG MINE, VANADIUM, N. MEX.

The lead-zinc ore deposit occupies a fault fissure. The sampling methods as described by Richard³⁴ have been given in an earlier chapter. Richard has also described the estimating practice as follows:

Tonnage and grade are calculated for each block, the blocks being defined by main levels and sublevels, about 65 feet apart on the slope of the vein, or 50 feet vertically, and by raises between levels, which are 50 feet apart along the strike. The average area of the top and bottom of the block, normal to the dip, multiplied by the length of the block along the strike, gives the volume of the block. A factor of 9 cubic feet per ton is used to convert volume to tonnage. An average assay value of the ore in each block is obtained by multiplying the width of the ore at each sample cut by the average assay of the cut and dividing the sum of all such products by the sum of all the widths.

³¹ Wade, James W., work cited.

³² Hewitt, E. A., work cited.

³³ McGilvra, D. B., and Healy, A. J., work cited.

³⁴ Richard, F. W., work cited.

Experience to the present time has shown that this method of estimating gives results which are too high in assay values and too low in tonnages. One reason for the high assays is that the sublevels are driven close to the hanging wall, where the best ore is concentrated, and not opened over the full width of the vein, as are the main levels. An explanation of the low estimated tonnages is that in stoping, a considerable amount of low-grade ore is mined from the footwall. About 25 per cent more ore is mined than is estimated, and the grade is correspondingly lower.

PECOS MINE, SAN MIGUEL COUNTY, N. MEX.

Here the reserves are computed from closely spaced vertical transverse sections. Grade estimates are based on the average of all assays available in a given block of ore. No account is taken of probable or possible ore. Therefore the ore outline is extended only a short distance into unknown ground.

IRON MINES

LAKE SUPERIOR DISTRICT

The methods of estimating ore reserves on the iron ranges of this district have been well described by Wolff, Derby, and Cole.³⁵

The discussion is too lengthy for repetition here. Both the average depth and area method and the cross-section method are employed, the latter being generally much more accurate than the former. Vertical sections are used for undeveloped mines, while either vertical or horizontal sections may be used for developed mines.

Certain empirical rules have been adopted for estimating tonnages and for guidance in interpreting the geological data. These rules vary with the structural conditions, which differ somewhat on the various ranges. These rules have been stated in the paper referred to above.

In computing average analysis of the ore of each class the foot-units method is employed; that is, each assay is multiplied by the footage it represents, and the summation of the like units is divided by the total footage of samples. This method of computing the average analysis of ore can be applied to estimates made from records of mine workings, using drift, raise, winze, and shaft records instead of drill-hole records, taking care to use the different records in the proper proportion. Thus, on the Mesabi range, where the layers of ore are flat lying, a 25-foot drift sample (along the bedding) may only be given the same weight as a 5-foot raise or drill sample (across the bedding).

The cubic-feet-per-ton factor varies considerably with the grade and porosity of the ore, and some companies use a curve based on the grade of ore, worked out many years ago from many tests of Mesabi ores, for computing tonnages from volume. (Fig. 32.)

³⁵ Wolff, J. F., Derby, E. I., and Cole, W. A., Sampling and Estimating Lake Superior Iron Ores: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 641-652.

MINE NO. 4, MARQUETTE RANGE, MICH.

Graff³⁶ states that at No. 4 mine, Marquette range, 12 cubic feet per ton of ore in place is the factor used.

MESABI RANGE MINE, MINNESOTA

Haselton³⁷ makes the following statement regarding the estimation of ore reserves at a Mesabi range mine:

To estimate the tonnage developed by the drilling, the holes are plotted on sections, and a careful study is made to properly connect up the various strata of ore and other materials in the holes on each section. After the sections are developed, the square footage of ore on each two adjoining sections is computed, averaged, and multiplied by the distance between them to give the cubic feet of ore. The whole ore body is similarly reduced to a cubic-foot basis, and the figure obtained is divided by a factor of approximately 12 to 15, for most Mesabi ores, to give the tonnage. Generally a discount of about 10 per cent is taken to cover loss in mining and to allow for rock within the ore body.

To determine the analysis of the ore body the percentage for each element in each sample is multiplied by the footage represented, generally 5 feet, the total of such units for all holes being divided by the total footage of sample to give the average for the ore body. It is necessary, as a rule, to estimate tonnages and values separately for Bessemer and non-Bessemer grades, and in some cases for manganiferous grade of iron ore also. Experience has proved the above method of estimation to be substantially correct under normal conditions for Mesabi range ore bodies.

MINE NO. 1, MENOMINEE RANGE, MICHIGAN

Eaton³⁸ says of the practice at No. 1 mine, Menominee range:

Tonnage is estimated by calculating volumes from the data given by geological maps and cross sections, on which all of the information available is posted. A factor of 12 cubic feet per ton is used, and a deduction of 10 per cent for rock and 10 per cent for loss is made. Pillars and stopes are laid out on the maps, and deductions are made for the supporting pillars.

Analysis is estimated from the results of sampling in drifts, crosscuts, and raises, and from the records of diamond-drill holes and "deep-hole" drill holes. The dried ore averages 57.50 per cent iron and 0.40 per cent phosphorus.

OTHER MINES, MARQUETTE RANGE, MICH.

Eaton³⁹ describes the practice at another mine as follows:

Prospecting and development drifts and raises are surveyed geologically, the formations mapped, and when in ore are sampled at 5-foot intervals. These samples and those from diamond-drill holes are recorded on maps and cross sections, and from them the outline and analysis of commercial ore bodies are determined.

A factor of 12 feet per ton is used in calculating ore developed, and a deduction of 10 per cent for incidental rock and 10 per cent for loss of mining is made.

Three grades of ore are mined, two non-Bessemer standard ores, carrying 59 per cent iron dried and 0.100 and 0.120 phosphorus, respectively, and a siliceous ore carrying 51 per cent iron, 0.080 per cent phosphorus and 17 per cent silica.

³⁶ Graff, W. W., work cited.

³⁷ Haselton, W. D., *Underground Mining Practice and Costs at a Mesabi Range (Minnesota) Mine Using the Top-Slicing System*: Inf. Circ. 6325, Bureau of Mines, 1930, 11 pp.

³⁸ Eaton, Lucien, *Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Mich.*: Inf. Circ. 6180, Bureau of Mines, 1929, 10 pp.

³⁹ Eaton, Lucien, *Mining Soft Hematite at Mine No. 2, Marquette Range*: Inf. Circ. 6179, Bureau of Mines, 1929, 15 pp.

Eaton further writes of estimating practice at another mine.⁴⁰

Estimating the tonnage of ore available in the mine is an intricate and arduous task. With a few exceptions the ore bodies are most irregular, and their outline is not fully known until a large part has been extracted. The ore bodies on each level are carefully mapped, and the amount of available ore is calculated as closely as possible. The gross tonnage is also calculated, and the difference between the gross tonnage, the sum of the available ore, and the ore already extracted is the amount left to support the surface. The different pillars and floors are numbered, and a record is kept on special maps which are brought up to date once a year by subtracting the ore mined from and adding the new ore developed to the figures for the previous years. This system is practically a continuous inventory. It is checked by recalculations every few years.

In calculating developed tonnage allowances of 10 per cent for incidental rock and 10 per cent for loss in mining (in this case mostly ore left to protect the hanging-wall slate and in the rounded corners of the arches of the rooms) are deducted from the gross amount.

The analysis of the ore in the different parts of the mine is estimated from composite stope samples for each year and from the results of diamond drilling allowance is made for contamination by incidental rock.

Values are calculated according to the formula of the Lake Superior Iron Ore Association. These calculations are too intricate to be included here, but may be found in the 1927 edition of Crowell and Murray's *Iron Ores of Lake Superior*, page 107 et seq. At the present time a base ore (51½ per cent iron natural) of non-Bessemer grade is worth \$3.25 a ton at the mine. For specific ores the price fluctuates according to analysis.

EUREKA-ASTEROID MINE, GOGEBIC RANGE

In Michigan, annual ore-reserve estimates are required by the State tax commission for valuation purposes.⁴¹ Schaus says:

Estimates of ore reserves are required once each year by the State tax commission for valuation purposes. A method of valuing ore reserves known as the Finlay system is required of the mining companies. According to this system ore above the lowest developed level of the mine is called "developed ore" and is estimated by means of cross-sections in the ordinary way. Ore below the bottom level is called "prospective ore" and the cubical content is arrived at by projecting the bottom level ore area downward a distance of 100 feet or to such a depth as is definitely indicated by structural conditions. Ten cubic feet of ore in place is usually considered equivalent to one long ton.

Schaus also writes⁴² as follows:

In the development of an ore body, the ore is blocked out so that enough trench samples may be taken across the formation to allow accurate estimate of the analysis of the ore in place. Tonnage in dike ore bodies is calculated by regular prism formula.

CORRIGAN-M'KINNEY CO., MICHIGAN AND MINNESOTA

Figure 35 is a sample of an estimate sheet used by the Corrigan-McKinney Steel Co.⁴³ On this sheet are shown tonnage estimates both by the average depth and area method (No. 1 ore body) and by the cross-section method (No. 2 ore body).

⁴⁰ Eaton, Lucien, *Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Mich.*: Inf. Circ. 6138, Bureau of Mines, 1929, 14 pp.

⁴¹ Schaus, O. M., *Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.*: Inf. Circ. 6348, Bureau of Mines, 1930, 13 pp.

⁴² Schaus, O. M., *Mining Methods and Costs at the Montreal Mine, Montreal, Wis.*: Inf. Circ. 6369, Bureau of Mines, 1930, 26 pp.

⁴³ Gillies, Donald B., McNeil, E. D., and Blackwell, Frank, *Standardization of Mine Maps*: Presented at annual meeting, Am. Inst. Min. and Met. Eng., 1930; not published.

ORE BODY	PLAN OR SECTION	PIT OR HOLE	AREA SQUARE FEET	AVERAGE AREA SQUARE FEET	DISTANCE BETWEEN AREAS	VOLUME CUBIC FEET	VOLUME DRIFTS, RAISES, ETC. CUBIC FEET	NET VOLUME CUBIC FEET	CU. FT. PER TON	TONS	REMARKS
ABOVE 7TH LEVEL											
No. 1 Ore Body											
1235-1335N & 1425-1615E	3RD SUB 7 LEV.		10 400								
	2ND SUB 7 LEV.		10 400	10 400	18'	187 200	10 462	176 738	11	16 067	
1200-1550N & 1320-1615E	2ND SUB 7 LEV.		32 000								
1235-1480N & 1365-1630E	1ST SUB 7 LEV.		30 400	31 200	18'	561 600	36 273	525 327	11	47 757	
1250-1485N & 1400-1650E	7TH LEVEL		27 000	28 700	20'	574 000	37 168	536 832	11	48 803	
No. 2 Ore Body											
1200 - 1500 N.	X-SEC. 2000		0 000	4 980	100'	498 000	5 600	492 400	11	44 764	
" "	X-SEC. 2100		9 960	7 960	50'	399 000	6 880	392 120	11	35 647	
" "	X-SEC. 2150		6 000	3 000	40'	120 000	2 240	117 760	11	10 706	
" "	X-SEC. 2190		0 000								
										203 744	
										20 374	10% ROCK & EXTRACTION LOSS
										183 370	TOTAL DEVELOPED ORE
PROSPECTIVE ORE											
BELOW 7TH LEVEL											
1250-1485N & 1400-1650E			27 000	13 500	100'	1 350 000	0 000	1 350 000	11	122 727	No. 1 ORE BODY
1300-1420N & 2000-2190E			9 000	4 500	50'	225 000	0 000	225 000	11	20 455	No. 2 ORE BODY
										143 182	
										14 318	10% ROCK & EXTRACTION LOSS
										128 864	TOTAL PROSPECTIVE ORE
SUMMARY OF ORE RESERVE											
										183 370	TOTAL DEVELOPED ORE
										128 864	TOTAL PROSPECTIVE ORE
										312 234	GRAND TOTAL

FIGURE 35.—Example of tonnage-estimate sheet

BIBLIOGRAPHY ON ESTIMATION OF ORE, LAKE SUPERIOR DISTRICT.

The following papers are here referred to for further information on ore estimation in the Lake Superior district:

- WOLFF, J. F. Estimating on the Gogebic Range: *Trans. Am. Inst. Min. and Met. Eng.*, vol. 72, 1925, pp. 653-656.
- BOWERS, R. W. Method of Sampling Ores on the Marquette Range: *Trans. Am. Inst. Min. and Met. Eng.*, vol. 72, 1925, pp. 657-658.
- WOLFF, J. F. Ore Estimation on the Menominee Range, Including Iron River, Crystal Falls, and Florence Districts: *Trans. Am. Inst. Min. and Met. Eng.*, vol. 72, 1925, pp. 659-660.
- ZAPFFE, CARL. Estimating in the Cuyuna Iron Ore District, Minnesota: *Trans. Am. Inst. Min. and Met. Eng.*, vol. 72, 1925, pp. 661-664.

HANOVER MINE, FIERRO, N. MEX.

Kniffin⁴⁴ states that:

Weighted assays are used in obtaining final grade in ore estimates. These estimates of ore tonnage are prepared by taking the areas from the sections with a planimeter. From these areas the volume is calculated by the use of the prismatic formula. Ore of high-iron content is taken at 9 cubic feet per ton and lower-grade material at 10 cubic feet per ton.

RÉSUMÉ

In the preceding pages the authors have attempted to discuss in a practical manner the various phases of the sampling and estimation of ore deposits. The purposes sought, the principles involved, the methods employed, and the applicability of different methods to different conditions and types of ore bodies, as well as the accuracy and reliability of the methods under various conditions have been discussed.

The practices employed at a large number of mines in widely separated districts for sampling and estimation of different ores occurring in different types of deposits have been described, the descriptions in many instances being in the form of quotations from previously published articles by authors who were in charge of the work. The authors of the present paper do not concede that all of these examples represent the best practice, but since the methods employed in each instance have for the most part been evolved through years of experience with each deposit, it is assumed that the results obtained were satisfactory for the object in view, the cost of the work being considered.

It is realized that the discussions presented herein might be greatly elaborated upon, but it has not been possible to compress within a single volume an exhaustive treatment of each phase of the general subject. It is hoped, however, that the principles and technique presented may prove of value to the prospector, student, and less-experienced engineer and that the experienced engineer may find in the examples of practice cited, information which will assist in selecting suitable methods of sampling for different conditions and in the interpretation of results.

⁴⁴ Kniffin, Lloyd M., work cited.

In conclusion, it is reiterated that although mathematical principles enter into the sampling of ore deposits and the calculations involved in estimation of tonnages and grades, the sampling and estimation of ore deposits is not an exact science, but that sound judgment based upon wide experience is usually required in the selection of methods and the interpretation of results in order to arrive at correct conclusions.

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