Methods of Mining and Ore Estimation at Lucky Tiger Mine

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(New York Meeting, February, 1925)

The Lucky-Tiger mine is a silver-gold property, situated at El Tigre, in the northeastern part of Sonora, Mexico, at an elevation of 6000 ft. in the Sierra Madre Mountains. It is 30 miles by wagon road from the Esqueda station on the Nacozari railroad.

Since its discovery in 1903, the mine has produced forty million ounces of silver. The ore has averaged 40 oz. silver and 0.25 oz. gold per ton. Copper, lead and zinc have been of minor importance, having averaged 0.4 per cent., 1.1 per cent., and 1.5 per cent., respectively. The richness of the ore, combined with its occurrence in narrow veins, required the use of the more expensive methods of mining.

Geology

Wall Rock

The wall rock is rhyolite and rhyolite tuff, both of Tertiary age. In rare instances, post-mineral andesite dikes have been intruded along the veins. In the upper part of the mine, the walls were fairly solid. On the lower levels, the walls are usually either kaolinized and soft or silicified and fractured, both conditions requiring closely filled stopes or much timber.

Veins

The veins have been deposited along fracture planes in the volcanics. There are three principal veins, about 600 ft. apart, all striking nearly north-south and dipping steeply to the west. Table 1 records the approximate dimensions and assay of the veins. Although the average

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depth below the surface is recorded as 1100 ft., it is probable that the lowest workings are 3000 ft. below the former top of the volcanic series.

TABLE 1.—*Approximate Dimensions and Assay of Veins*

Vein	Developed		Ratio Area Stoped to Area Developed, Per Cent.	Average Width of Ore, Feet	Average Assay Ounces, Silver	Stoping Width, Feet	Silver Assay to Stoping Width
	Length, Feet	Average Depth, Feet					
Tigre	6,000	1,200	35	1.9	68	3.5	40
Seitz-Kelley	5,000	1,000	11	1.1	105	3.0	44
Sooy	1,500	1,000	20	1.4	80	3.5	37
Totals	12,500						
Averages		1,100	25	1.7	73	3.4	40

Note: All of the vein assaying 15 oz. or more is considered as ore.

Ore Deposits

As shown in Table 1, a quarter of the area developed has been sufficiently rich to warrant stoping. The individual orebodies, within the veins, are irregularly lenticular, usually with the horizontal axis longer than the vertical. The lenses range from 500 by 2000 ft. to 10 by 50 ft. Usually the veins are continuous between deposits, the vein filling being too low grade to mine. Branch and parallel veins are common, especially on the lower levels. The width of the deposits rarely reaches 20 ft. and in such cases the veins usually consist of multiple stringers separated by waste. Usually, the deposits are less than stoping width, requiring stripping or breaking of waste and ore together. The average width of all deposits has been 1.7 ft.; the average stoping width, 3.4 feet.

Ore

The ore consists of intergrowths of sphalerite, galena, pyrite, chalcopyrite, tetrahedrite and stromeyerite, in a gangue of kaolinized or silicified rhyolite. The silver lies mostly in the tetrahedrite and stromeyerite, though all the sulfides contain commercial quantities of silver. The average grade of the clean sulfides is 550 oz. silver per ton; that of the ore seams (1.7 ft.) 73 oz. per ton; and that of the ore delivered to the concentrator, 40 oz. silver per ton. Thus the dilution is shown to be 82 per cent. This is partly low-grade ore that can be treated at a profit, but most of it is practically waste, broken to give stoping width. Stripping is practiced wherever possible, but when both vein and walls are friable, loss in the fill is avoided by breaking ore and waste together and sending both to the concentrator. In mining rich ore, low dilution must be subordinated to maximum recovery.

Development

The upper part of the mine has been developed through adits. The lowest, No. 7 adit, was driven one-half mile to cut the vein and another one-half mile along the vein to the principal deposit. Below No. 7 level, the mine is developed through underground shafts.

Levels and raises are usually driven at 100-ft. intervals; drifts are commonly 7 by 5 ft. Raises are 8 by 4 ft., and are provided with chute and ladderway, with partition between. A few crosscuts are driven during preliminary development, but most crosscutting is postponed till stoping commences, in order to provide fill for the stopes. Machine

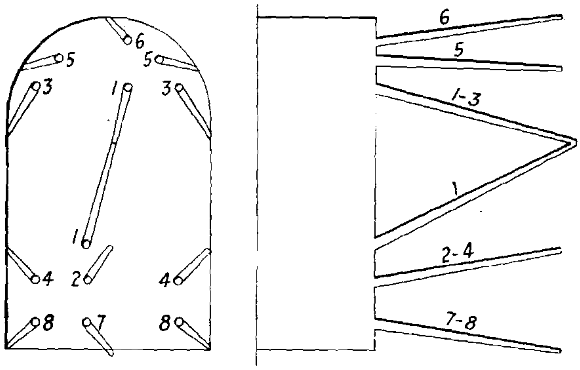

Fig. 1.—Thirteen-hole round, numbers indicate rotation of firing; in hard ground, holes are drilled upward to meet those indicated as 3.

drills are usually employed in development. The round most commonly used is illustrated in Fig. 1.

Practically all development work is done by contract. The prices range from \$4 to \$9 (United States currency) per foot, depending on the hardness of the rock. The contractor pays for his explosives and trams his broken rock to the nearest station. When timber is required, he is sometimes paid \$7.50 extra per set for enlarging the drift and erecting the set. The earnings of the contractors and their men average \$3 per day.

Diamond-drill holes are driven into the walls at intervals of 500 ft. and to depths ranging from 400 to 1000 ft. One excellent deposit has been found by this means.

Stoping

Stoping methods depend on the character of the walls and the width of the veins. Shrinkage stoping is used where the walls are firm and the vein is over 2 ft. wide. Chutes are erected at 25-ft. intervals, and the bottom of the stope is securely timbered and lagged; or, if the bottom of the block is poor, pillars are left between chutes, up to the limits of the ore. The back of the stope is carried horizontally. Stoping machines are usually employed for drilling; though, when the vein is wide, drifting machines may be used to secure the advantage of horizontal holes. About one-third of the ore is continuously drawn, to maintain the surface of the fill at a convenient level for drilling. Every effort is made to break all boulders and to support all slabs that might fall while the ore is being drawn, but in spite of all precautions, boulders, slabs, and even timber, become embedded in the fill and eventually block the chutes. Often dynamite fails to dislodge such obstructions and it is necessary to cut the lagging above the drift in order to finish drawing the stope. Rather imperfect records, kept over a period of two years, indicate that the cost per ton for shrinkage stoping is practically the same as for other methods.

Open stopes are employed where the ore extends only 20 or 30 ft. above the level, and the walls are firm. The miners drill on platforms supported by stulls. The ore falls upon the track, where boards are laid between and alongside the rails to aid in shoveling.

Underhand stoping is used where the ore extends only 10 or 20 ft. below the level. Usually the broken ore is shoveled or wheeled to development raises; in rare instances it is hoisted to the level above.

Cut and Fill Stopes

Cut and fill methods are most commonly used. They are especially suitable where the veins are narrow or the walls soft. When the veins are 2 ft. wide or less, the stopes are started in the same way as shrinkage stopes, except that chutes are established at 50-ft. instead of 25-ft. intervals. The backs of the stopes are carried horizontally. The ore and waste are blasted separately, cowhides being laid to receive the former. The broken ore is carefully sorted and all fines are shoveled into the chute. The coarse waste is left in the stope. When all the ore is removed, the skins are taken up and sufficient waste is blasted from the walls to keep the stope filled. The broken waste is sorted to remove pieces of ore, but fine ore unavoidably enters the fill. In very rich veins, to minimize this loss, ore and waste are blasted together upon hides, and all the fines are shoveled into the chutes, only the coarse waste being left in the stopes.

In narrow stopes, chutes are formed of two lines of stulls, spaced 5-ft. centers and lagged on the outside to support the fill. In wide stopes, cribbed chutes, at 25-ft. intervals, built of 6-in. round timber, are used.

When the vein is over 3 ft. wide, the stopes usually become too wide and dangerous when all the fill is blasted from the walls; therefore diagonal raises are driven into the walls, or waste from development work is dumped into the stope through the development raises. In the latter case, the chutes at the ends of the stope cannot be employed as ore passages, and wheelbarrows must be used for handling both ore and waste. Under such circumstances, flat-backed stopes are used only when the ore requires much sorting, or the waste can be blasted separately and left in the stope.

Inclined cut and fill stopes are employed when the vein is wide and requires no sorting. A single chute is established in the center of each 100-ft. block, from which the stopes extend diagonally upward, at an angle of 40°, to the development raises at each end of the block. The ore falls upon inclined plank floors, nailed to stulls, and slides of itself to the central chute. Temporary timber grizzlies, over the chutes, retain large boulders until they are broken. When vein and walls are fairly firm, slices as thick as 10 to 12 ft. are commonly taken. After all broken ore is withdrawn, the floors are removed and waste is dumped into the raises from the level above. This runs into the stope without shoveling and establishes an angle of repose parallel to the back. When it reaches a distance of 4 ft. from the back, the floor is again laid and blasting of ore is resumed. This method is cheap and occasions little loss of ore in the fill, but practically no sorting is possible in the stopes.

Contract Stoping

Where the ore occurs in one or more narrow veinlets and close sorting is necessary, contracts are often made with the miners to break and deliver the ore. Payments are based on the number of cars and the silver assay. Four rate scales are in effect, the scale selected for each stope being determined by the anticipated width and hardness of the vein. The rates range from $1 (U. S. currency) per car for 30-oz. ore, in wide veins, to $7 per car for 100-oz. ore, in narrow veins. The car holds ¾ ton.

The contractors exercise great care in breaking and sorting, in order to secure maximum rates, therefore the grade of ore from contract stopes is usually double that from similar stopes operated by the company. Contract stoping often makes it possible to mine ore that otherwise cannot be worked at a profit.

The contractors invariably use the flat-back cut and fill method, leaving the sorted waste in the stope. They are guaranteed $1 per day per man; when barren zones occur in the backs of their stopes, they are often paid for raises on a footage basis, until good ore is again encountered. When fortunate, they often earn as much as $5 (U. S. currency) per day per man. The average earnings are about $2.50 per day. The system is used throughout Mexico and gives good results with Mexican miners.

Synopsis of Stoping Methods

Shrinkage.—Where walls are firm and vein over 2 ft. wide.

Flat-back Cut and Fill.—Where vein is less than 2 ft. wide (often worked by contract); where walls are soft and vein requires sorting.

Inclined-back Cut and Fill.—Where walls are fairly firm, and vein is wide and requires no sorting.

Open and Underhand Stopes.—Where ore extends only short distances above or below level.

Drilling

All drilling in company stopes is done on contract. Hand drillers receive 12½ or 17½ cents (U. S. currency) per foot and machine drillers receive 5, 7½, 10, or 12½ cents per foot, all rates depending on the hardness of the rock. Hand drills are used on fairly soft rock, where the ore must be broken separately from the waste; machine drills are used when the ore is hard and there is little danger of the waste's falling with the ore. In soft and medium hard ore, hand drilling is cheapest; in very hard rock, the advantage lies with the machine drills. Table 2 gives a comparison of the cost of hand and machine drilling in medium hard rock.

TABLE 2.—*Hand Drilling vs. Machine Drilling in Medium Hard Rock*

	Cost per Foot, U. S. Currency	
	Hand	Machine
Drilling, labor (contract price)	$0.175	$0.107
Power		0.110
Sharpening	0.030	0.058
Nipping	0.025	0.046
Pipe lines		0.049
Drill repairs and renewals		0.065
Totals	$0.230	$0.435

Though the contract price of machine drilling is only 60 per cent. that of hand drilling, the total cost, including power, pipe lines, repairs, etc. is almost double. The greater depth of machine holes, which increases the breaking power per foot, introduces a factor in favor of machine drills, which might easily offset their higher cost per foot drilled. Contract prices in development work furnish a fairly exact means for comparing the relative cost of hand and machine work. In soft rock, the contract price is the same for hand as for machine drilling; in medium hard rock, it is 10 to 20 per cent. lower for machine work, but this advantage is more than offset by extra mechanical charges of machine drilling. In soft and medium hard rock the only advantage of machine drilling is increased speed in advancing the face; in very hard rock (hardness 5 or over) the advantage of cost lies with the machine drills. These comparisons apply only with labor at $2 to $3 (U. S. currency) per day.

About 25 machines are in use, practically all of Ingersoll-Rand manufacture. In drifting, 148 and 448 Leyners are used; in stoping, CC11 and CCW11 stopers; and in sinking and plugging, DCRW13 and BCRW430 Jackhamers. Almost all are wet machines. In very hard rock, wet stopers drill twice as fast as dry stopers and the steel runs twice as far.

Compressed air is supplied under 85-lb. pressure by one Twin Angle Compound WN4 Sullivan compressor and two Tandem Compound WH2 Sullivan compressors. The capacity of the three combined is 2000 cu. ft. of free air per minute. The air is piped to the mine through a 6-in. main; 2-in. laterals are used on the various levels. Water, under 50-lb. pressure, is piped to most of the drills. A few pressure tanks are used for isolated drills.

A standard round for drift work, employing a wedge cut, is illustrated in Fig. 1. Rounds employing the same type of cut are used in shafts and raises.

Drill Sharpening.—The machine drills are sharpened in one Sullivan and two Ingersoll-Rand sharpeners. The drills are heated in a Denver Fire Clay furnace. After forging, the bits are annealed in quicklime, reheated in a second furnace, and quenched in water. All work is done on the day shift, the daily average of drills sharpened being 500. The sharpening force consists of three sharpener operators, two drill heaters and one temperer. The cost of sharpening averages 7 cents per drill.

Timbering

Native timber is used exclusively. Delivered at the mine, round pine timber costs $20 (U. S. currency) per thousand, pine plant $45 per M, and oak lagging (4 in. by 6 ft.) 25 cents each. The pine is soft, about two-thirds as strong as Oregon pine.

The type of drift set used is shown in Fig. 2. The diagonal framing between cap and post develops the full strength of the cap. The timber is framed by hand at a cost of $1 per set. In damp drifts, water soaks up the posts for about 18 in. and causes excessive rotting just above the wet portion, therefore 3 ft. at the bottom of all posts is dipped in creosote or carbolinium. All timber for the shaft and permanent haulageways is completely creosoted.

In stopes, both stulls and square-sets are used, depending on the width of the stope. The square-sets are of standard design with the caps butting upon each other.

Tramming

Most of the tramming is done by contract. On all levels except the main haulage, the ore is trammed in 16-cu. ft. rotary dump cars. In tramming from chutes the prices are 5 and 7½ cents per car, depending on whether the distance is less or more than 500 ft. When the cars are loaded by shoveling, the price is 12½ cents per car. The cars are counted by toolroom attendants at each station. They record the cars trammed by each man and the cars trammed from each stope or other

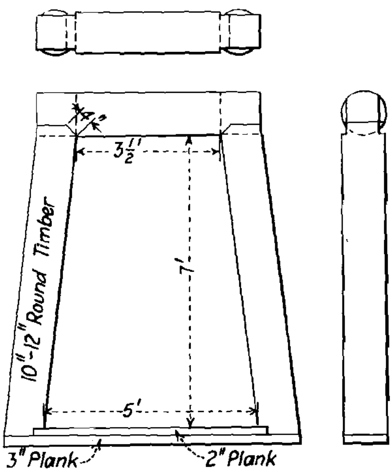

Fig. 2.—Tunnel set.

working face. The latter record is consolidated for production statistics. The count of the toolroom attendants is checked by counting the skips hoisted from each level.

On the main haulage level, the ore is trammed in 32-cu. ft. gable-bottom cars and in 16-cu. ft. rotary dump cars. Trains of 10 to 30 cars are hauled by a 3-ton trolley locomotive. The distance from shaft pockets to mill bin is 6000 ft. The contract price for loading, tramming and dumping is 5 cents (U. S. currency) per ton, settlement being based on the automatic scale reading at the concentrator. The total cost of tramming, including power and repairs, is 13 cents per ton, or 12 cents per ton mile.

Hoisting

Two-thirds of the present production is hoisted through an underground shaft at the south end of the Tigre vein. The shaft is timbered with standard shaft sets, spaced 5-ft. centers, and is divided into two 4½ by 5 ft. hoisting compartments and one 3 by 5 ft. pipe and ladderway. Between levels 7 and 10, the shaft is inclined, at an angle of 54° from the horizontal. Below No. 10 level, the inclination changes to vertical. The change in inclination is made on an arc with a radius of 50 ft. In the inclined section, the skips ride on 20-lb. rails. In the curved and vertical sections, the wheels run between timber guides, reenforced with steel wearing strips. The guides were first made of 6 by 6 in. timber, but those proved too light on the curve and 6 by 12 in. guides were substituted, and have been satisfactory. Iron idlers, equipped with roller bearings, support the cable around the curve and in the inclined section of the shaft.

Ore and waste pockets, from 50 to 150 tons capacity each, are provided below each of the lower levels and above the haulage level. A hinged gate makes possible the dumping of either skip into either of the upper pockets.

The skips are 32 cu. ft. capacity. The cables are special steel, ¾ in. in diameter. The hoist is geared to a 50-hp. slip-ring induction motor. Power is transmitted underground through submarine cable, at 2200 volts, and is transformed at the hoist to 440 volts.

The hoist is operated on three shifts and handles 200 tons of ore and waste per day, as well as all timber, steel and supplies for the lower levels. Hoisting costs 16 cents (U. S. currency) per ton of ore and waste hoisted.

Sampling and Estimating

Development Work

In sampling development work, dilution is accepted as an unavoidable evil. Samples are cut over full stoping width (3½ ft. on the Tigre and Sooy veins, and 3 ft. on the Seitz-Kelley vein); if the vein is over stoping width, the full width is sampled. Each channel 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. Usually only two samples are taken, one of ore and another of waste. Widths and assays of the various seams are recorded in the engineering office and the average assay to stoping width is calculated. 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-ft. 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 whenever the chute samples show that the ore produced is lower than milling grade.

Assay Map

The assay map is a combination plan and section. The various levels are shown in plan, with sufficient space between them to permit drawing longitudinal sections of the intervening ground. The sections represent as nearly as possible the plane of the vein, so that the dimensions of the various blocks may be readily scaled from them. The scale of 40 ft. per inch has been found most satisfactory. The assays, figured to stoping width, are plotted on the map. Gold and silver are shown to left and right of level or raise, and the width (if more than stoping width) between.

Calculation of Ore Reserves

Ore reserves are estimated as of January 1 and July 1 of each year. The backs of all stopes are surveyed on those dates and plotted on the assay maps. 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 of the back. The same general principle is followed if the assays on any side of the block indicate that side should be left as a pillar. If all the assays surrounding a block represent stoping width, their arithmetical average is taken as the assay of the block. The tonnage is calculated by multiplying the area of the block by the stoping width and dividing by 11.5 (the cubic feet per ton factor). 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 consider that the ore extends 30 ft. from the drifts and raises. The same assumption is made concerning ore below the lowest level. However, the depth of such blocks is never allowed to exceed one-third of the length. Estimates of partly developed blocks are regarded as provisional and are recalculated when the block is completely developed.

The full assay of each sample is employed. The modification of high assays is warranted when the determination of ore reserves is based on only a few samples, but when several thousand samples are taken, abnormally high assays will be offset by those abnormally low and the average will be more nearly correct if the high assays are not modified. When the valuable metals occur in minerals of higher specific gravity than those of the gangue, the average assay as calculated above will be lower than that of a single channel sample accurately cut across full stoping width.

Usually this difference is of theoretical rather than practical significance and can be regarded as introducing a slight safety factor into the calculation of average assay.

To illustrate the accuracy of the above method of sampling, averages have been prepared of two series of alternate assays along four of the principal levels on the Tigre vein. Each series represents sampling the levels at 10-ft. intervals; see Table 3. In all, 1316 assays were tabulated, ranging from 0.2 oz. to 440.0 oz. silver per ton. The average assay was 36.6 oz. Assuming this average correct, the average sampling error of each of the alternate series was 5.3 per cent.

TABLE 3.—*Averages of Alternate Assays along the Tigre Vein as Developed to Date*

Level	Number of Assays	Average of Assays, Ounces Silver per Ton			Error, Per Cent.
		Odd Numbered Assays	Even Numbered Assays	Odd and Even Numbered	
7	428	35.4	41.4	38.4	7.8
8	368	34.3	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..........	1316	34.7	38.6	36.6	5.3

Maximum assay, 440 oz.; minimum assay, 0.2 oz.

Another indication of the accuracy of sampling is seen in a comparison of assays of reserves with assays of production. During the past 14 years, the estimated ore reserves have averaged 34.0 oz. silver per ton; the ore mined, during the same period, has averaged 37.1 oz. per ton. The ore reserves having remained fairly constant, the discrepancy of 3.1 oz. in favor of the ore mined indicates that the assays of the reserves were underestimated 8.4 per cent.

A precise comparison of estimates and production is possible only by recording, over a number of years, the original estimates and the production of each block of ore. Such records were started at Tigre a few years ago. They do not yet cover a sufficiently long period to yield valuable comparisons, but they do indicate an underestimation of both tonnage and assay.

Sampling Shrinkage Stopes

In shrinkage stopes, weekly samples are taken across the fill at 10-ft. 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. The back and surface of fill are surveyed at the end of each month and plotted on stope maps. The stope maps, together with the monthly widths and assays, are employed in calculating the reserves of broken ore.

Chute Samples

Grab samples, consisting of two double handfuls, are taken from each car of ore as it is loaded. 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.

Tonnage and Assay Report

The assays of the grab samples, and the record of cars trammed, furnish the basis for the monthly tonnage and assay report (Fig. 3). The cars and cars plus assay are summed for each chute. The chutes are combined in accordance with the stopes they serve, ore from develop-

THE TIGRE MINING COMPANY, S. A.

ENGINEERING DEPARTMENT

MINE TONNAGE AND ASSAY REPORT

MONTH OF November 192 4 PRICE OF METALS, AU 20^{67} AG 0.69.

Level	Stope	Factors			Ore Broken					
		0.762	0.815	0.691	Tons	Assay		Contents		$ U. S. Currency Gross Value
		Cars	Cars × Assay Au	Cars × Assay Ag		Au	Ag	Au	Ag	
9	106–117	550	115.50	21,450	419	0.22	50.4	94.13	14,822	12,173
9	117–130	224	33.60	8,960	171	0.16	51.2	27.38	6,191	4,838

FIG. 3.—MINE TONNAGE AND ASSAY REPORT.

ment is included, and the total cars and cars plus assay are summed for the whole mine. The actual tonnage delivered to the concentrator is divided by the total number of cars, to secure a factor by which to calculate the actual tonnage from each stope and face. The gold and silver produced from each stope, and face, are estimated similarly, in ounces. The gross value is calculated from current quotations. The corrected tonnage, metal content, and gross value of the ore from each stope are tabulated by months and by years; thus, when a stope is exhausted, a fairly accurate record of its production is available.

The factors used for correcting the tonnage and metal content of the ore derived from the various stopes furnish an interesting side light on the

U. S. CURRENCY

TONS AT 2000 POUNDS

THE TIGRE MINING COMPANY

FINANCIAL STATEMENT OF STOPES

Au 20.67

Ag 0.69

MONTH OF Nov. 1924

Level	Stope	Mill Ore		Mining Costs										Milling and General	Tailing Loss		Marketing	Total Deduction Mill Ore	Total Profit or Loss	Total Profit or Loss per Ton
		Tons	Gross Value	Miners	Air Drills	Miners and Air Drills	Mech. Supervision Sampling	Timbering	Hoisting and Pumping	Sorting	Gen. Tramming	Electric Haulage	Total Mine Costs		Assay Per Cent.	Total				
9	106–117	550	12,173	110	330	440	435	495	127	154	204	60	1,915	2,030	5.5	669	2,170	6,784	5,389	9.80
9	117–130	224	4,838	67	159	226	177	224	52	63	83	25	850	826	2	258	864	2,798	2,040	9.10

FIG. 4.—FINANCIAL STATEMENT OF STOPES.

accuracy of counting cars and of grab sampling. The tonnage factor seldom varies more than a few per cent. on either side of 0.75. Since the cars actually hold ¾ ton, the tonnage records can be regarded as very accurate. The silver factor is usually a little low. For the past three years it has averaged 0.68, when it should have averaged 0.75. This represents a sampling error of 9 per cent., which is not considered unsatisfactory for grab samples. The discrepancy is probably due to the tendency of the rich ore to break fine and to the natural inclination of the trammers to take the fine ore rather than the coarse in their grab samples. Perhaps, also, the subconscious desire to secure a good sample may contribute to the discrepancy.

Financial Statements of Stopes

The profit or loss from each stope is estimated monthly in the financial statement of stopes (Fig. 4). The tonnage and gross value of the ore from the various stopes and faces are taken from the monthly tonnage and assay report. The tailing loss corresponding to the ore from each section of the mine is determined by periodical laboratory tests. The numbers of shifts worked in each stope, by hand drillers, machine drillers, timbermen and shovelers, are taken from shift boss reports and consolidated for the whole month. The total costs of drilling, shoveling and timbering, derived from the monthly mining cost statement (Fig. 5), are distributed to the various stopes on the basis of shifts worked. The cost of maintaining nonproducing stopes is distributed to the producing stopes on a tonnage basis. Costs in contract stopes are taken directly from the contract liquidations. Mechanical charges are distributed in proportion to the total cost of machine drilling. Supervision, filling, hoisting, pumping, shoveling, tramming, milling and general are all distributed on a tonnage basis. Marketing charges are distributed according to the gross value of the ore. The sum of all deductions, subtracted from the corresponding gross value of the ore, gives the profit or loss for the stope.

The stopes are grouped according to natural divisions of the mine—Upper Tigre, Lower Tigre, Sooy, Upper Seitz, Lower Seitz, Santa Maria, etc. Development costs are distributed to these same divisions. The total profit or loss from each division is then calculated. Thus at the end of each month a record is available, showing the profit or loss from each stope (with development cost disregarded) and from each division of the mine (with the development cost included). It is believed that this method shows, much better than mere assays, when a stope or division requires special attention, and when they should be stopped.

No pretense of extreme accuracy is made for this report. It is based on many approximations. It must be drafted by an experienced engineer

THE TIGRE MINING COMPANY, S. A. — DEVELOPMENT AND MINING COSTS

ORE PRODUCED 6079.87 TONS

DEVELOPMENT 926.0 FEET

MONTH OF November 1924

	Labor		Explosives		General Supplies		Power		Development		Mining		Development and Mining		Account
	Development	Mining	Development	Mining	Development	Mining	Development	Mining	Total	Per Foot	Total	Per Ton	Total	Per Ton	Number
Hand drilling—company acct	9.50	2,543.38	45.91	527.94	19.36	165.12			74.77	0.08	3,236.44	0.53	3,311.21	0.54	3,000
Hand drilling—contract	894.04	960.93	233.19	151.03	24.59	23.43			1,151.82	1.24	1,135.39	0.19	2,287.21	0.38	3,100
Machine drilling—company acct	231.51	3,230.22	221.05	2,349.28	123.81	678.22	178.35	1,889.11	754.72	0.81	8,146.83	1.34	8,901.55	1.46	3,200
Machine drilling—contract	3,682.37	17.38	1,927.36	25.84	228.23	206.46	1,214.24		7,052.20	7.62	249.68	0.03	7,301.88	1.20	3,300
Shaft sinking	522.13		24.23		556.99		39.31		1,142.66	1.23			1,142.66	0.19	3,400
Total	5,339.55	6,751.91	2,451.74	3,054.09	952.98	1,073.23	1,431.90	1,889.11	10,176.17	10.98	12,768.34	2.10	22,944.51	3.77	
Sharpening machine steel	357.32	543.79			145.15	406.69	62.30	129.58	564.77	0.61	1,080.06	0.18	1,644.83	0.27	3,500
Sharpening hand steel	214.25	220.45			3.10	12.40	5.66	5.66	223.01	0.24	238.51	0.04	461.52	0.08	3,550
Tool nipping	534.01	1,071.18			11.28	31.82	25.83	12.06	571.12	0.62	1,115.06	0.18	1,686.18	0.28	3,600
Drill pipe lines	502.04	568.25			329.34	753.88			831.38	0.90	1,322.13	0.22	2,153.51	0.35	3,700
Drill repairs and renewals	110.90	112.40			488.51	628.49	5.32	5.66	604.73	0.65	746.55	0.12	1,351.28	0.22	3,800
Total	1,718.52	2,516.07			977.38	1,833.28	99.11	152.96	2,795.01	3.02	4,502.31	0.74	7,297.32	1.20	
Timbering	1,722.65	6,274.40			1,494.62	4,304.06		27.75	3,217.27	3.47	10,606.21	1.74	13,823.48	2.27	4,000
Filling		673.18		19.03		16.51					708.72	0.12	708.72	0.12	4,100
Total	1,722.65	6,947.58		19.03	1,494.62	4,320.57		27.75	3,217.27	3.47	11,314.93	1.86	14,532.20	2.39	
Shoveling and sorting	227.20	3,604.81			4.79	297.84			231.99	0.25	3,902.65	0.64	4,134.64	0.68	4,200
Hoisting	48.85	1,710.87			51.29	235.36	34.83	104.49	134.97	0.15	2,050.72	0.34	2,185.69	0.36	4,300
General tramming	2,318.12	3,386.10			196.25	475.39			2,514.37	2.71	3,861.49	0.63	6,375.86	1.05	4,500
Tramming to mill		980.65				49.40		241.56			1,271.61	0.21	1,271.61	0.21	4,600
Total	2,594.17	9,682.43			252.33	1,057.99	34.83	346.05	2,881.33	3.11	11,096.47	1.82	13,967.80	2.30	
Sampling and assaying	169.91	1,196.58			31.32	348.72	6.94	93.12	208.17	0.22	1,638.42	0.27	1,846.59	0.30	4,700
Surveying	149.75	133.62			5.81	5.95			155.56	0.17	139.57	0.02	295.13	0.05	4,800
Clerical	521.57	1,669.77			10.12	38.17			531.69	0.57	1,707.94	0.28	2,239.63	0.37	4,900
Superintendence	1,088.50	2,473.63			31.22	32.04			1,119.72	1.21	2,505.67	0.42	3,625.39	0.60	5,000
Total	1,929.73	5,473.60			78.47	424.88	6.94	93.12	2,015.14	2.17	5,991.60	0.99	8,006.74	1.32	
Pumping	498.82	498.88			250.27	250.27	225.41	225.41	974.50	1.05	974.56	0.16	1,949.06	0.32	4,400
Diamond drilling	134.67				144.68		47.18		326.53	0.35			326.53	0.05	
Grand total	13,938.11	31,870.47	2,451.74	3,073.12	4,150.73	8,960.22	1,845.37	2,734.40	22,385.95		46,638.21	7.67	69,024.15	11.35	
Cost per ton	2.29	5.24	0.40	0.51	0.69	1.47	0.30	0.45	3.68		7.67		11.35		
Cost per foot	15.04		2.64		4.48		1.99			24.15					

Ore stoped 5,383 tons 8.66 cost per ton

Total hoisted 4,022 tons 0.54 cost per ton

Trammed to mill 6,409 ton miles 0.20 cost per ton mile

Shaft sinking ______ feet ______ cost per foot

Development and mining 0.64 tons per man-shift

Development direct 0.73 feet per man-shift

Machine drilling 20,065 feet 0.71 cost per foot

Drill sharpening 14,632 drills 0.112 cost per drill

______ Cost accountant

FIG. 5.—MONTHLY MINING COST STATEMENT.

thoroughly familiar with the operation of the mine, and he must exercise his judgment continually in distributing the various charges. Perhaps this close touch with changing conditions underground renders the report more valuable than would be possible were a more thoroughly systematized method employed. The total cost of drafting this financial statement of stopes is less than $50 U. S. currency per month. It is believed that this is returned many times through improved control of mining operations.

Costs

All costs at the mine are divided between development and mining. Development includes drifting and raising at 100-ft. intervals or more and maintenance of the passageways until stoping starts. Mining includes sublevels, intermediate raises, waste raises, stoping and maintenance after stoping has commenced. The costs of development and mining are further divided into labor, explosives, general supplies and power; and, under these headings are segregated the cost of the various occupations—drilling, timbering, shoveling, tramming, hoisting, superintendence, etc. The form of the cost sheet, with a record of typical monthly costs, is shown in Fig. 5. All important cost details are plotted monthly in order to show at a glance any unusual increases.

Safety

All possible steps are taken to prevent accidents. Detailed safety rules are posted about the mine and copies are given to new men. Besides the regular inspection of bosses and mechanics, special inspectors are chosen each month from among the best miners, to visit all working places and report to the mine superintendent. Bonuses are given to bosses who have had no serious accidents during the month. First-aid and helmet classes are held monthly.

DISCUSSION

Fred Hellmann,* New York, N. Y.—We used to take in all the raises weighted with their lengths on the dip. I think that the arithmetical average of all assays—if evenly spaced—could be used, but the method of weighting with the lengths would be more correct.

Benjamin F. Tillson,† Franklin, N. J.—When estimating the cubature to apply the values, the accepted engineering method is to use a prismoidal formula and calculate the volume on such a basis as to give a true volume on a regular figure. I would suggest the following method: Take the average area, or the area of each section, and for the block

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† Mining Engineer, New Jersey Zinc Co.

between two take the square root of the sums of the product, or the square of the first area plus the product of the two end areas plus the square of the other end areas; divide the sum of all these by three and extract the square root of the quotient. If you have figures and sections on which you can readily apply this method, it would be interesting to see how it checks out with the prismoidal formula, and whether it would not be more accurate when dealing with the geometric value of prismoids.

Fred Hellmann.—Such accuracy is rarely needed in the calculation of orebodies. When figuring tonnage, you do not calculate by the prismoidal formula the contents of the triangular prism. It would be perfectly satisfactory to take the mean height and the mean cross section, and multiply the two. I cannot imagine any case where such accuracy would be demanded as to involve the use of the prismoidal formula.

Benjamin F. Tillson—I take an interest in such accuracy because it is rather awkward to be called upon a witness stand and have the lawyer doubt the accuracy of the methods involved. It is valuable to him to point out some inaccuracy in scientific deduction.