1929

The Coarse Crushing Plant of the Desloge Consolidated Lead Company

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THE COARSE CRUSHING PLANT OF THE

DESLOGE CONSOLIDATED LEAD COMPANY.

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T H E S I S

submitted to the Faculty of the

SCHOOL OF MINES AND METALLURGY OF THE UNIVERSITY OF MISSOURI

in fulfillment of the work required for the

Degree of

METALLURGICAL ENGINEER.

Rolla, Missouri,

1 9 2 9.

Approved: ____

Professor of Metallurgy and Ore Dressing.

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P R E F A C E.

This thesis is presented to the Faculty of the Missouri School of Mines and Metallurgy in fulfillment of the work required for the degree of Metallurgical Engineer.

The results of the investigation embodied in this thesis were obtained from work carried on at the Coarse Crushing Plant of the Desloge Consolidated Lead Company, Desloge, Missouri.

ACKNOWLEDGEMENTS.

Acknowledgments are due to the Desloge Consolidated Lead Company for permission to carry on the work and submit the results, and to Mr. Will H. Coghill, Supervising Engineer of the Mississippi Valley Station of the United States Bureau of Mines for his constructive criticism.

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INTRODUCTION.

The Desloge Consolidated Lead Company is located at Desloge, Missouri, in the disseminated lead district of St. Francois County.

The crushing plant was built during 1925 to replace crushing installations at three mines. The plant was designed to deliver 2,000 tons of 7 to 9 mm. feed per day to the concentrating plant. Operating conditions have since improved so that the present tonnage is about 2,200 tons of 6.5 mm. feed per day, operating 16 hours.

The essential mineral is galena, more or less finely disseminated in a dolomitic gangue. The ore is comparatively soft and easily crushed.

GENERAL DESIGN.

The building housing the plant is of steel frame construction, the main floor being concrete.

The flow sheet of the crushing installation, Figure 1, shows the metallurgical steps in the preparation of the run-of-mine ore for the concentrator. The drawings of the plant show the mechanical details of the flow sheet. The plan view, Figure 2, shows the position of the shaft, which adjoins the crushing plant, the run-of-mine ore bin, the plant equipment, and the finished ore bin. The conveyance and the discharge of the Telsmith crusher and Buchanan rolls is shown in the side elevation, Figure 3. Section A-A, Figure 4, shows the relative position of the Telsmith crusher, pan conveyor feeder and ore bin. A section through B-B, Figure 5, shows the shuttle conveyor feeding the fine roll, roll drive, and the delivery of the finished ore to the mill bin.

DELIVERY AND STORAGE OF ORE.

The mines are operated one eight-hour shift per day, with two eight-hour shifts in the crushing plant. Two of the mines are connected by an underground haulage system, and the ore is hoisted in a two-compartment shaft adjacent to the crushing plant. The ore from the third mine is hauled to the crushing plant in 40-ton railroad cars.

The ore bin is of reinforced concrete, 37 feet by 30 feet at the top, with a bottom sloping at angles of 27° and 38° toward a central discharge. The discharge opening is 3 feet six inches by 4 feet nine inches. No protection from abrasion

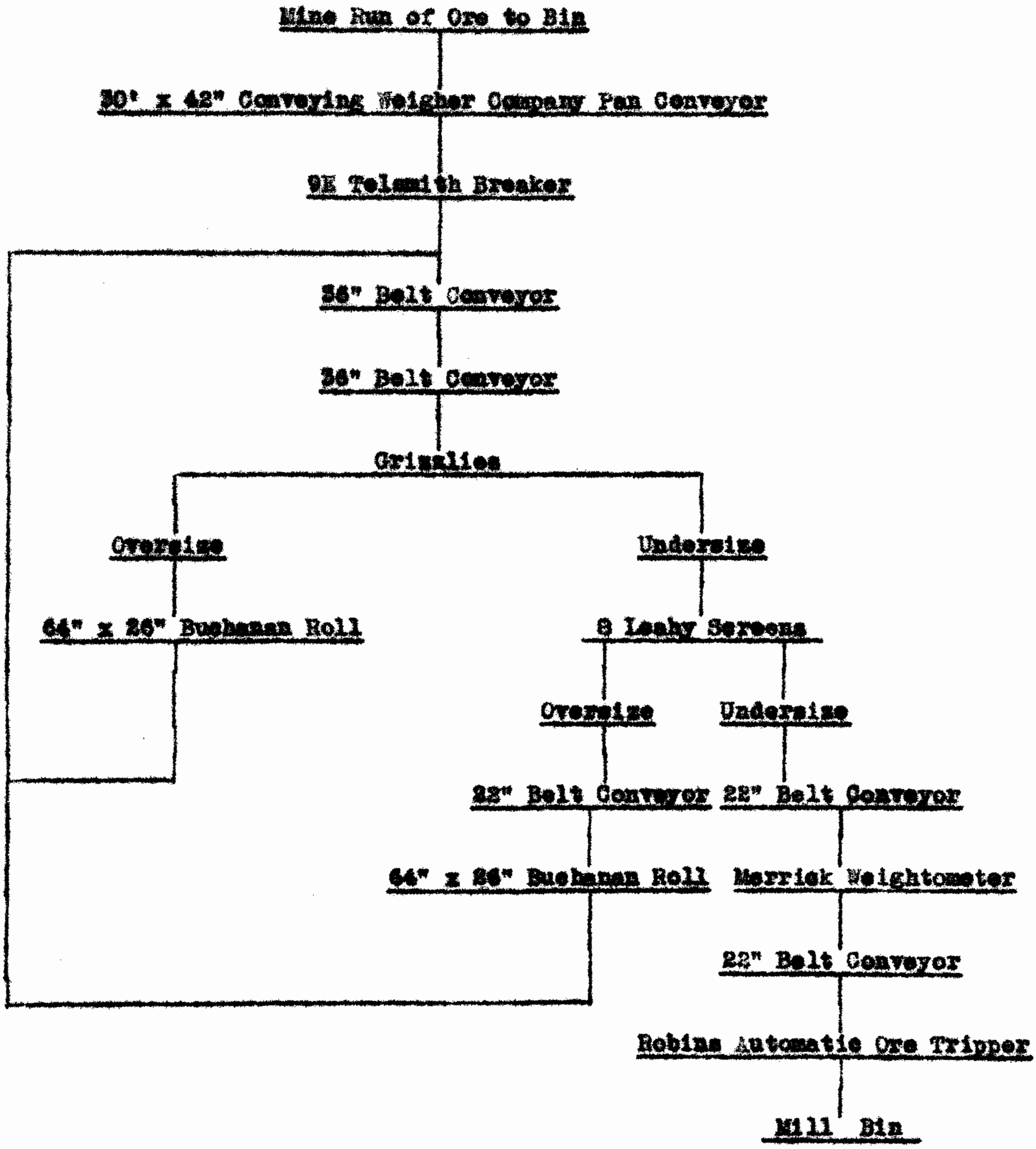

Figure 1. Flow Sheet of Desloge Consolidated Lead Company's Coarse Crushing Plant.

Figure 2. Plan View of Coarse Crushing Plant.

Figure 5. Side Elevation of Coarse Crushing Plant.

Figure 4. Sectional View of Coarse Crushing Plant Along "A-A".

Figure 5. Sectional View of Coarse Crushing Plant Along "B-B".

has been found necessary in the bin, as the fine ore and dirt form a protective coating along the sloping sides. The edges of the discharge opening are subject to severe wear, and are protected with old rod mill liners bolted to the concrete.

INTRODUCTION OF ORE TO THE PLANT.

The ore discharges upon a pan conveyor which feeds the run-of-mine ore directly to the crusher. There is no separation of fines at this point. While such a separation would, no doubt, be good metallurgical practice, the sacrifice of head room would make the repairing of the crusher difficult. Also, the expense of installing and maintaining grizzlies or screens would probably be greater than the metallurgical saving.

The pan conveyor is of the steel pan type manufactured by the Conveyor Weigher Company, having 30-foot centers and a width of 42 inches, and runs at an average speed of 9.5 feet per minute. About 13.4 horsepower are consumed for a tonnage of 150 to 180 tons per hour. The conveyor is set at an angle of 24.5°, which is about the maximum inclination for efficient conveyance. However, even at this slope there is a tendency for large boulders to roll or creep back on the conveyor, thus preventing it from carrying a full load. The angle of inclination allows six feet of clearance between the discharge point of the conveyor and the crown of the crusher, which gives ample head-room for the removal of the crusher crown.

A regular rate of feed is very desirable for smooth operation. Since the ore loads heavier at some times than others, depending upon the size of the pieces, it is planned to install a variable speed transmission on the conveyor drive.

The conveyor has side linings of 2-inch pine covered with 1/2-inch steel plate. The steel pans are lined with 3-inch oak boards. The steel side linings give twelve months service and the oak boards on the pans six months.

The pan conveyor is controlled by an operator stationed on a platform at the head of the conveyor. The plant is so designed that the operator has practically the entire plant under his observation. Control buttons within easy reach permit the stoppage of any of the equipment in case of emergency. The amount of power being used by the various units is recorded by ammeters mounted nearby. These meters are of great value in that they give warning of overloading, thus enabling the avoidance of choke-ups and other troubles.

CRUSHER OPERATION.

The crusher is a 9E Telsmith primary breaker with a steel frame and operates at a speed of 430 r.p.m., which seems high but results in increased capacity. There is no apparent increase in wear on the babbitt sleeves or gears on account of this relatively high speed. Wear on the crusher is not severe; a renewal of the babbitt sleeves for the mantle and eccentric is necessary about every six months, which is practically the only repair work done. Three years operation has produced but slight wear on concaves and gearing, consequently no data are available on the life of these parts.

The consumption of oil varies, being less when the babbitt sleeves are new, and gradually increasing as these parts become worn. The usual force feed lubrication is effected by means of a pump driven from the pinion shaft. An auxiliary pump, direct connected to a motor, is maintained for emergency use. An oil-cooling equipment is provided consisting of a 50-gallon steel drum with a one-inch water coil inside, a 50-gallon wooden barrel, and an intercooler from a Norwalk compressor. The return oil from the crusher is pumped to the drum, thence it flows to the intercooler, then to the crusher, then to the wooden barrel, and from there to the crusher oil tank. The steel drum and the intercooler may be by-passed during cold weather, when excessive cooling of the oil would result.

A mixture of high grade engine and cylinder oils is used, the proportion of each depending upon the season of the year. During warm weather a mixture of two-thirds cylinder oil and one-third engine oil is used, while in cold weather the proportions are changed to about half-and-half. The average oil consumption for 1927 was 7.5 gallons per day of two shifts.

The crusher is operated at a very close setting considering the reduction made. The maximum discharge opening is, theoretically, 1-5|8 inches and the minimum 1 inch. These openings are slightly larger at points in the bowl which receive the most feed, and, consequently, wear the most. While the above setting is quite close a tonnage of 150 to 180 tons per hour is maintained with only nominal wear on the mantle and eccentric babbitt sleeves.

The close setting of the crusher is necessary to smooth operation because if the crusher product is too coarse, the coarse mill will refuse to nip the rock. No secondary crusher is employed, consequently a crusher product must be obtained that will not be too coarse for the coarse roll feed.

A recent crusher test, giving representative data on the size of the product, power, and tonnage, is shown in Table I.

Crushing costs per ton during 1927, including power, labor and supplies, was $0.0090.

CONVEYING AND PRIMARY SIZING ON GRIZZLIES.

The crusher delivers to a 36-inch belt conveyor, running at 420 feet per minute, with 162-foot centers, horizontal for the first 61 feet, while passing all loading points, and then assuming a vertical curve of 200 feet radius for the remaining distance. The power consumption is 36.2 horsepower. This belt discharges to a second 36-inch belt conveyor with 118-foot centers, inclined 17° 46', running at 410 feet per minute, and equipped with a Robins

Table I. 9% Talsmith Crushing Test.

holdback to prevent back travel and spillage when the power is cut off. The power consumption is 42.9 horsepower. Both conveyors are equipped with 30-inch heads and 24-inch tail pulleys, and are driven by belt and spur gears.

The conveyor belts are of six-ply construction with open weave fabric all around. These belts have a rubber covering 5|32-inch thick on the carrying side and 3|32-inch thick on the pulley side. Good service is obtained with a circulating load of 900 tons per hour. Data on the life of the belts are not yet available.

The second belt delivers to the grizzlies where the first sizing is made, the grizzly undersize going to the final screens and the oversize to the coarse roll. The grizzly occupies a space 5 feet by 11 feet and is set at an inclination of 37.5°. The first grizzly used was of the ordinary straight bar type with tapered openings, but it clogged badly and required constant attention.

To overcome this difficulty a grate type of grizzly was developed, which proved to be quite satisfactory for this scalping operation. The grizzly consists of grates with openings 2-7|8 inches by 1-1|2 inches on the upper side, and 3 inches by 1-5|8 inches on the lower side, the long dimension running lengthwise of the grate. The longitudinal bars are 5|8 inch thick on the top and the cross bars are 1-1|4 inches thick, both bars being slightly thinner on the lower side to give a flared opening. The depth of the longitudinal is 2-1|2 inches and the cross 1-1|4 inches, the former being raised 1|2 inch above the cross bars, thus permitting a freer flow of material. Each grate has 56 openings and occupies a space 33-3|4 inches by 14-3|4 inches. The cross bars project on each side one-half the width of an opening, so that the grates match up when laid side by side. Four small lugs are cast on the corners to provide a means of anchorage.

As the cross bars are slightly lower than the longitudinal bars the flow of ore is not checked, but just enough resistance is offered to give what might be called a "rattling" action, which materially helps in permitting small particles to pass through, while preventing the screening through of large slabs.

The grizzly grates are made of manganese steel and last about nine months. The wear is not evenly divided over all the grizzly and the grates are shifted around to new positions occasionally, thus allowing each unit to receive its maximum wear before being discarded. The grates can be changed in a few minutes and never choke in operation except when muddy ore is encountered.

Table II shows the tonnages and sizing analyses of the products to and from the grizzlies.

Table II. Stationary Grizzly Crushing Test.

COARSE ROLL CRUSHING.

The grizzly oversize is laundered to the first or coarse roll. This is a special Buchanan roll, 64 inches by 26 inches, with ordinary shim adjustment. In purchasing the rolls it was specified that sufficient width be provided between the side frames, so that the roll face could be widened to 29 inches if desired. The larger pulley on the fixed roll is extra heavy, with shrouded rim, 25-inch face, and a diameter of 9 feet, thus giving 70 per cent more belt surface than is usually offered. The pulley on the movable roll is cast steel with a diameter of six feet and a face of 13 inches. The roll cores are double tapered and sufficiently wide to take the wider shells above-mentioned. The cores are pressed on the shafts and keyed with 3-1|4-inch square keys. They are further secured by six 1-1|4-inch dowel pins through the hub of the core into the shaft. The roll shafts are 19-1|2 inches in diameter through the cores, 16 inches through the bearings, and 12 inches through the pulleys.

Dust proof bearing caps are used and grease lubrication is effected by means of ordinary grease cups.

The roll is set at one inch and requires re-setting about every seven days, the amount of set being dependent upon conditions; usually a 1|8-inch shim is removed. Considerable trouble was formerly experienced by the roll refusing to nip the rock. This was practically eliminated by (1) reducing the speed to 54

r. p. m., a rather low speed, but still giving sufficient capacity, and (2) by burning six grooves 1 inch wide and 1|2 inch deep across the face of each shell. One such grooving of the shell lasts approximately three months and needs renewal at the end of that time.

The roll shells are of open hearth steel with a carbon range of 0.70 to 0.80 per cent and 0.65 to 0.85 per cent manganese. A shell of this composition is sufficiently soft to prevent polishing, which on one set of rolls gave trouble by causing the roll to refuse the rock. The life of the shells are 12 to 16 months. No new shells are used on this roll, as the shells from the fine roll are transferred when a change is necessary, new shells being installed on the fine roll. The new shells are six inches thick and when moved into the coarse roll are usually worn down to about four inches. The shells become badly corrugated after about two months service, but the corrugations do not seem to be deleterious to crushing.

The cheek plates wear rapidly, but only over an area of a few square inches. Small plates of manganese steel are bolted over the worn place, thus avoiding the renewal of the entire plate at frequent intervals.

The roll product is delivered to the 36-inch belt conveyor which handles the crusher product.

The tonnage and grinding accomplished in the coarse roll is shown in Table III.

The crushing cost per ton of mill feed during 1927, including power, labor, and supplies, was $0.0114.

Table III. Coarse Roll Crushing Test, Buchanan Roll,

Type E, 64 inches by 24 inches.

SCREENING.

The grizzly undersize drops into a hopper and is laundered to eight Leahy vibrating screens. The screen vibrators are run at a speed of 220 r.p.m., which gives 1,760 vibrations per minute to the cloth. The undersize of these screens is collected in a hopper which discharges to a 22-inch belt conveyor, which conveys the material to the mill bin.

The screen jacket used is a steel wire cloth 3 feet by 6 feet, with wire diameter of 0.192 inch and openings 1|4 inch by 7|16 inch. Screens with square openings were used for a while but the "Rek-Tang" variety was adopted because of greater capacity, the presence of certain amounts of slivery pieces in the undersize not being objectionable.

Screen operation is quite satisfactory. No trouble is experienced except for occasional choking caused by muddy material. If the rolls are kept fairly tight even this trouble may be ameliorated. The screens are placed on an angle of 32°, which experience has shown to be the most satisfactory. The material does not discharge well on a flatter slope, and if the screen is much steeper there is considerable fine material in the oversize.

A representative test of screening operations is given in Table IV.

The life of the screen cloth averages seven weeks. Screening costs for 1927 were $0.0036 per ton of mill feed.

Table XV. Dry Screen Test.

FINE ROLL CRUSHING.

The screen oversize discharges to a 22-inch horizontal belt conveyor, running 360 feet per minute. The length of the conveyor is 22 feet between centers. The conveyor discharges directly into the feed hopper of the fine, or finishing, roll. This roll is the same in size and construction as the coarse roll. It is run at a speed of 75 r.p. m. and is set at 1|4 to 5|16 inch.

The life of the shells on the fine roll is the same as that of the coarse roll, as both sets of roll shells are changed at, or nearly, the same time. It is important that the face of the shells on the fine roll are fairly smooth. By the time the shells on the coarse roll need renewing, the shells on the fine roll are uneven enough to make their renewal advisable.

The roll discharges on the same conveyor belt that serves the crusher and the coarse roll.

A representative test of the work done by the fine, or finishing, roll is given in Table V.

The fine roll costs for 1927, including power, labor, and supplies, were $0.0128 per ton of mill feed.

Table V. Fine Roll Crushing Test, Buchanan Roll,

Type E, 64 inches by 24 inches.

DISPOSITION OF FINISHED PRODUCT.

The screen undersize is the finished product, ready for delivery to the mill bin. The undersize falls from a hopper to a 22-inch belt conveyor, as previously stated. This conveyor is inclined 10.5°, travels 348 feet per minute, and has a power consumption of 9.4 horsepower. The conveyor passes over a Merrick weightometer, where the ore is weighed; the accuracy of the weightometer is tested every two weeks and corrected to within 0.5 per cent.

The ore discharges from the inclined conveyor to a 22-inch flat belt conveyor over the mill bin. The concentrator feed is distributed evenly in the mill bin by means of a Robins automatic tripper.

ELECTRICAL EQUIPMENT.

All of the machinery is equipped with individual motors and belt drives. Alternating current of 2,200 volts is used on the crusher and roll motors; 440 volts is used on the conveyors and screens. Westinghouse and Allis-Chalmers motors are used, all of them being of the induction constant speed type. The size of the motors used are:

	Horsepower
Crusher	150
Coarse roll	150
Fine roll	250
Screens	10
Pan Conveyor	10
Conveyor (handling screen oversize)	5
36" Conveyors (handling crusher and roll discharges). . .	40
22" Conveyor (handling finished ore to mill bin)	15
22" Conveyor (distributing ore in mill bin)	10
Crane	5.

The crusher motor starting equipment consists of a Westinghouse oil circuit breaker with a Type H Westinghouse controller, resistance grids, and a trip coil operating at 110 volts off a separate transfomer.

The roll starting equipment consists of Westinghouse Type F-10 oil circuit breakers with Type AF non-reversing controllers, and reversing switches. These reversing switches permit the roll to be reversed, which is a convenience in cleaning them in case of choking.

The 36-inch belt conveyer motors have Allis-Chalmers compensation type starters and Westinghouse 815, Type L, motor starting switches.

The current for the magnet, discussed below, is supplied by a motor generator set, consisting of an Allis-Chalmers 8 horse-power, 440-volt motor, direct connected to an Allis-Chalmers D. C. 5 K. W. generator.

MISCELLANEOUS EQUIPMENT.

The troughing and return idlers on all the belt conveyors are of Robins manufacture, equipped with Timken roller bearings and Alemite fittings, and require lubrication about once in three months.

Brushes are installed on the lower side of all conveyor belts at the discharge end, and the material cleaned off is discharged to some convenient point. Before the installation of these brushes fine damp material adhered to the lower side of the conveyors and caused undue wear on the return idlers and belts.

A Whiting 15-ton traveling crane serves the rolls and the crusher, and all heavy parts are easily and quickly handled.

A 36-inch Ohio mill type lifting magnet is placed over the first 36-inch belt conveyor between the discharges of the two rolls. Tramp iron gives but little trouble as the operator stationed at the crusher removes most of the larger pieces and the magnet takes care of the remainder.

EXPLANATION AND DISCUSSION OF TESTS.

The crushing or grinding done in a machine is often expressed by the reduction ratio, which commonly means the numerical value of the diameter of the coarsest size in the feed to the diameter of the coarsest size in the discharge. A more significant figure is obtained by first determining the mean mesh of the feed and discharge by the use of the lever arm principle*, and then obtaining the reduction ratio in the usual manner.

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*Coghill, Will H., Evaluating Grinding Efficiency by Graphical Methods; Eng. & Min. Jour., Vol. 126, No. 24, pp. 934-938, Dec. 15, 1928.

To illustrate this method the fine roll crushing test, Table V, is used. The actual feed and the discharge of the roll are sized on Tyler standard screens. In the actual feed, under the column heading "mesh", are recorded, at equal distances along an imaginary lever arm, the sizes occurring. Thus the product is minus 1.05 inches and extends through the 200 mesh screen. Opposite each screen size is recorded in the "Weight Per cent, W" column, the percentage weight of the size. Thus, the -1.05 inches + 0.742 inch size constitutes 12.9 per cent of the total weight; the -0.742 inch + 0.525 inch, 4.5 per cent. The weight percentages are considered as parallel forces equidistant along the imaginary lever arm. The fulcrum of the system may be taken at any point, but for this calculation it is taken as the limiting screen above the coarsest size, or at 1.05-inch size. So the lever arm of the -1.05 inches + 0.742 inch size is 1; for the + 0.525 inch size, 2. The lever arms for calculating the mean mesh are recorded in the column "M." As in a moment problem, the product of the lever arm and the force represents a moment. Thus, the moment for the 0.742-inch size is the product of 12.9 and 1, or 12.9, and for the 0.525-inch size, 19.0. The summation of the moments of all the sizes, 371.2, represents the moment of the system. Hence the whole sample, 100 per cent of the weight, with a lever arm of 3.71 would have the same moment. The mean mesh of the product is, then, 3.71 sieves below the fulcrum, or it is between 0.371 inch and 3 mesh. The numerical value of the mean mesh, 7.475 mm., is obtained by interpolation between the two sizes.

A similar method of calculation is used on the actual discharge. The mean mesh of the discharge is 5.63 sieves from the fulcrum, which is between the 4 and 6 mesh. The actual numerical value of the mean mesh of the discharge is 3.863 mm.

Since the reduction ratio is the ratio of the mean diameter of the feed to the mean diameter of the discharge, 1.94 is the reduction ratio obtained in the roll. This ratio may also be obtained by raising 1.414 to the $(5.63 - 3.71)^{th}$ power, since the screen scale varies by a fixed ratio, the square root of 2.

Assuming that the work done in crushing varies as the amount of surface produced, the lever arm principle may again be employed to evaluate the grinding results.

Again considering the fine roll crushing test, Table V, the screen size, and weight per cent are recorded in the same manner as for calculating the mean mesh and reduction ratio. The lever arm is, however, no longer the arithmetical series 1, 2, 3 . . . , but the geometric series 1.41, 2, 2.82 . . . . Zero may be taken at any point for the former, but for the latter, if different tests are to be compared, it must be taken at the same point. The plus 1.05 inch-size is considered as 1; hence the 0.742-inch is 1.41 and the 0.525-inch is 2. The lever arm mean surface "S" is obtained from the principle that when rock of a given configuration is crushed from a given size to a smaller size of the same configuration, the total surface varies inversely as the lineal dimension. Since the openings of the Tyler standard screens vary by a fixed ratio, the square root of 2, the surface of a unit weight of each successive screen size, ranging from coarse to fine, is 1.414 that of the preceding one.

The moments for the mean surface calculation are the product of the "Weight Per cent, W", and "Lever Arm," "Mean Surface, S." Hence the moment for the 0.742-inch size is 18.2; for the 0.525-inch size, 19.0. The summation of the moments for all the sizes is 599.1, or the resultant for 100 per cent of the weight is 5.99, and represents the mean surface of the feed. A similar calculation for the roll discharge gives 18.50 units of surface. The units of surface produced is, thus, the difference between the mean surface of the feed and discharge, or 12.51. On the basis that the work done in crushing varies as the amount of surface formed, the work done by the roll is the product of the units of surface reduction, 12.31, and the tons per hour, 284, or 3,496 surface tons per hour. Similarly, the work done per horsepower hour is the product of the units of surface reduction, 12.31, and the tons per horsepower hour, 1.825, or 22.47 surface tons per horsepower hour.

The lever arm calculations are only carried to the 100 mesh screen, all of the minus 65 mesh being considered as − 65 + 100 mesh. This measure was adopted because of the relatively small amount of the fine sizes.

In the 92 Telsmith crushing test, Table I, it was impossible to obtain a sizing of the feed, which was run-of-mine ore. The mean surface of the feed was assumed to be 1. As the "Lever Arm," Mean Surface, S" is extended to the coarse sizes, it diminishes very rapidly and for this reason the assumed figure 1 is approximately of the proper magnitude. The work done by the crusher is 11.61 surface tons per horsepower hour.

The coarse roll crushing test, Table III, shows that the roll is handling a large tonnage, 450 tons per hour, but that the units of surface reduction is small. Hence the work done by the roll is only 10.51 surface tons per horsepower hour. This is less than half the work accomplished by the fine, or finishing, roll.

The stationary grizzly test, Table II, and the dry screen test, Table IV, are calculated by means of the lever arm principle of determining mean mesh. While the sizing tests indicate that the separation is far from perfect, the results are satisfactory enough to meet the requirements of the plant.

GENERAL CONCLUSIONS.

In conclusion, a few miscellaneous topics may be briefly discussed. The first to be considered is the rather unique feature of the omission of secondary crushers from the flow sheet. This is contrary to the usual practice, especially in this district. The omission of the secondary crushers was decided upon because of simplicity of flow sheet and reduced initial and operating expenses. The results obtained in the operation have justified this seemingly radical procedure. It is not to be understood that a stage of crushing is omitted, but that the work of secondary reduction has been distributed between the primary breaker and the coarse roll. Each of these units must bear its share of the crushing.

The comparatively small operating force is worthy of comment. The elimination of secondary crushers made possible a smaller operating crew. In times of emergency two men have operated the plant successfully.

Another item closely related to the foregoing is that of centralized control by the crusher feed. This feature has been mentioned previously, but is again brought up to point out how the

the centralized control of the machinery makes a smaller operating force possible. The installation was made to follow the present trend of industrial progress, that is, to operate with modern machinery requiring a small working crew.

The operating force consists of eight men, five on day and three on night shift. Each shift has an operator, who supervises the plant, attends to oiling and makes periodical examination of the machinery, one operator who feeds the crusher, and one rock dumper. In addition, the day shift has one man in general charge of the plant, who makes such repairs as can be made during operation, and a laborer for cleaning floors and similar work.

The safety problem, which occupies such a prominent part in present day industry, has been adequately coped with in this plant. All hazards, such as belts, pulleys, and moving parts are well guarded; the plant is well lighted with numerous skylights and windows. During the twelve months of 1927 and the first three months of 1928 no accidents requiring medical attention occurred.

One way of indicating the efficiency of operation is to tabulate the delays over a period. The delays for the last three months of 1928 amounted to 10.3 per cent of the actual operating time. Of this 10.3 per cent, 63.1 per cent was due to unavoidable causes, such as coarse rock in coarse ore bin and on the crusher, lack of ore, power failures, and lack of room in the mill bin. The delays are tabulated as follows:

Cause of Delay	Per cent of Total
Mill bin full	35.9
Rolls	13.7
Screens choked	13.5
Crusher	.5
Power and electrical	5.4
Coarse rock in ore bin and on crusher	7.9
Out of rock	13.9
Conveyors	7.3
Miscellaneous	1.9
	100.0

The causes of delays are gradually being reduced, and an improvement is being noted from year to year. For 1927, 35.7 per cent of the delays were from causes beyond the control of the crushing plant.

The ultimate efficiency of any industrial plant is measured on a dollar and cents basis. The costs given below include power, labor, and supplies, and include all expenses from the time the ore is brought to the coarse ore bin until it is conveyed into the fine, or finishing, ore bin.

Crushing costs per ton of dry ore for 1928 are:

Month	Cost
January	0.0771
February	.1017
March	.0605
April	.0666
May	.0693
June	.0703
July	.0887
August	.0958
September	.0654
October	.1100
November	.0600
December	.0706

The months showing costs of 10 and 11 cents were due to the purchase of expensive equipment, such as motors and conveyor belts.