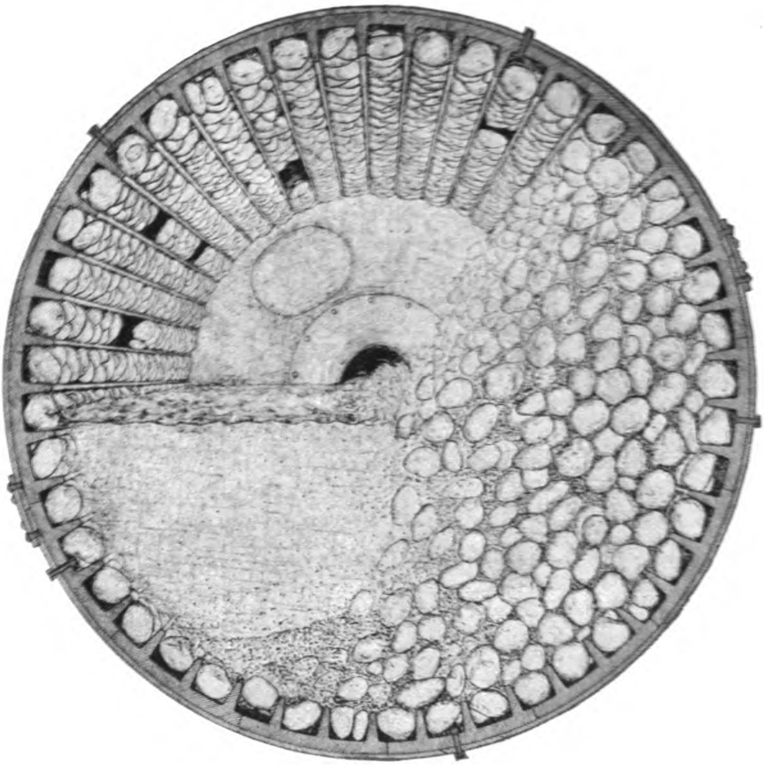
TUBE MILLING



Frontispiece

Interior of tube mill.

TUBE MILLING

A TREATISE ON THE PRACTICAL APPLICATION OF THE TUBE MILL TO METALLURGICAL PROBLEMS

By

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PREFACE

A book entirely devoted to the subject of grinding ores in tube mills is a new feature in the literature of ore dressing, this being the only book entirely devoted to the subject.

The author, in his travels and various experiences has noticed the growing popularity of the tube mill for increasing the capacity of existing mills as well as for the purpose of preparing ores for concentration, cyanidation and flotation. In fact a new field has been opened for the metallurgist and as few mills now erected or to be constructed in the near future will be without at least one tube mill, it is important that the principles involved in their operation should be made common property. It is for this purpose that the author has compiled the present volume, hoping that it will meet the requirements of the engineer and millman.

This book covers the use of the conical and cylindrical tube mills for grinding ores, indicating in detail the best means of obtaining capacity at the least cost. It contains descriptions of the latest installations and foreshadows the use of the tube mill as an intermediate crusher doing the work formerly done by stamps, rolls and chilian mills.

ALGERNON DEL MAR.

LOS ANGELES,
December, 1916.

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TUBE MILLING

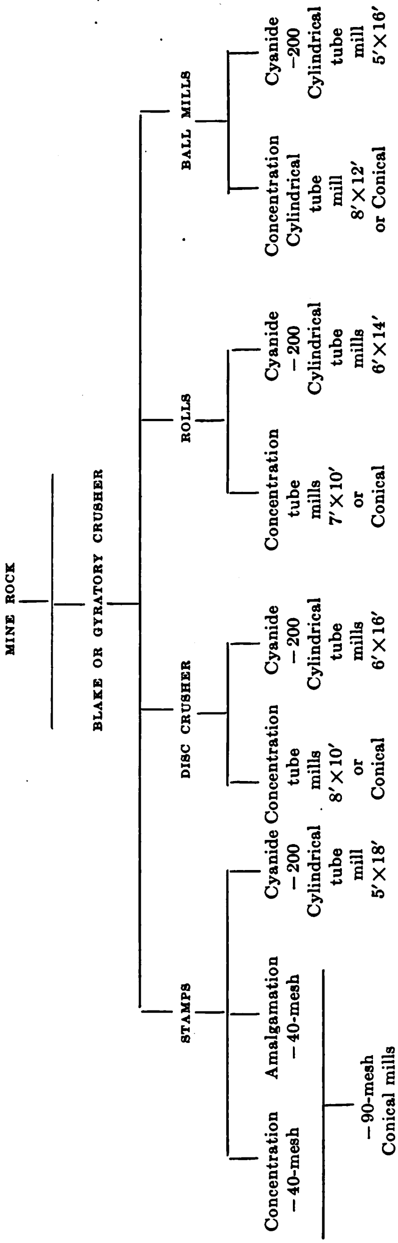
INTRODUCTION

THE GROWING POPULARITY OF THE TUBE MILL

In crushing and grinding machinery a certain type gains or loses popularity, maybe by reason of good advertising, by successful installations, or by some big company adopting that type of machine. One cannot look about at the present time without noticing the popularity of the tube mill, either in the form of a ball mill, a long cylindrical mill, a short tube of big diameter and short length, the marathon mill, or of the conical mill, with its big diameter and sizing cone. The need of finer grinding, whether for concentration, amalgamation, cyaniding or flotation, has made it necessary that we have a machine that covers a wider range of reduction than that formerly possible, so the tube mill has been lengthened, and shortened, and expanded, and contracted until we have a choice of many sizes to suit the particular work necessary. Ball mills are made with screens on the surface of the cylinder, and without screens, operating like the usual tube mill, by displacement. The 5-ft. cylindrical mill was thought to be the ultimate diameter a few years ago, now we have them 8 ft. in diameter while the conical mill still leads the way in increasing diameter, by giving us a 10-ft. mill. With the increase in diameter, the tube mill has been shortened, for a forced feed with a return of oversize is economically right, and a short mill is particularly adapted for this class of work. Costs have been lowered by studying the various factors that promote economy, so that we have grinding in these large-diameter tube mills at as low as 7 cts. per ton of ore ground.

I have made a chart showing the usefulness of the tube mill in stage grinding, whereat advocates of rolls and chilian mills will, of course, be disappointed because, in the first place, I have considered rolls useful only as secondary crushers, and the chilian not at all.

FLOWSHEET SHOWING THE USEFULNESS OF THE TUBE MILL IN STAGE GRINDING



Our primary crusher is a Blake or a gyratory, both equally effective. Our secondary crushers may be stamps, disc crushers, rolls or ball mills. If our object is amalgamating with or without concentrating, and we desire nothing finer than 40-mesh, the stamps alone are sufficient. If we desire a 90-mesh product for amalgamating or concentration, we may regrind with a conical mill, or a cylindrical tube mill of short length and big diameter, in a closed circuit with a classifier. If we desire a 200-mesh reduction for cyanide treatment, we must use a cylindrical tube mill, in circuit, with a classifier.

When we use a disc crusher for secondary reduction for concentrating ore, it must be followed by a conical or a cylindrical tube mill of big diameter and short length, and when for the production of a slime for cyanide treatment, by the cylindrical tube mill.

If we have rolls as secondary crushers, and the object is concentrating, they may be followed by the conical mill, a cylindrical tube mill of medium diameter and short length, and if for 200 reduction, by the cylindrical tube mill.

If ball mills are used as secondary crushers, they may be able to deliver a product sufficiently fine for concentration, but if not, they must be followed, as in the last instance, by a conical or cylindrical mill, and if for an all-sliming process, by the cylindrical tube mill.

From a summing up of my ideas on the subject certain facts may be gathered: First, the stamp mill is still with us for amalgamating and a medium-size product, and in combination with the cylindrical tube mill, for cyanide treatment of a slime. Second, the popularity of the conical mill for a product up to 90-mesh, in competition with a cylindrical tube mill of big diameter and short length in circuit with a classifier. Third, the deserved popularity of the cylindrical tube mill with varying diameter and length according to the size of the feed.[1]

[1] Algernon Del Mar, *Mining World*, Jan. 1, 1916.

CHAPTER I

GENERAL DESCRIPTION

A tube mill is a hollow receptacle made of rivetted steel plates, lined with some hard material to protect the shell from wear. Whether in the form known as a pebble mill or a ball mill it may be made in the shape of a cone, sphere or cylinder or combinations of these three geometrical forms. Referring to Fig. 1,

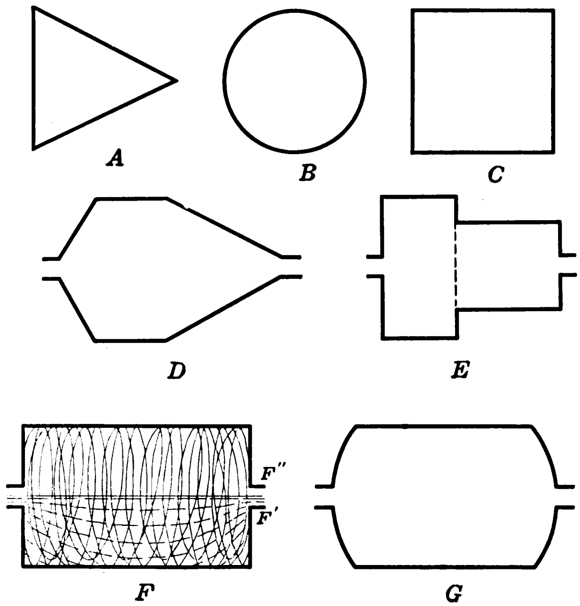


FIG. 1.—Types of tube mills.

A, *B* and *C* represent the three primary forms; *D* a combination of cone and cylinder used by Hardinge in his "conical mills;" *E* a combination of two cylinders of different diameters represented by the Giesecke mill; *F* shows a plain cylindrical mill and if we divide it into two sections by a grating or screen we have the "Compeb" mill; *G* is a combination of sphere and cylinder used more frequently for short rather than for long mills.

By combining a section of a cone and two half circles we have an egg shaped mill which so far has not been manufactured.

The mill is closed at the ends by cast-iron heads, with openings through the center of the castings for the purpose of feeding and discharging the ore. The grinding surface is obtained by lining the mill with silex or iron and filling the mill to or above the center with flint pebbles, mine rock or steel balls. When the machine is rotated at a suitable speed the pebbles (grinders) are carried around until they reach a point near the top when they fall, coming into contact with each other and with the lining, grinding any ore particles which may be at the points of contact.

Although the contact of pebble with pebble and lining is only at the points where spheroid hits spheroid or spheroid hits a curved surface, yet, because there are so many of these pebbles in the mill, there is an immense grinding surface which can be renewed by the addition of more pebbles while the machine is in motion.

The ore is fed into the machine by a device attached to the feed head of the mill and the discharge is automatic. The fineness of the product is governed chiefly by the amount of ore fed to the machine, in other words the fineness depends upon the time the material is in the machine, for the ore travels from the feed to the discharge end by displacement.

When a machine of the character described is made of short length and iron or steel balls are used for grinding instead of pebbles, it is called a ball mill. The mode of operating and the principles involved are the same, but the mill instead of being used for fine grinding is now used to do the work of stamps, rolls and other intermediate crushers.

THE BALL MILL

The advantages to be gained by the use of the ball mill are the small amount of space required for the machine, the lesser first cost as compared with other intermediate crushers or grinders, the ability to grind through a wide range in one machine, the small amount of attention required during operation and the fact that the only repair necessary is to reline the mill, at long intervals. That the amount of attention required during operation is practically nothing accounts for the small labor cost, for the operator who takes care of the mill may look after rolls or other crushers, concentrators or other machinery.

I have been unable to obtain figures that show the ball mill to be an economical crusher for hard quartz ores, its sphere of usefullness being rather for ores of moderate hardness or soft friable ores. In Julian and Smart's "Cyaniding of Gold and Silver Ores," page 193, is given the horsepower per ton of ore crushed in several ball mills.[1]

RESULTS OBTAINED WITH BALL MILLS

Name of plant	Mesh of screen	Tons crushed in 24 hr.	Hp.	Hp. for each ton crushed per day	Nature of material
Commonwealth, Ariz..	40	23.0	12	0.52	Not stated.
Mt. Morgan, Queensl'd	20	23.0	13	0.57	Soft oxidized ore.
Mt. Morgan, Queensl'd	35	19.0	10	0.53	Soft oxidized ore.
Atacama, Chili.......	80	5.5	11	2.00	Compact tough quartz.
Atacama, Chili.......	30	8.0	11	1.38	Blue unoxidized ore.
Associated G. M. of W. A............	30	22.0	15	0.68	Sulphide ore.
Lake View Consols....	40	25.0	15	0.60	Sulphide ore.
Lake View Consols....	40	40.0	25	0.63	Sulphide ore.
Kalgurli gold mines...	35	25.0	15	0.60	Sulphide ore.

Von Bernewitz gives the data for a No. 5 Krupp ball mill at Kalgoorlie as follows: The mill consuming from 16 to 20 hp. crushed 40 tons of ore per day to 27-mesh with a load of 2,350 lb. of steel balls and a consumption of steel amounting to 3½ oz. per ton of ore ground. Converting this into horsepower per ton of ore ground we have Kalgoorlie, 0.40 to 0.50 hp. per ton of ore.

These are examples of ball mills crushing quartz ores through the same range of sizes as the usual stamp mill and for comparison we pick out the gravity stamps doing the best work as follows:

	Hp. per ton of ore
City Deep, South Africa..................	0.20
Rosario, Honduras........................	0.29
Churchill, Nevada........................	0.21
Plymouth Con., California................	0.27

Objections may be made that the data here given for ball mills does not represent present-day practice. In a sense it does not,

[1] "Cyaniding Gold and Silver Ores," H. FORBES JULIAN AND EDGAR SMART, 1904.

for the ball mills now being installed are being fed with a product from a disc crusher or an intermediate crusher between the coarse rock breaker and the ball mill which of course largely increases the output of the latter machine and consequently decreases its apparent power factor.

The Giesecke mill, a combination ball and tube mill was tried on the Rand and at the time it was predicted by enthusiasts that stamps were doomed to extinction, but new stamps have been erected and not ball mills. This mill may be described as a modified tube mill some 24 ft. long divided into two sections (separated by a screen) of approximately one-fourth and three-fourths of its length, the smaller section being 7½ and the longer 6 ft. in diameter. The mill revolved at the rate of 25 r.p.m. and the grinding was done by steel balls 1¼ in. in diameter in the smaller section and 2½ to 4 in. in the longer section. While the mill showed great capacity and the first cost of the mill was comparatively low, yet the cost of operating and metallurgical considerations proved that the stamps in use in South Africa had all the advantage of the ball mill. Mr. H. Stadler says, in regard to the results of the investigations of the Mines Trial Committee of South Africa, that

> "it appears that the merits of the three methods of crushing, single stamp, stamps and tube mills combined, and single tube mill grinding (Giesecke) are so close, as regards the mechanical reduction efficiency, that the final decision in each case depends upon other considerations, such as metallurgical requirements, nature of the ore, local conditions, etc. In any case, the investigation clearly proved that nothing could be gained by the promotion of the tube mill from auxiliary to primary grinders."[1]

Inventors and manufacturers are now trying every variation of shape and every means of screening to attain capacity with the least expenditure of power, so it is possible that from this mass of new trials a machine may be evolved that will be able to stand the test of time. Hardinge has adopted the principle of the sizing action of a revolving cone; Abbé, Fig. 2, places a screen at the discharge end, returning the oversize; Herman, Fig. 3, uses a screen on the periphery of the cylinder; Chalmers and Williams have an open screen on the discharge end of their ball mills, Fig. 4; while the Marcy ball mill, Fig. 5, has an adjustable quick dis-

[1] *Mining and Scientific Press*, March 18, 1916, page 399.

charge somewhat similar to that used on the Chalmers and Williams tube mill.

The Herman mill is particularly suited for dry grinding to a fineness that will readily pass through a screen. The size of the product is determined by the mesh of the screën covering the

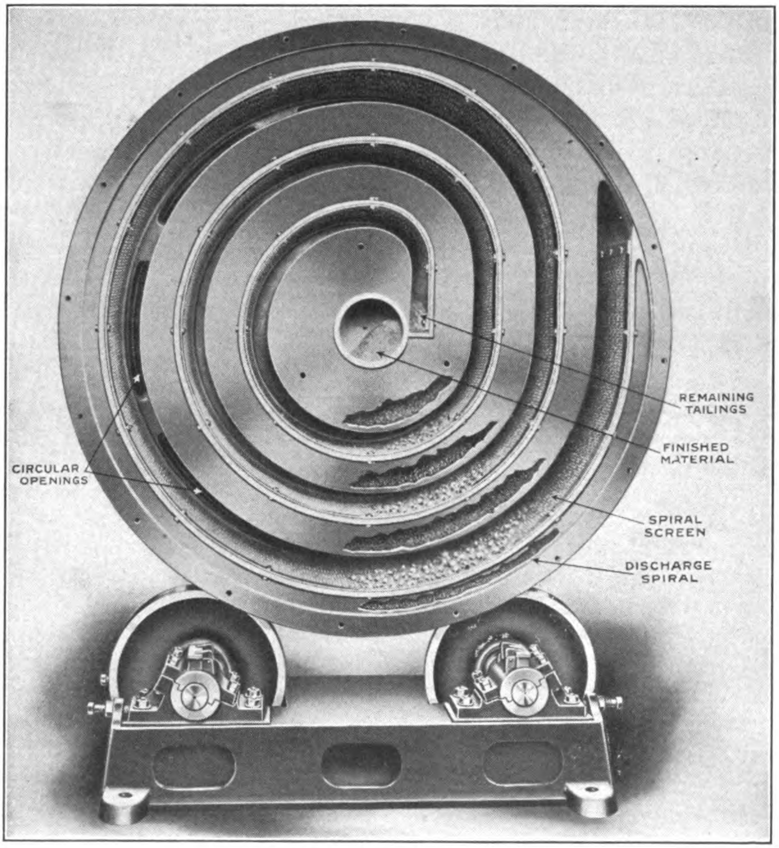


FIG. 2.—Abbé ball mill, inside screen.

slots on the periphery of the cylinder. A mill of this sort will naturally give a product with but little undersize.

The "Compeb" mill, Fig. 6, lately brought out by the Allis-Chalmers Mfg. Co., is a combination mill having two compartments separated by a grating. The mill is made in different sizes for both wet and dry grinding. The size most used for dry grinding is 7 ft. in diameter and 22 ft. long, with a rolled-steel shell and cast-steel heads. The lining is chilled cast iron about $2\frac{1}{2}$ in. thick in the preliminary chamber and $1\frac{1}{2}$ in. thick in the finishing chamber. Forged-steel balls 5 in. in diameter and

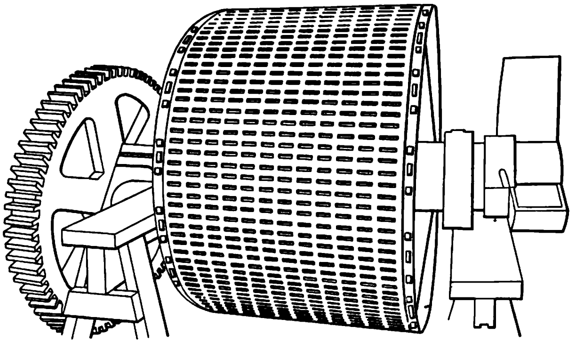


FIG. 3.—Herman screening tube mill.

smaller are used in the preliminary chamber and $\frac{7}{8}$-in. diameter special-alloy iron balls in the finishing chamber. The manufacturers have recently been recommending for use in the finishing

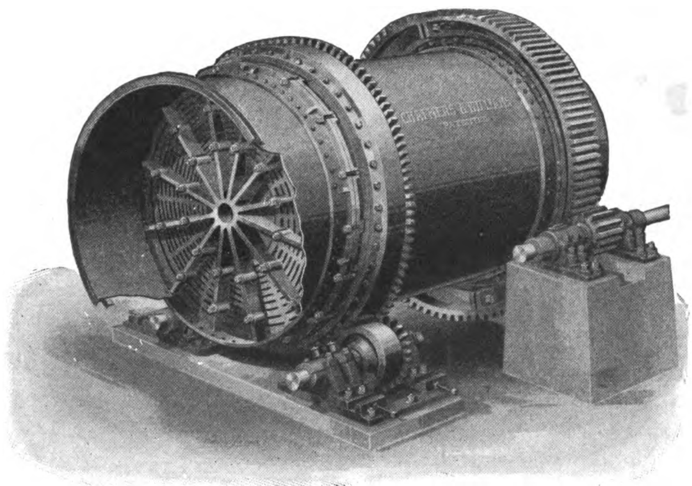


FIG. 4.—Chalmers ball mill.

chamber their special patented shape, the "Concavex." The concavex, as its name implies, has both concave and convex surfaces, and preferably consists of a sphere with two opposite sides spherically recessed, the radii of the recessed surfaces being approximately equal to the radius of the sphere, the object being to attain a mortar and pestle effect.

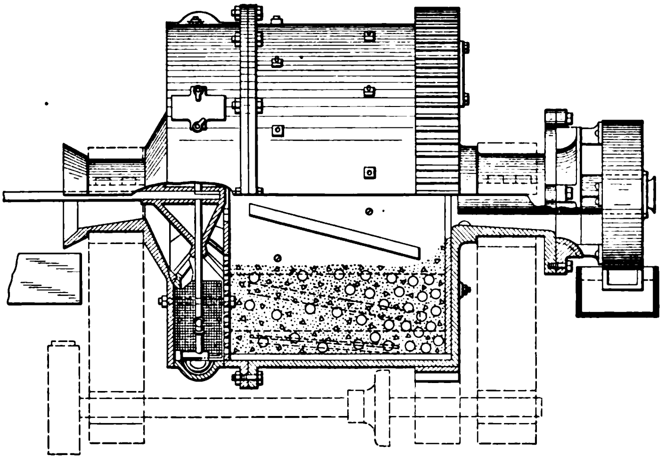


FIG. 5.—Marcy mill.

The charge of balls in the preliminary chamber is maintained by the addition of 5-in. balls. Where concavex are used in the

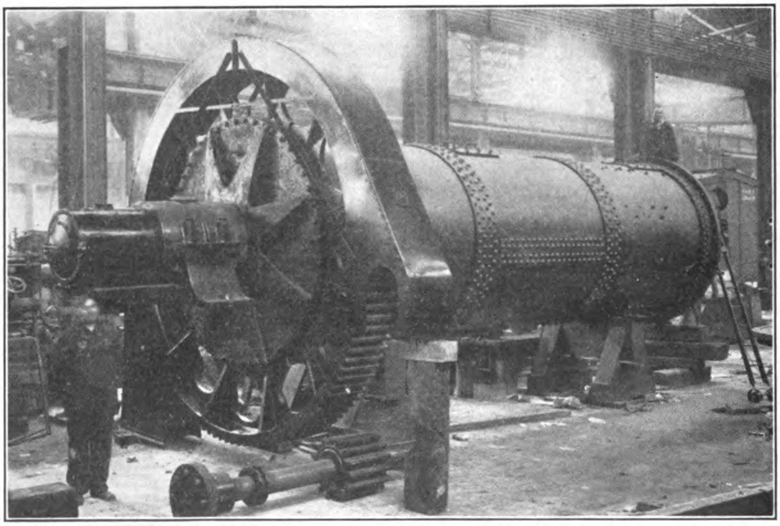


FIG. 6.—"Compeb" ball or pebble mill.

finishing chamber they are generally $1\frac{1}{4}$ in. in diameter. This special shape has the advantage that, when it wears, the concave surfaces tend to flatten slightly so that these surfaces, on concavex which have been reduced in size through wear, fit the convex surface of the full-size concavex.

The ball wear in the preliminary chamber is approximately 75 lb. per 250 tons of cement clinker. The wear of the grinding medium in the finishing chamber is about two-thirds of this amount.

The mill is operated by a direct-connected motor through herringbone gears or plain-cut teeth-spur gears, also by belt drive through cast teeth-spur gears. The fineness of discharge varies with size and quantity of material fed to the mill, but averages 97 per cent. through 100-mesh screen and 81 per cent. through 200-mesh screen.

The mill carries a charge of 40 tons of balls and requires 375 hp. to drive it. Its capacity for grinding cement clinker is stated to be 70 bbl. per hour, or say 330 tons a day.

The following description of the Marcy mill may be of interest:

"The Marcy ball mill is of the drum type, equipped with the usual spiral feeder and loaded with pebbles or balls as may be required. The drum is divided into two parts—the crushing compartment, equipped with lifters to insure the elevation and dropping of the balls to secure maximum crushing efficiency, and a small compartment at the end of the mill, separated from the crushing space by means of a manganese-steel plate and having perforations about ⅛ in. wide. In this end compartment there are radial screens and lifters, the lifter forming the bottom of a box the top of which is formed by the screen itself. These are the finishing screens, and material which will pass through them is considered as being of finished size. The material passing through the screens is diverted by arrangement of the compartment so that it will be discharged through the central orifice; while the oversize—material which remains on the screen—is diverted back into the mill through a proper opening. The screens and lifters form a box, as has been stated, and are placed so that they may be removed from the outside of the mill for rearrangement or cleaning. In order to give the screens maximum efficiency and to increase their life, an arrangement of pipe is made so that water or solution under head can be thrown against them while the screen is passing through the upper half of the circle formed by a cross-section of the mill.

"The action of the lifters, in addition to forming the bottom of the box and assisting in the screening operation, is to maintain a low pulp level in the mill. It is clear that, by constantly removing the solution and pulp from the bottom of the crushing drum, a removal of the material already sufficiently finely ground is accomplished rapidly and no energy is lost in doing work where it is not required. The coarser material is left in a comparatively dry state upon the balls or pebbles, and the crushing and grinding action of the mill can have maximum effect upon it. The accompanying drawing shows a longitudinal view and partial cross-section of the mill, giving a clear idea of its construction and operation."

These illustrations of ball mills are intended to show the relation between the ball and the tube mill of which latter we have two types, the cylindrical tube mill of various lengths and diameters and the Hardinge conical mill. As this latter type of mill has at times been put forward as a competitor of the long cylindrical mill for sliming ores and as our object is to consider the tube mill as an adjunct of the intermediate crusher for further reducing ore, we must either accept or reject the use of the conical mill for this specific purpose.

In a class by itself is the Marathon mill, which is a cylindrical tube mill using steel rods for grinding instead of balls or pebbles.

"The mill contains a charge of these rods sufficient to do grinding work, and the tube is revolved at a speed comparable with that of the tube mill. Grinding is performed by the falling of the pieces of cylindrical steel rods one upon the other, or by their attrition against each other. It is claimed that by the use of rods instead of pebbles or balls, the pulp is subjected to more grinding action, as in contradistinction to the mill charged with pebbles, the contact is over a long line instead of being limited to a point. A test of the Marathon mill has been carried out at the concentrator of the Detroit Copper Mining Co., of Arizona, at Morenci, in which the advantages of the mill have been pointed out. The diameter of the rods used varies from 2 in. to 3/8 in., these various diameters causing them to fit compactly and thus requiring a minimum falling distance to deliver a crushing blow.

"Two tests were made of the Marathon mill, and in the first test the most favorable results were obtained with a pulp which contained 36.7 per cent. solids. In test No. 2 the amount of water and slime in the feed was reduced to a minimum by shovel-wheels to secure a less volume of pulp and a consequent slower velocity of material through the mill. The tonnage in test No. 2 was greatly increased over that in test No. 1. The feed contained an average of 63.5 per cent. solid and only 2.92 per cent. – 200-mesh slime. The speed in all of the tests of the Marathon mill was 30 r.p.m. In a diameter of 3 ft. at 30 r.p.m., there is no dead zone in the periphery of the mill, while excessive slipping of rods is prevented by having the liner plates thicker at each edge than in the middle. This results in a corrugated interior surface, and the lining is said to retain these corrugations until it is worn out. The rods are said to wear evenly from end to end, do not get crosswise in the mill, and are reduced to a diameter of 3/8 in. before they begin to crush, flatten out or break into pieces. When worn to this thinness, some rods roll up and are discharged; the short-length pieces retain their positions in the charge of rods without any serious result, but may reduce crushing efficiency to some extent. It is said to be good practice to take out the entire charge of rods every 10 days, take out the small and disabled rods, and replace the loss in weight with 1¼-in. rods. As soon as the weight of rods consumed per day was determined, their loss in weight was replaced daily with fresh rods. The mill must be closed down to add the daily charge of rods. During the test the Marathon mill was tilted ¼ in. per foot, but later tests showed that it would produce equally as good results when run horizontally.

"In No. 1 test the mill crushed 236.5 tons of dry feed in 24 hr. and in test No. 2 the mill crushed 440 tons of dry feed in 24 hr. The horsepower consumed per hour in test No. 1 was 18.6 and in test No. 2, 22.5. This makes 0.5316 tons of dry feed per horsepower-hour in test No. 1 and 0.8148 tons of dry feed per horsepower-hour in test No. 2."[1]

It has always been the opinion of the careful operator that mills of this character lose efficiency by uneven wearing of rods and the consumption of steel is liable to be excessive for many rods must be discarded before completely worn out.

The ball mill is by no means a new invention and some of the ideas incorporated in the latest machines were tried years ago. Both 6 by 6-ft. and 6 by 8-ft. mills were in use in the '60s, but like other inventions the time had not arrived when the object for which the machine was specially adopted was a factor in metallurgical processes and the use of alloy steels was unknown.

It will perhaps be noted that most of the improvements in facilitating discharge and obtaining capacity pertain particularly to ball mills partly because there is more competition in this class of machinery and partly because a ball mill has a different function than a tube mill in the metallurgy of ores; the ball mill is simply used for grinding while the tube mill may be used not only for grinding but also for obtaining a partial solution of the metals contained in the ore. Where an ore is ground with cyanide solution in a tube mill, the heat generated in the mill, the thorough mixing as the pulp is tumbled over in the mill and its intimate contact with the solvent, give ideal conditions for getting the precious metals in solution and we generally expect about 75 per cent. extraction in the tube mill alone and never under 50 per cent.

[1] *Engineering and Mining Journal,* July 1, 1916.

CHOICE OF CYLINDRICAL OR CONICAL MILLS

The only conical tube mill manufactured is the Hardinge conical mill which is made of two cones joined by a cylinder. The length and diameter of the latter designates the size of the mill; for example, an 8-ft. mill with 18-in. face means that the diameter of the cylindrical portion of the mill is 8 ft. and 18 in. long. Fig. 7 illustrates a modern conical mill, which is one of a series of such mills at the Timber Butte mill, Butte, Montana, while Fig. 8 is the style of conical ball mill being installed at the Inspiration Copper Co. Note how rugged the ball mill is compared to the other. The conical mill is used extensively for regrinding ore already crushed by stamps, rolls, etc., especially

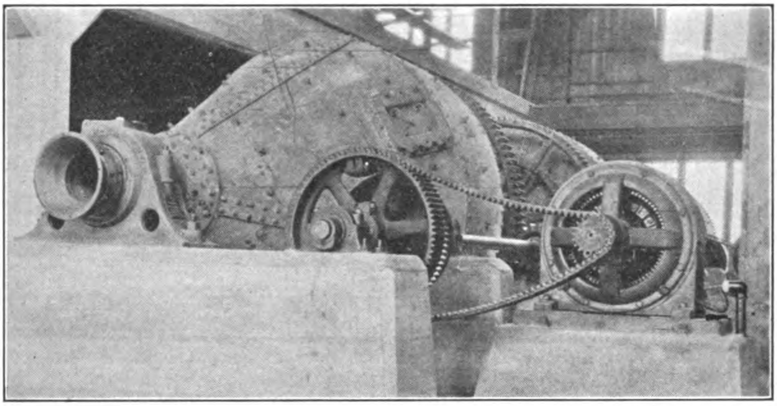


FIG. 7.—Hardinge conical tube mill.

where a granular product is required for concentration, or a medium sized product for flotation.

The cylindrical tube mill illustrated in Fig. 9 is made in any diameter desired between 3 and 10 ft. and practically of any length from 4 to 24 ft., these being the limits of present-day practice.

I consider the conical mill to be most useful for grinding to 100-mesh or thereabout while the cylindrical mill covers this degree of fineness but goes beyond. Therefore, for sliming ores to 200-mesh the cylindrical tube mill must be given preference.

For regrinding a stamp mill product to say 90-mesh the conical mill offers splendid opportunities as it is self-contained and occupies little space. We know the limitation of the stamp mill, that it will not grind economically to the finer meshes; hence if it is desired to increase the capacity of a mill the best plan is to crush coarse in the stamps and regrind in a tube mill. If the

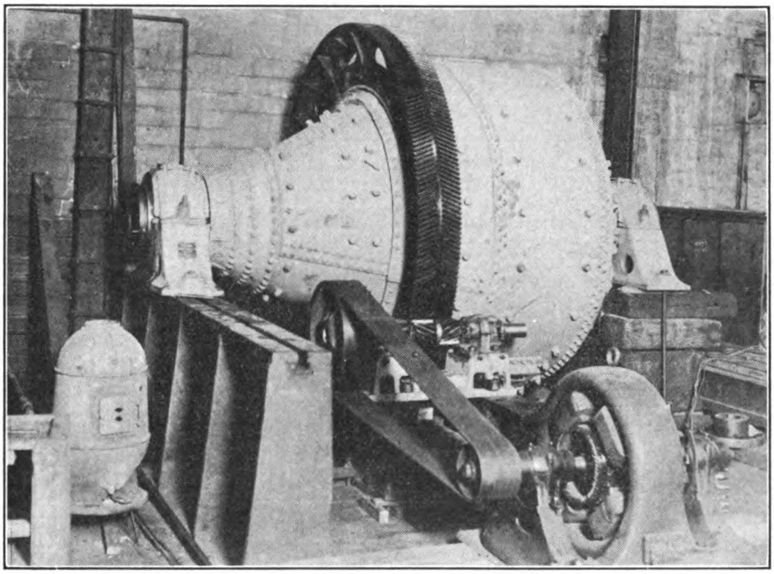


FIG. 8.—Hardinge conical ball mill.

object is to concentrate the ore, the conical mill is a good regrinding machine as it can be made to produce a granular product with little oversize. I would not have it understood that the conical

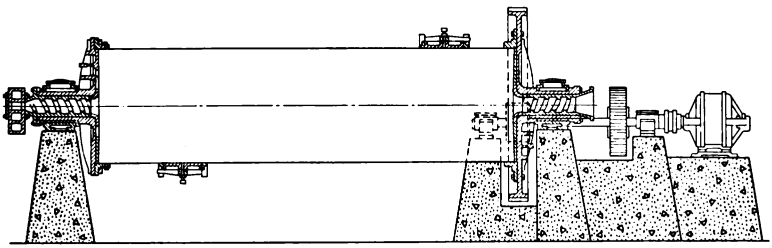


FIG. 9.—Cylindrical tube mill.

mill alone would be more economical than a short cylinder of large diameter in a closed circuit with a classifier for this method of regrinding has been found both economical and efficient.

There is one point we must bear in mind that while the conical mill has been developed to a high degree of utility the cylindrical tube mill makers are only just awakening to the possibilities inherent in a machine of this character.

The following quotation from a paper by George E. Collins on the metallurgical practice at the Mary Murphy mill is given because it contains the experience of other operators as well as my own opinion on the subject.

"The conical end of the Hardinge mill acts, to some extent, as a device to expedite continuous discharge, in much the same way as the device recently brought out by Chalmers and Williams for improving the discharge from a cylindrical tube mill. The conical end of the Hardinge, while of limited efficiency, is simple and positive, and really works. One can obtain nearly as small a proportion of slime in a long tube mill as in a Hardinge, by crowding the feed and using a large quantity of water; but apparently there is then a slight tendency for the discharge to contain still more oversize.

"Mr. Hardinge's original contention that with his mill the weight of blow is automatically adjusted to suit the progressively finer particles of pulp as they pass from head to discharge end, and that owing to this automatic adjustment there is practically no necessity for return of oversize, is, according to our experience, not borne out by the facts. As in the case of the tube mill, if you crowd the tonnage and increase the water sufficiently to obtain a granular product, you have to arrange for the return of a very considerable amount of oversize.

"In various publications Hardinge has illustrated this supposed zonal action in his mill by cuts in which he superimposes on a picture of the conical mill, an outline of several gravity stamps, commencing with one of heavy weight and high drop, and progressively lessening to one of small weight and low drop at the end; the stamps being supposed to represent the force of blow struck by the pebbles in the corresponding stages of the Hardinge mill as you pass from feed to discharge end.

"This is, however, largely a fallacy. It is true that the size of pebbles is automatically graded, to some extent, and obviously the force of blow struck would be thereby lessened. But the peripheral speed is at the same time lessened to such a degree that near the discharge end the pebbles merely slide on one another without cascading, so that there is no crushing action worth mentioning, and what actually occurs is a slight process of abrasion, which is just what we want to avoid. In fact, in comparing the work of the Hardinge with that of the tube mill, according to our experience, it is pretty much a question of relative first cost and operating expense. We have at Romley one 3½ by 16-ft. Gates tube mill, two 6-ft. by 22-in. Hardinge mills, and one 4-ft. by 22-in. Hardinge. We find that the capacity of the tube mill and that of one 6-ft. Hardinge, using the same feed, are almost identical; and the cost of the Hardinge was about 20 per cent. greater. As to power consumption, we have no means of measuring. The Hardinge uses more pebbles, and has to be relined oftener."[1]

This is rather a summary way of disposing of the Hardinge conical mill as a slimer but we know from experience that its sphere of usefulness does not go down so far in the scale as −200-mesh and we must look to the cylindrical tube mill for this work.

The accompanying table of tube mill data of the Homestake mill, South Dakota, compiled from earlier records, when none of the tube mill discharge was returned for further grinding, is interesting as containing a comparison of the work of different size mills. It will be noted that a 5 by 14-ft. tube mill has about the same capacity as a 6 by 6-ft. Hardinge conical mill and consumes slightly less horsepower per ton of ore ground to −100-mesh.[2] Each mill discharges its tailing over an amalgam table, and about 40 cts. is recovered per ton ground.

REGRINDING PLANT DATA

Mill number	1	2	3	4
Manufacturer	Denver Eng. Works	Allis-Chalmers Co.	Allis-Chalmers Co.	Hardinge Conical Mill Co.
Dimensions, feet	5 by 14	5 by 18	5 by 18	6 by 6
Speed, rev. per min.	27.0	28.5	28.0	26.5
Actual horsepower (motor input)	32.0	42.0	42.0	27.0
Fed from	Dewatering cone	Dorr classifier	Dorr classifier	Dewatering cone
Tons fed per day	98.0	138.0	133.0	93.0
Pebbles per ton	1.40	1.24	1.38	1.71
Tons to pass 100-mesh	41.0	52.0	49.0	34.0
Tons to pass 100-mesh per horsepower	1.28	1.24	1.17	1.26

An interesting comparison may be made between a cylindrical tube mill and a conical mill of the same length and major diameter in the case of a discarded 7 by 12-ft. cylindrical mill at the Morning mill of the Federal Mining & Smelting Co., Idaho, which was so changed as to make it into a conical mill. The tran-

[1] *Metallurgical and Chemical Journal,* April, 1914.

[2] ALLAN J. CLARK, *Transactions,* American Institute of Mining Engineers, 1913.

sition was effected by bolting heavy timbers inside the tube mill, forming a cone. Over the timbers steel rails were spiked or bolted to form a lining which resembled the El Oro ribbed type. The capacity of the mill before the change was 98.64 tons per day, while afterward it was 88.2, a reduction of about 10.5 per cent. The horsepower required before the change was 75.6, and afterward, 65, a saving of about 14 per cent. The revolutions of the mill was not changed, remaining at 22½ r.p.m. in each case. Moisture in feed before transition was 58.9 per cent. and after, 57.7 per cent.

The following screen analysis shows the work done by the mill both before and after the transition.

Classification of Work of Tube and Cone Mills

Before change			After change		
Screen	Feed, per cent.	Product, per cent.	Screen	Feed, per cent.	Product, per cent.
+ 10	29.5	....	+ 8	19.10	.....
+ 20	45.5	....	+ 10	22.05	.....
+ 30	12.0	0.5	+ 14	17.05	.....
+ 40	6.0	0.5	+ 20	14.50	0.10
+ 60	4.0	2.5	+ 28	11.80	0.45
+ 80	1.5	5.0	+ 35	7.05	2.25
+ 100	1.5	8.0	+ 48	4.20	4.50
+ 150	....	5.5	+ 65	1.90	7.65
+ 200	....	20.5	+ 100	1.05	10.50
− 200	....	57.5	+ 150	0.60	17.20
.....	....	....	+ 200	0.20	16.00
.....	....	....	− 200	0.35	41.20

Comparing the work done by the two forms of mills we see that the cylindrical mill produced 57.5 per cent. −200 while the conical mill produces 41.20 per cent. through the same mesh. Using the method of comparison explained in another chapter under the heading of "Crushing Efficiencies," we find that the amount of work done before the change is in proportion to that done after the change as 1.3 to 1. If we knew the amount of grinding space occupied by the cone, it might be found that the amount or work done in each case was proportional to the amount of space actually occupied by the pebbles. Considering alone the power factor, that for the cylindrical mill is 1.3 while that for the conical is 1.35. The author states that the power required before the change may be underestimated so that there may not be that difference between the power factors here indicated.

PROPORTIONAL NUMBER OF TUBE MILLS TO STAMPS

The number of tube mills necessary to form a combination for economical stage grinding will depend upon the size at which the feed is delivered to the mills and the required size of the product. If we have heavy stamps crushing a big tonnage to a coarse mesh and this pulp must be slimed, we require more tube mill area than where light stamps are delivering a 20-mesh product to be reground to a 90-mesh screen; so unless we know the conditions of the metallurgical treatment and the physical nature of the ore, it is possible to make only an approximation of the proportion of tube mills to stamps or to other crushing machines.

The following list will show present-day practice:

Mill	Number stamps	Weight, pounds	Tonnage 24 hr.	Battery screen	Number tube mills	Size tube mills, feet
Waihi, New Zealand	90	1,000	160	20	3	$4\frac{3}{4} \times 18$
Hollinger, Canada	40	1,500	384	6	4	5×20
Dome, Canada	40	1,250	...	10	4	5×22
Nipissing Low-grade	40	1,500	268	2, 3	4	6×20
Conecopia, Oregon	20	950	...	8	2	5×20
Rainbow, Oregon	15	1,050	100	4	1	5×22
Liberty Bell, Colorado	80	850	...	12, 14	5	5×22
Motherlode, British Columbia	10	1,250	70	14	1	5×20
Gold Roads, Arizona	40	1,050	360	4	4	5×22
Tom Reed, Arizona	20	1,250	150	12	2	5×22
Montana-Tonopah	40	1,100	145	25	2	5×22
Goldfield Con., Nevada	100	1,050	940	4	6	5×22
West End, Nevada	10	1,250	75	6	2	5×18
Mac Namara, Nevada	10	1,400	75	8, 12	1	5×16
Nevada Hills, Nevada	20	1,250	134	8	2	5×18
Nevada Wonder, Nevada	10	1,400	157	$\frac{3}{8}$	1	5×22
Black Oak, California	20	1,250	100	6, 19	1	5×18
Nevada Colorado, Nevada	10	1,250	60	10, 12	1	5×14

Consulting this table it will be seen that with few exceptions the diameter of the tube mill for sliming gold ores is 5 ft. and that the length varies from 16 to 22 ft.; likewise we see that one tube mill may be used in combination with 10 stamps or with 20 stamps.

In a tabulation based on a comparison of results from 75 stamps and one tube mill versus 75 stamps and four tube mills working under South African conditions, C. O. Smidt shows that the total costs of operating the two plants, when based on pulp of equal fineness, are as 35 to 46 in favor of the larger number of tube mills.

THE POSITION OF THE TUBE MILL

Whether or not we amalgamate after stamping we must fix the position of the tube mill in respect to the stamps, classifiers and plates so that the whole process of extraction may be benefited. The most important modification noticed in stamp milling in consequence of regrinding in tube mills is that the point in the process at which the gold is extracted has changed. In times past the gold was always amalgamated in the mortar of the stamp mill and the plates were immediately in front of the battery. We then saw the plates go to a separate building, still taking the pulp from the stamps; then with the advent of the tube mill in the metallurgical treatment of gold ores the plates were placed in the tube mill circuit and abandoned in front of the battery; then amalgamation was tried in the tube mill and classifier circuits; then we find amalgamation taking place in the tube mill and not in the stamp mortar and then amalgamation was confined to the classifier circuit. Nearly the whole cycle of possible combinations has been tried, and, like many other problems in gold extraction, we must study each particular ore as a distinct problem. The following diagrams, Fig. 10, begin with a free-milling ore and trace the position of the tube mill in various combinations by flowsheets of actual mills now operating and while it does not exhaust the possible combinations, it is sufficiently extensive to show the great variety of ore-reduction processes now used for gold extraction by amalgamation and cyanidation.

A represents an absolutely free-milling process as at the Yellow Aster mill, California, where the whole process of gold extraction is in the battery and on plates.

B shows amalgamating in stamp battery on plates and saving the concentrate, as in the usual California mills.

C notes the first departure from former regular practice as here we have plates in front of the batteries and plates in the tube mill circuit.

D shows the next step where the plates have been taken from the batteries where for so long they occupied an honorable position to be put entirely in the tube mill circuit.

E is a development of the same idea using a Hardinge conical mill to further reduce the stamp mill product, amalgamation taking place in the Hardinge mill and on plates following.

F is a step further showing the plates in the tube mill and classifier circuits.

G shows the plates entirely in the classifier circuit the tube mill being in a closed circuit with a classifier. This flowsheet is applicable for ores that contain gold in a condition where, even with tube mill grinding, the particles of gold are still too coarse for efficient solution in cyanide of potassium.

H shows amalgamation in pans, and really has nothing to do with the tube mill but shows a variation of gold extraction that will no doubt soon pass into oblivion.

We then have various modifications of the all-sliming process, using cyanide as the sole means of gold extraction with the tube mills in closed circuits with classifiers.

I, the tube mill takes the underflow from cone, the overflow from cone and tube mill product going to the classifier where the slimed material is eliminated.

J shows a simple flowsheet with the tube mill in a closed circuit with a classifier. The elevator may be dispensed with by giving the feed scoop of the tube mill a greater radius. The idea is shown in *J*2 which also illustrates the same simple combination feeding the dry product of roll crushing to the tube mill at the feed scoop with the coarse material from the classifier.

K shows a distributing cone feeding a classifier, the elimination of slime being from the classifier in closed circuit with tube mill.

L shows two tube mills and two classifiers working in closed circuits, the slime being separated in both classifiers.

M is a complicated flowsheet with cones, classifiers and tube mills, the slime being separated in the last cone.

N is an example of stage reduction with stamps, Hardinge mill and tube mill where each machine is doing the work for which it is best fitted. We note that a shaking screen precedes the

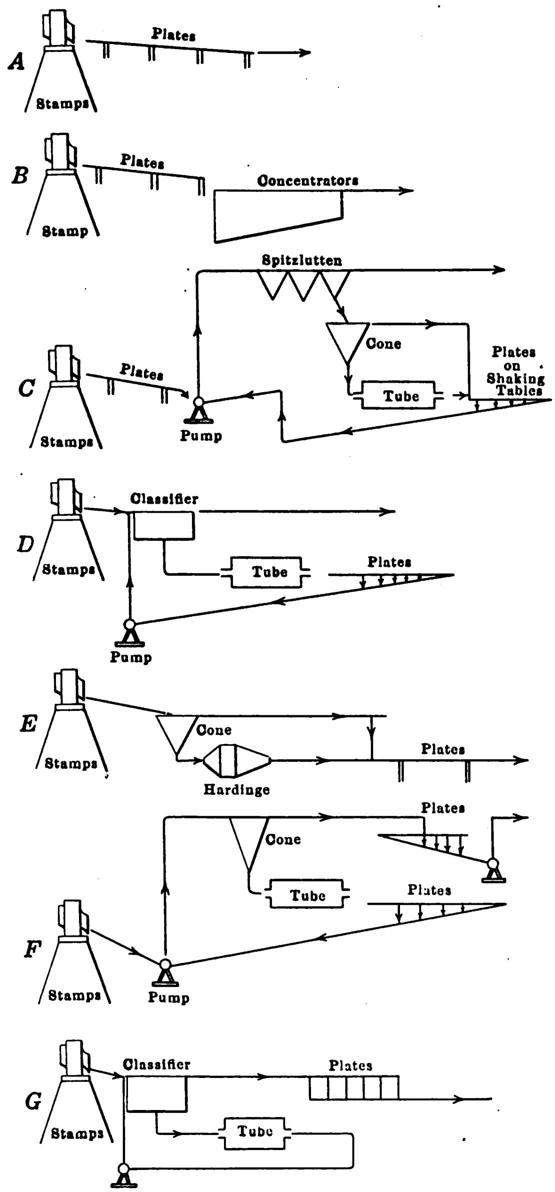
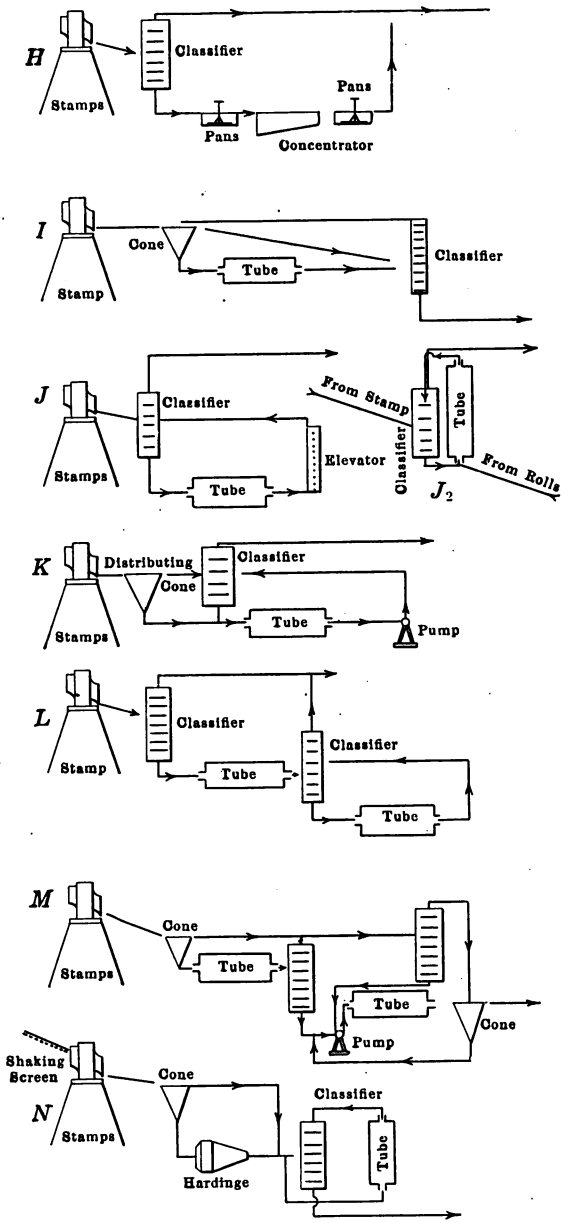


FIG. 10.—Various flowsheets



with and without tube mills.

stamps, thus bypassing the fine material that does not need crushing in the stamp-mill mortar to the classifying cone, the coarse material from cone going to Hardinge mill. From there on the cylindrical tube mill and classifier are in a closed circuit. This is an ideal arrangement for getting the best results from three types of mills, no machine being compelled to do work that could be done better in another machine.

THE ESSENTIAL FACTORS GOVERNING CAPACITY

The amount of ore ground to a particular mesh by a tube mill will depend upon the character of the ore, which is not pertinent to the machine itself and, to the following factors which may be varied according to the work to be done and the cost of material.

I. Diameter and length.
II. Speed of rotation.
III. Size of feed and discharge.
IV. Volume of feed.
V. Amount of moisture in pulp.
VI. Load of pebbles.
VII. Size and character of pebbles.
VIII. Character of the lining.
IX. Character of the discharge opening.

The following table by M. K. Rodgers[1] will illustrate the fact that the amount of ore ground in a tube mill differs with the character of the ore. The realtive crushing duty of Hardinge mills on three different ores, other conditions being the same, shows a remarkable difference in tonnage.

I. Diameter and Length

The most usual diameter of tube mills now in use is 5 ft., few being of greater diameter, but by the nature of the problem of ore grinding we must recognize that the diameter of the mill, which regulates the height or amplitude of fall of the cascading pebbles, should be proportioned to suit the conditions of the feed. If the mill is fed coarse particles of ore the pebbles should fall a greater distance than when grinding fine particles; therefore, the diameter of the mill should be such that the largest-sized pieces of ore fed to the mill are effectively broken by the pebbles. We find this

[1] *Transactions* American Institute Mining Engineers, 1915.

Rock and feed	Mill	Per cent. solids	Hp. of machine	Tons per machine, 24 hr.	Tons per hp.-day	Tons, 200-mesh per hp.-day	Screen scale 4−10−20+40+60+100+200−200							
							4	−10	−20	+40	+60	+100	+200	−200
							Feed							
Calumet & Hecla conglomerate, ¼-mesh.	Conical tube 8 ft. by 30 in.	40	50.0	65	1.30	0.44	0	65	29	5	1			
							Product							
							..	..	..	3	9	13	41	34
							Feed							
Cœur d'Alêne quartzite, F. M. & S. Co., 5 mm.	Conical tube 8 ft. by 22 in., 28 r.p.m.	43	47.5	112	2.50	0.67	0	43	43	11	1	1	1	1
							Product							
							..	..	2	22	19	19	12	27
							Feed							
Miami, ¼-mesh.	Conical tube 8 ft. by 22 in.	27	47.5	229	4.85	1.80	4	45	33	10	2	2	1	7
							Product							
							..	..	2	14	12	17	14	38

Relative crushing duty of 1 hp., due to character of ore only:

Calumet & Hecla conglomerate	1.00
Cœur d' Alêne quartzite	1.92
Miami	3.75

observed to a certain extent in practice for it will be observed that the tube mills taking the product of light stamps and consequently medium-size ore particles are from 4 to 4½ ft. in diameter while those taking the feed of heavy stamps are from 5 to 6 ft. in diameter.

The greatest portion of the ore fed to a tube mill is ground by the impact of pebble against pebble and the lesser portion by attrition when pebble slides on pebble or on the lining. Some operators have contended that the fine grinding is done more particularly by the sliding of the pebbles, that is by attrition, but we know that to attain capacity of −200-mesh material we must cascade the pebbles and not let them slide any more than possible. We must have the mill of such length that most of ore is reduced to the desired fineness by the time it reaches the discharge end of the mill. Should we use a short mill the proportion of oversize would be so great that it would require a complication of tube mills and classifiers to accomplish the work that can be done by one cylindrical mill of suitable length. The finer the ore is crushed the less chance there is of further reduction; hence the length of the mill is proportioned to give the least amount of oversize with the greatest capacity. If all we require is a medium-sized product suitable for concentrating, we use a short tube of large diameter, but for sliming ores the long cylindrical tube is preferred.

At Tonopah, Nev., and South Africa it was found that the usual 22-ft. tube mill may be shortened 4 to 6 ft. without any decreased capacity, which is accounted for by the fact that in this type of mill an excess of length causes overgrinding, useless work being done on ore already ground to the desired fineness. At the El Oro, Mexico, a portion of the end of the mill has been used as a classifying chamber by the use of a perforated diaphragm plate 6 in. from the discharge head of the mill.

Until lately the 8-ft. conical mill has been the standard diameter for this type of mill but lately Hardinge mills with 10-ft. major diameter have been made. No doubt this will be followed by the 10-ft. diameter cylindrical tube mill of short length. The success of the Hardinge mill may be attributed in great part to the great diameter of the cylindrical portion of the mill, this enterprising metallurgist leading the way for the cylindrical-mill manufacturers.

Cylindrical mills are now made 8 ft. in diameter. That illustrated in Fig. 11, made by the Power & Machinery Co., is 12 ft. long and is in use at the Engels Copper & Anaconda Copper Mining Co. The mill is driven by herringbone gears, the pinion shaft being connected to the motor by means of a flexible coupling. A mill 7 ft. in diameter and 10 ft. long overall, with gears 14 in. wide has been installed by the Butte & Superior Copper Co., Butte, Mont. The feed scoop is of the three-cup spiral type so that the mill operating at 22 r.p.m. is fed oftener than once a second. No. 5 Danish pebbles are used. The mill is reported to have an average capacity of 300 tons dry ore per day, of which quantity approximately 75 tons are returned to the mill as a

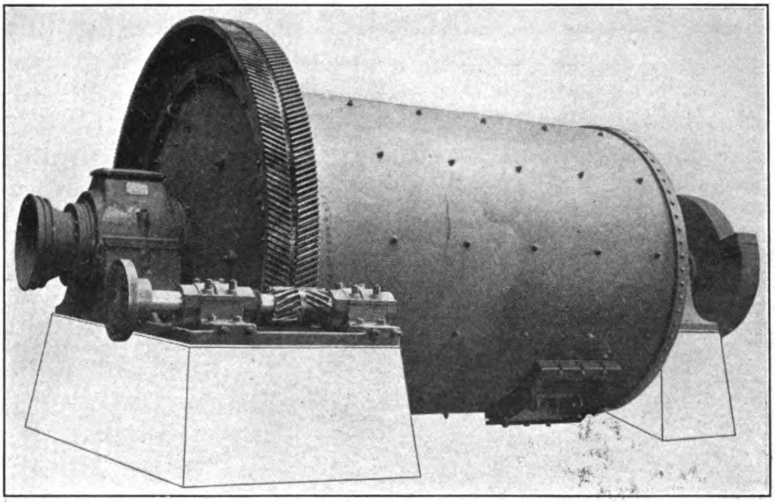


FIG. 11.—The 8-ft. diameter tube mill.

circulating load. This type of mill has proven economical and satisfactory.

In a previous paragraph I have stated that a number of short tube mills with classifiers would unnecessarily complicate a reduction plant and had in mind for comparison a mill 5 or 6 ft. in length. That the 22- or 24-ft. tube mill has been shortened I have also indicated but this contraction is apt to be more radical than would appear at first sight, for from the nature of the problem of ore reduction we can predict that in future a tube mill 8 or 10 ft. in diameter and 6 to 8 ft. long or a ball mill of smaller dimensions will take the place of the intermediate crusher, to be followed by a tube mill 5 to 6 ft. in diameter and 10 to 12 ft.

long, both discharging into the same classifier with the oversize going to the longer mill for final reduction. Several examples of mills of this character follow:

At the Elko Prince mill, Nevada, the rockbreaker product is fed to a 4-ft. Marcy mill using steel balls for grinding. This in turn discharges into a 5 by 14-ft. tube mill. The mill is presumed to have a capacity of 50 tons a day.

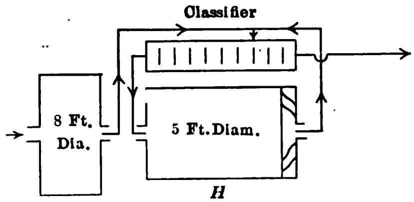


FIG. 12.—Two-stage tube mill flowsheet.

The combinations of two tube mills or a ball mill and a tube mill in circuit with a classifier, of which Fig. 12 is a flowsheet, offer a solution of two-stage grinding with mills of moderate length which have the advantage of decreased weight and the possibility of using roller bearings, thus decreasing the amount of power required to rotate them. The idea embodies sound metallurgical principles

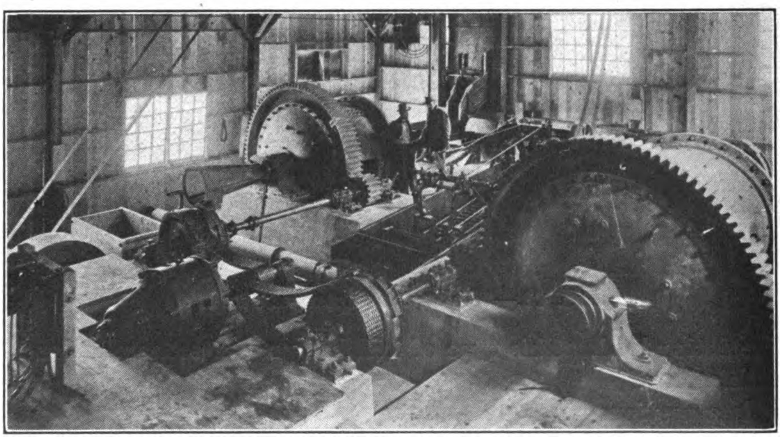


FIG. 13.—Nevada Packard Mines Co. Installation of tube mills.

and as we are liable to see many future mills designed along these lines the installation at the Nevada Packard Mines Co. Fig. 13 is of particular interest. I quote from R. Freitag, the designer and builder:

"We have in operation a tube mill unit composed of two mills, mill No. 1 is 6 ft. in diameter and 5 ft. long. This mill takes the roll crusher feed which has been crushed to pass a 2-mesh No. 12 wire screen. This material is fed dry by means of a sheet slat apron feeder, solution being added in the feed box, and by the agitation caused by the scoop is thoroughly pulped. The discharge is delivered to a Dorr classifier from which the slime is discharged to the first Dorr thickener and the oversize to the second mill which is 6 ft. in diameter and 10 ft. long. The discharge from this mill is delivered to the same Dorr classifier for classification. I must say that this grinding unit is doing wonderful work and it has demonstrated to my satisfaction that the days of the long mills are past. The mills handle a feed of 100 tons easily and we have had a 120-ton feed on the mills without crowding them. We are grinding to 80 per cent. through 200-mesh."

The screen on the discharge end of No. 1 mill was discarded, being unnecessary.

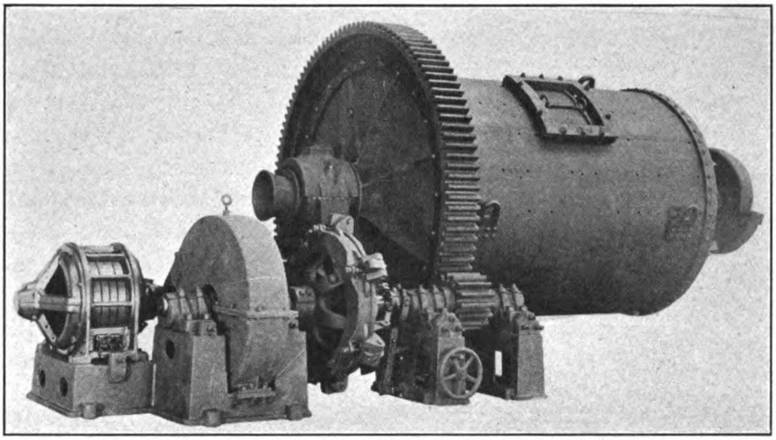


Fig. 14.—The Alaska-Juneau short tube mill.

I understand this is not a hard quartz ore but one well suited for this class of machine.

In the Alaska-Juneau mill we see the initial crushing being done in 8 by 6-ft. ball mills with forged chrome-steel balls and the final grinding in tube mills, 6 ft. diameter and 12 ft. long, which embodies this idea of stage grinding with ball and tube mills. Fig. 14 illustrates this type of short tube mill.

The mill installed by the United Gold Mines Co., near Sumpter, Oregon, late in 1916 contains two tube mills each 5 × 8 ft., the first taking the ore from rolls without screening, the pulp going to a Dorr classifier, while the second tube is in closed circuit with the same classifier. Thus we see the short tube mill gaining popularity. The capacity of these two tubes with a coarse feed of fairly soft ore using local stream pebbles is about 40 tons a day.

At the Santa Gertrudis "tube mill grinding is done in two stages, using 5-ft. diameter mills throughout, the primary series being 16 ft. long and the secondary series 20 ft. and 22 ft. Comparison of this system with single-stage grinding has failed to show conclusive results in its favor. A slight benefit is apparent, but insufficient to warrant a repetition of this refinement of design unless in conjunction with water concentration not required with this ore."[1]

Had the primary series of tube mills been of greater diameter than 5 ft. and of shorter length say a 6×10 ft. mill delivering a coarser product, the advantage of grinding in two stages would have been more apparent. In section III we have shown that to obtain the maximum capacity from a tube mill it is necessary to feed some coarse material with the fines so in any system of two-stage grinding the first mill must either grind rather coarse, or the second mill be fed with some coarse material unless the diameters of the two mills are proportioned to the size of ore they are intended to grind.

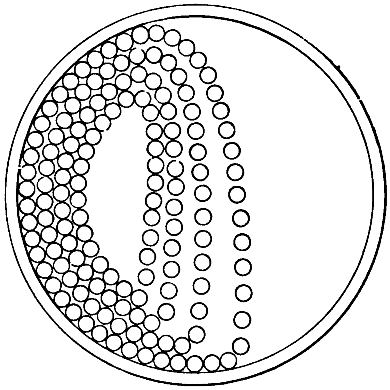


FIG. 15.—Movement of pebbles in tube mill. (H. A. White.)

II. Speed of Rotation

The speed of rotation of a tube mill is such that the pebbles are thrown over the center in a cascade and must not be so fast that the whole mass of pebbles and pulp revolves with the shell of the mill. Fig. 15 shows the theoretical path of the pebbles. This diagram serves the purpose of impressing upon the reader the necessity of grinding by impact for the balls are shown falling over the center in a cascade. It is impossible to obtain this action entirely in a tube mill for there is a constant tendency for the pebbles to skid on the lining and for the pebbles to roll down the incline (of pebbles) before reaching the apex of their ascent.

[1] *Engineering & Mining Journal*, Aug. 5, 1916.

Owing to the fact that the internal diameter becomes greater as the lining wears and that linings of different materials are of different thicknesses, we find mills of the same external diameter revolving at different rates of speed while the peripheral velocity of the inside of the lining may be identical.

The formula most generally used for determining the rate of rotation of a tube mill is that given by Davidsen, namely:

R.p.m. $= \frac{200}{\sqrt{d}}$ where d is the internal diameter of the mill in inches. The factor 200 will pertain more particularly to mills lined with flint pebbles or silex blocks for it has been found that the character of the lining determines to a certain extent the speed of rotation. A smooth liner, whether of steel or iron, whether originally placed in the mill as a smooth liner or an El Oro liner with the ribs worn off, will require a speed somewhat over that required for a rough liner because there is more tendency for the charge of pebbles to slip and the peripheral speed must be greater to lift the pebbles to the apex of the mill. A rough liner with lifting bars which firmly hold the pebbles on the liner without slipping will require a somewhat slower speed of rotation, so we may revise this formula for the classes of liners

r.p.m. smooth liner..............................	$\frac{220}{\sqrt{d}}$
r.p.m. rough liner, such as El Oro or Silex..........	$\frac{210}{\sqrt{d}}$
r.p.m. lifting bar, such as the Komata.............	$\frac{185}{\sqrt{d}}$

That these figures can only be taken as guides is true for we know that every mill and every ore is a distinct problem and what may be satisfactory in one case may be unsuited for another but the figures are sufficiently near to ensure but little chance of failure from not having the speed of the mill at the right point.

The following table shows the rates of rotation for many mills in various parts of the world.

This list of mills shows that the Hardinge mills run at a higher peripheral speed than any of the cylindrical mills and this speed is far above that given by Davidsen's formula. It will be noticed that the 5-ft. cylindrical tube mill is the most common size for regrinding gold ores following stamps, and that at least two of them, the Dome and the Gold Roads, run at a higher pe-

Mill	Size in feet	R.p.m.	Peripheral speed
Hannan Star, W. A.	4 × 16	29.0	364
Porcupine Crown, Canada	4 × 20	31.0	489
Esperanza, Mexico	4 × 20	32.0	402
Nipissing High-grade, Canada	4 × 20	37.0	464
Corinthian North, W. A.	4'1" × 16	31.25	400
Guanajuanto, Mexico	4½ × 20	30.0	424
El Oro, Mexico	4½ × 19	31.0	438
Techatticup, Nevada*	5 × 22	24.0	376
Standard, California	5 × 22	24.0	376
Mac Namara, Nevada	5 × 16	26.0	408
West End Con., Nevada	5 × 18	26.0	408
Black Oak, California	5 × 18	26.0	408
Cornecopia, Oregon	5 × 20	26.0	408
Motherlode, British Columbia	5 × 20	26.0	408
Dos Estrallas, Mexico	5 × 24	26.0	408
Nevada Hills, Nevada	5 × 18	27.0	424
Tonopah Ext., Nevada	5 × 18	27.0	424
Lucky Tiger, Mexico	5 × 22	27.0	424
Alaska Treadwell	5 × 22	27.0	424
Nevada-Colorado, Nevada	5 × 14	28.0	439
Hollinger, Canada	5 × 20	28.0	439
Nevada Wonder	5 × 22	28.0	439
Broken Hills, N. S. W	5 × 10	28.0	439
Cougar, Oregon	5 × 8	29.0	455
Porcupine Crown, Canada	5 × 16	29.0	455
Prospero, Mexico	5 × 14	29.0	455
Rainbow, Oregon	5 × 22	30.0	471
Dome, Canada	6 × 22	32.0	502
Gold Roads, Arizona	5 × 22	32.0	502
Buffalo, Cobalt, Canada	5½ × 22	27.0	466
Rand, South Africa	5½ × 22	28.0	483
Silver Peak, Nevada	6 × 10	24.0	452
Nipissing, Canada	6 × 20	25.0	471
Butte and Superior	7 × 10	22.0	484
Hardinge conical mills			
Winona	6	30.5	574
Commonwealth, Arizona	8	28.0	703
Winona	8	26.3	660
Marcy mill			
Big Four, Utah	4 × 6	32.0	402

* Changed to 28 r.p.m.

ripheral speed than the 7-ft. mill of the Butte and Superior. The best speed for a 5-ft. mill appears to be 28 r.p.m., at all events that is my experience; those revolving at 24 r.p.m. or less do most of the work by the attrition of the sliding pebbles. For ores containing high percentages of heavy sulphides with the values in this heavy material this speed may be justified, but not when capacity is wanted. The pebbles in a 5-ft. mill running at 24 r.p.m. become flat, indicating that the wear is caused by sliding while the same mill running 28 r.p.m. cause the pebbles to be more or less rounded showing that the grinding has been done by impact.

The amount of ore ground in a tube mill (to −200-mesh) depends largely upon its speed of rotation; for example, a 5 by 22-ft. mill revolving at 18 r.p.m. had a capacity of 28 tons a day, while the same mill revolving at 26 r.p.m. ground 48 tons and at from 27 to 28 r.p.m. about 55 tons. There was no means of measuring the amount of power required at the various speeds, or it might have been possible to balance tonnage against power cost but there is no doubt that in this particular instance the increased speed amply repaid in grinding efficiency for the extra power required. A 5 by 22-ft. mill at 28 r.p.m. will require about 55 hp. while the same mill revolving at 32 r.p.m. will require 85 hp. There must therefore be a greatly increased capacity to warrant this increased expenditure of power and, while figures for comparison are not available, judging from the actual mill tonnages in the mills where this speed is maintained the increased cost for power is not justified, unless, contrary to the usual rule, the power cost is an unimportant item of expense.

A. M. Merton[1] says in regard to the speed of cylindrical mills:

"The speeds of from 360 to 400 ft. per minute will cover most cases, and the mean of 380 ft. may be confidently adopted in a new plant. The best speed at which a tube mill should run in a given case may be determined in a very simple way. The method is to operate the mill at various speeds determining the power consumed at each speed. By dividing the power consumed by the number of revolutions a factor is obtained which is really the power absorbed at each revolution. The higher the factor the most effective the crushing power of the pebbles. The following is an example:

[1] *Mining and Engineering World*, June 1, 1914.

R.p.m.	Hp.	Factor	Peripheral vel. ft. per min.
27	54.6	2.022	378
28½	58.0	2.035	400
30½	61.6	2.019	430
32	64.5	2.015	450

"The speed of 28½ r.p.m. gives the highest factor, and would therefore be adopted. However, some power might be saved and the efficiency increased by experimenting a little closer around the peripheral velocity of 400 ft. per minute."

Before applying this method it is necessary first to determine the most economical load of pebbles for the mill, because we may fill the mill above the center with pebbles with a small increase or even less power consumption than when the pebbles are below the center and so do more grinding with a lower factor.

At some mills the operator judges by the temperature of the tube mill discharge whether the mill is grinding properly, a high temperature indicating that the mill is doing good work. While the theory that the amount of heat generated in the mill and absorbed by the pulp might give an indication of the amount of work being done on the ore, it is not a safe theory to bank on. The following notes on the subject are given to indicate the method of calculating and the result of one experiment.

The proper speed of rotation might be determined by the amount of heat generated in the mill if all the heat generated represented useful work. By taking the temperature of the material entering and discharging from the mill we can determine the amount of work represented by these heat units, but we have no means of calculating the amount of heat lost in radiation, friction or sound. These unmeasurable losses of heat will no doubt be fairly constant for a given atmospheric temperature so that if we measure the heat units generated we will no doubt find that the greater the amount of heat generated the greater the grinding efficiency, but not necessarily in the same ratio.

The following data were obtained from two different mills on the same day and while the figures are approximate the result shows that the method may have some merit.

	Mill No. 1	Mill No. 2
Size	5 by 14 ft.	5 by 20 ft.
Speed	28 r.p.m.	21 r.p.m.
Horsepower	35 (estimated)	48 (estimated)
Temperature rise	10°C.	4°C.
Capacity, 200-mesh.	50 tons per day	22 tons per day
Moisture	50 per cent.	50 per cent.

The number of horsepower represented by the amount of heat generated may be found by the following formula:

T = temperature rise of the water, degrees C.

T' = temperature rise of the ore, degrees C.

(If these are fed separately both temperatures should be known.)

P = pounds of water discharged per minute.

P' = pounds of ore discharged per minute.

S = specific heat of water = 1

S' = specific heat of ore = 0.21.

$$\text{Hp.} = \frac{\{(P \times S \times T) + (P' \times S' \times T')\}1{,}402}{33{,}000} \tag{1}$$

Substituting the figures for the two mills we have:

$$\text{Mill No. 1} \quad \frac{\{(35 \times 1 \times 10) + (70 \times 0.21 \times 10)\}1{,}402}{33{,}000} = 21 \text{ hp.}$$

$$\text{Mill No. 2} \quad \frac{\{(15 \times 1 \times 4) + (30 \times 0.21 \times 4)\}1{,}402}{33{,}000} = 3.6 \text{ hp.}$$

The figures for mill No. 1 show that 60 per cent. of the energy put into the mill has been returned as heat in the pulp and only 7.5 per cent. of that put into mill No. 2. Had mill No. 1 been run with less moisture, no doubt as much as 65 per cent. of the energy would have appeared as heat in the pulp. Dowling gives the efficiency of tube mills on the Rand at 57.4 per cent. The lesson to be learned from the fact that mill No. 2 has returned in the pulp but 7½ per cent. of the energy put into the mill is that the mill is running too slow which is likewise shown by the tonnage which should be at least 65 tons per day, instead of 22 tons.

W. J. Pentland says:

"When a tube mill is grinding its full load, a distinct rise of temperature of the shell can be noted. If the pebble load is light, or the moisture content of the pulp so high that the increased temperature is not apparent, the mill is not doing the work it can be made to do."

III. Size of Feed and Discharge

The feed for a tube mill may vary between 2- and 40-mesh and the discharged product contains usually at least 70 per cent.

–200-mesh. The size of ore particles fed to the machine varies with the diameter of the tube mill and the efficiency of the intermediate crushers (such as stamps) to grind to a particular size.

The experiments at El Oro proved that "the efficiency of the tube mill increases with the coarseness of the sand fed to the mill," within limits.

It was found at the West End, Tonopah, that the crushing efficiency could be increased 25 per cent. by the addition of sufficient coarse material to the feed, together with the fines, to permit the coarse material to travel almost to the discharge end of the tube mill. To accomplish this 10 stamps crushed through 10-mesh, and 10 through 6-mesh.

The following list will give an idea of present-day practice:

Mill	Mesh of stamp screen	Size of mill, feet	Capacity		
			Tons	Screen	
				Per cent.	Mesh
Waihi, New Zealand	10	4¾ × 18	77.0	73	–120
Mines of El Oro, Mexico	25	4½ × 19½	100.0	90	–200
San Rafael, Mexico	10, 12	4 × 20	50.0	74.5	–200
Guanajuanto Mexico	26	4½ × 20	75.0		–120
Hollinger, Canada	6	5 × 20	90.0	90	–200
Nipissing Low-grade	2, 3	6 × 20	65.0		
Conecopia, Oregon	2	5 × 20	50.0	86	–200
Liberty Bell, Colorado	12, 14	5 × 22	73.4		–200
Gold Roads, Arizona	4	5 × 22	90.0	87	–150
Montana-Tonopah, Nevada	25	5 × 22	55.0	87	–200
Nevada Hills, Nevada	8	5 × 18	101.0	43.5	–200
			75.0	79	–200
Nevada Wonder, Nevada	⅜	5 × 22	69.4		–200
Big Pine, Nevada	1 in.	5 × 20	130.0	85	–200
Black Oak, California	6, 19	5 × 18	90.0	55	–200
Nevada-Colorado, Nevada	10, 12	5 × 18	60.0	80	–200
Porcupine Crown, Canada	2½	5 × 16 5 × 20	108.0 75.0	47	–200
Nevada Packard, Nevada	2 No. 12 wire	6 × 5 6 × 10	100.0	80	–200

One of the peculiar features of a tube mill is that a reduction in the size of the ore fed to the mill does not give a corresponding increase in the number of tons ground per day. This is accounted for by the fact that when ore particles are ground by the impact of pebble on pebble the spaces between the pebbles allow different size pieces of ore to be ground with the same blow. This is not so apparent when the ore is ground by the rubbing of the pebbles for then the particles at the point of contact only are ground. Referring to Fig. 16, A represents two large-size pebbles with pieces of ore between the surfaces, B a small and a large pebble and C two small pebbles. It can easily be seen that two large pebbles will crush larger grains of ore than one large and one small or two small pebbles, so the larger the pebbles the larger size ore can be fed to the tube mill with the expectation that the capacity will not be seriously reduced. There is a limit to this increase which can be determined only by experiment.

W. J. Pentland[1] relates the following experience to show the increased capacity of stamps and tube mills in combination by increasing the size of the ore particles fed to the tube mills:

"We had 200 stamps, cone classifiers and thickeners, two tube mills 5 by 20 ft. and seven tube mills 4 by 22 ft. We used 35- and 40-mesh screens on the batteries, yielding a product containing about 60 per cent. −200-mesh solids. From 730 to 750 tons could be treated, giving a final pulp of 80 per cent. −200-mesh. By substituting 25-mesh screens on 100 stamps and 30- or 35-mesh on the other hundred, we were able to raise the tonnage from 830 to 850 tons, but this was the most we could get through the tube mills. With cone thickeners we had to give constant attention to keep the moisture down to 38 per cent.

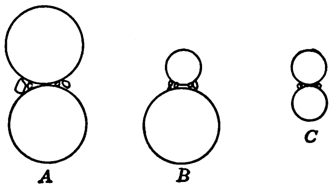


FIG. 16.—Showing how coarse particles of ore are crushed between pebbles of different sizes.

"We then hung up 100 stamps, put 8- and 10-mesh screens on the remaining 100, rearranged and improved the cone system of classification and thickening, and added three tube mills 5 by 24 ft. This system enabled us to increase the capacity to 1,050 tons daily; while battery, crusher and conveyor costs that used to total 35 cts. went down to 17 cts. . . .

"With coarse feeds we could get lower moisture, greater tonnage and better ground pulp; with the finer feeds it was almost impossible to get low moisture, the tonnage was reduced and the product never contained as much −200-mesh solids as was secured from the coarser feeds."

IV. Volume of Feed

Experiments at El Oro proved that the capacity of a tube mill "decreased proportionally to the rate of feed." This is partially true, for the increased capacity due to increased feed has its limits. The capacity of a tube mill increases with the amount of

[1] *Chemical and Metallurgical Journal*, 1914.

feed to a point where the returned oversize is sufficient either to overload the classifier or to increase the circulating load to the extent that the feed must be decreased. Feeding above this point provides the mill with too much coarse material to do efficient work.

V. B. Sherrod found at the Guerrero Mill, Pachuca, "That the grinding efficiency increases with the quantity of ore fed per minute, to a point which varies with the diameter of the mill, and that beyond this point capacity is not gained, and within reasonable limits there is no loss except that due to the return to the mill of an excess of oversize. The limit of profitable feed varies with the character of the pulp, and can be determined only by experiment."

An experiment at the Goldfield Con., Nevada, showed that with a medium feed 0.74 tons of ore were ground per horsepower, while with a heavy feed the same mill ground 0.97 tons per horsepower per day to −200-mesh.

In a case noted the cost of grinding, elevating and classifying with a light feed was $1.88 per ton of initial sand fed. With a medium feed the cost was reduced to 97 cts. and with a heavy feed, the cost dropped to 68 cts. per ton.

When designing a fine-grinding plant it is often necessary to figure the capacity of the classifier return feed which returns the oversize to be reground. This varies in different mills but may be placed at from 75 to 100 per cent. of the initial feed. At the Tonopah-Belmont the initial feed is 67.18 tons while the return feed is 60.20 tons or in the ratio of 0.91 to 1.

R. T. Mishler, assistant general manager of the Tigre Mining Co., Senora, Mexico, made experiments on the effect of varying the amount of ore fed to the tube mills at their plant and being of more than passing interest the entire paper is here reproduced.[1]

"The fine-grinding equipment of the Lucky Tiger plant consists of five 5 by 14-ft. tube mills. They are lined with manganese-steel liners, of a modified El Oro type, which reduce the effective dimensions to 4½ by 13½ ft. The feed ends of the tube mills are equipped with spiral sand feeders, and the discharge end with worm-pebble feeders. Each mill is driven, through gearing, by a separate 60-hp., slip-ring induction motor. There are ammeter and watt-hour meter connections to each motor, making it possible to ascertain the power used by any mill. The speed of the mills is constant at 27 r.p.m.

[1] *Engineering and Mining Journal*, Sept. 12, 1914.

"Danish flint pebbles of the best grade are used, the pebble load being kept fairly constant at 7 tons. This is accomplished by regulating the feed of pebbles by the readings of the ammeter, an amperage of 58 being maintained. With a voltage of 450, and a power factor of 0.75, this indicates a horsepower of 45.5 per mill.

"The tube mills receive from the concentrating plant and from old dumps, an average of 150 dry ton per day of clean, coarse sands. After passing through the tube mills, these sands are raised in bucket elevators, passed over Wilfley tables, and then distributed to four 16-ft. drag classifiers. The sand discharged from these machines is returned to the tube mills, and the slimed overflow is sent to agitation tanks, thus being removed from the tube mill circuit. An average of 80 per cent. of the overflow passes a 200-mesh screen, the remaining 15 per cent. being finer than 150-mesh.

"The experiments for determining the proper tonnage of feed to the tube mills, were made on single mills running under actual working conditions. Special efforts were made to keep the character and dilution of feed constant throughout each series of experiments. Before beginning any test, the mills were run for 90 min. on the same tonnage of feed to be used in the test. Sampling of feed and discharge was then commenced, the sampling continuing 90 min. At the end of this period the volume of feed was changed and the mill again allowed to run 90

TABLE I.—EXAMPLE OF CALCULATION OF WORK DONE BY TUBE MILLS
Tonnage, 45 Tons (Dry Weight) per 24 Hr.

Mesh	Average size of particle, inches	R = reciprocal of size	Feed		Discharge	
			Per cent. of various sizes	Per cent., R = 100 work units	Per cent. of various units	Per cent., R = 100 work units
+20	0.05500	18	0.8	0.1	....	....
+40	0.02450	41	3.6	1.5	....	....
+60	0.01185	84	18.8	15.8	0.4	0.3
+80	0.00775	129	14.2	18.3	2.0	2.6
+100	0.00615	163	15.0	24.4	6.0	9.8
+150	0.00458	218	14.8	32.3	10.0	21.8
+200	0.00330	303	11.2	24.0	11.6	35.1
−200	0.00250	400	21.6	86.4	70.0	280.0

Work units per ton in feed	212.8
Work units per ton in discharge	349.6
Diff. = work done per ton of feed	136.8
Tons × diff. = work done per day	6,156.0

NOTE.—Work unit = work required to grind a ton of inch-size fragments of ore to ½-in. size.

min. before sampling was commenced on the following test. Four times during each sampling period the feed was diverted into a suitable vessel and the flow of wet pulp measured. Knowing the dilution and the specific gravity of the pulp, it was then possible to calculate the tonnage per hour of dry feed. The samples of feed and discharge were retained for moisture and screen analysis, and, together with the tonnage, form the basis for the comparison of efficiency.

"Four series of tests were run. Table II gives the results from the first series, consisting of four tests made on coarse concentrator tailing, with no admixture of return feed. Table III shows the results of four tests made on dump sands, mixed with a small amount of return feed.

TABLE II.—EFFECT OF VARYING TONNAGE IN TUBE MILLS
(Coarse Feed)

Test No.	1		2		3		4	
Tons of dry sand per day	22		47		70		85	
Moisture, L : S ratio	1.1 : 1		0.94 : 1		0.84 : 1		0.90 : 1	
	Feed	Disch,	Feed	Disch,	Feed	Disch,	Feed	Disch,
Per cent. on 20-mesh	11.2	0.3	21.6	0.2	18.4	....	22.4	....
Per cent. on 40-mesh	28.0	0.6	39.6	0.6	45.2	0.4	39.4	0
Per cent. on 60-mesh	25.6	1.4	23.0	0.8	19.6	4.0	20.6	5
Per cent. on 80-mesh	13.4	0.4	3.8	2.2	5.0	4.2	4.2	6
Per cent. on 100-mesh	6.0	1.2	2.4	2.9	2.8	5.4	2.6	6
Per cent. on 150-mesh	4.6	2.4	2.2	5.4	1.8	8.4	1.2	9
Per cent. on 200-mesh	3.6	2.6	1.0	4.9	1.2	6.2	0.8	4
Per cent. through 200-mesh	7.6	91.1	6.4	83.0	6.0	71.4	8.8	66
Work units in feed	113		82		81		820	
Work units in discharge	381		367		341		3210	
Diff. = work done per ton	268		285		260		2390	
Tons × diff. = work done per day	5,896		13,395		18,200		20,310	
Power constant at 47 hp								

Tables IV and V give the results on tests run on a mixture of return feed with a relatively small amount of dump sand.

"In order to reduce the screen analyses and tonnages to terms of grinding work, a modification of the method suggested by Algernon Del Mar[1] has been employed. Table I furnishes an example of this system. The work units in Tables II, III, IV and V were all calculated in the same manner as in Table I, though the calculations have been omitted on account of space; only the final results being given. The work units in feed and discharge are obtained by multiplying the reciprocal of the average size (in inches) of the sand grains corresponding to each mesh, by the proportion (not the percentage) of the sample remain-

[1] *Engineering and Mining Journal,* Dec. 14, 1912.

TABLE III.—EFFECT OF VARYING TONNAGE IN TUBE MILLS
(Mixed Coarse and Fine Feed)

Test No.	1		2		3	
Tons dry sands per day	45		103		145	
L : S	0.44 : 1		0.46 : 1		0.57 : 1	
	Feed	Disch.	Feed	Disch.	Feed	Disch.
Per cent. on 20-mesh	0.8	....	1.0	....	1.0	
Per cent. on 40-mesh	3.6	....	8.4	....	5.4	
Per cent. on 60-mesh	18.8	0.4	18.4	2.2	17.4	4.4
Per cent. on 80-mesh	14.2	2.0	13.6	5.2	18.0	8.2
Per cent. on 100-mesh	15.0	6.0	13.8	9.8	17.0	12.2
Per cent. on 150-mesh	14.8	10.0	14.2	13.0	16.0	16.4
Per cent. on 200-mesh	11.2	11.6	10.2	12.0	10.2	13.0
Per cent. through 200-mesh	21.6	70.0	20.4	57.8	15.0	45.8
Work units in feed	213		203		194	
Work units in discharge	350		320		293	
Diff. = work done per ton	137		117		99	
Tons × diff. = work done per day	6,165		12,051		14,355	
Power constant at 47 hp						

TABLE IV.—EFFECT OF VARYING TONNAGE IN TUBE MILLS
(Fine Feed)

Test No.	1		2		3	
Tons dry feed per day	109		217		570	
L : S	0.47 : 1		0.49 : 1		0.53 : 1	
	Feed	Disch.	Feed	Disch.	Feed	Disch.
Per cent. on 20-mesh	0.52	....	0.76	....	0.42	
Per cent. on 30-mesh	0.08	0.08	2.72	0.28	1.76	0.6
Per cent. on 40-mesh	4.52	0.44	3.40	1.04	3.60	2.5
Per cent. on 60-mesh	6.48	2.96	7.24	4.64	10.92	10.5
Per cent. on 80-mesh	9.44	5.41	7.76	6.68	13.08	11.0
Per cent. on 120-mesh	27.36	19.20	25.40	22.60	29.60	30.5
Per cent. on 150-mesh	30.60	32.96	31.20	34.28	24.86	25.2
Per cent. on 200-mesh	12.44	18.88	13.28	13.92	9.28	9.2
Per cent. through 200-mesh	8.56	20.04	8.24	16.56	6.48	10.2
Work units in feed	218		216		198	
Work units in discharge	264		248		212	
Diff. = work done per ton	46		32		14	
Tons × diff. = work done per day	5,014		6,944		7,980	
Power constant at 45.5 hp						

ing on each mesh. The sum of all these products is the work units in 1 ton of the pulp sampled. The difference between the work units in the discharge and the work units in the feed gives the work units performed upon 1 ton of the pulp in passing through the mill; while this difference multiplied by the tonnage per day gives the total work units performed by the mill in 1 day. The advantage of this system is

TABLE V.—EFFECT OF VARYING TONNAGE IN TUBE MILLS
(Fine Feed)

Test No.	1		2		3		4	
Tons dry feed per day	67		80		239		693	
Moisture, L : S ratio	0.45 : 1		0.44 : 1		0.46 : 1		0.55 : 1	
	Feed	Disch.	Feed	Disch.	Feed	Disch.	Feed	Disch.
Per cent. on 20-mesh	0.20	....	0.32	....	0.51	....	0.99	0.10
Per cent. on 30-mesh	0.76	....	0.96	....	1.08	0.40	2.23	1.10
Per cent. on 40-mesh	1.72	....	2.12	0.40	2.21	1.00	4.52	3.40
Per cent. on 60-mesh	4.40	1.10	7.20	1.80	6.43	4.65	11.48	10.80
Per cent. on 80-mesh	8.56	2.85	12.46	4.25	12.78	9.05	12.78	12.96
Per cent. on 120-mesh	25.76	14.75	28.40	19.00	30.71	27.50	27.98	28.56
Per cent. on 150-mesh	30.20	25.78	27.18	29.30	24.98	27.05	24.92	24.68
Per cent. on 200-mesh	17.90	85.49	12.82	15.10	12.62	14.35	9.59	10.22
Per cent. through 200-mesh	10.50	30.03	8.54	30.15	8.68	16.00	5.51	8.00
Work units in feed	234		216		214		194	
Work units in discharge	293		281		241		205	
Diff. = work done per ton	59		65		27		11	
Tons × diff. = work done per day	3,952		5,200		6,543		7,623	
Power constant at 44.8 hp.								

that it furnishes a perfectly definite unit of grinding work; *i.e.*, the work required to reduce a ton of inch-sized fragments of ore to ½-in. size. In Table I, the work units per ton in the feed are shown to be 212.8. This means that in order to reduce the feed to the size shown by the screen analysis, it has required 212.8 times the amount of work which would be required to reduce a ton of inch-sized fragments of the same ore to ½-in. size. This system has the same disadvantage as all other systems for determining the amount of grinding work done from screen analyses; it disregards the fact that different sands vary in hardness, and it is, therefore, accurate for comparing grinding efficiencies only when the material ground is of the same hardness in all cases compared.

"That there are pronounced variations in the hardness of sands, even in the same plant, is shown by comparing Table II with Tables IV and V. At a given tonnage of feed to the mill, five times as much work is indicated in Table II as in Tables IV and V. Still the horsepower required is practically the same in each case. The explanation of this apparent anomaly is that in the first case the feed consists of comparatively soft fragments of the original ore, while in the last two cases the feed is composed of the harder fragments of the original ore and of particles from the flint pebbles, which were too hard to be pulverized in the first passage through the mill.

"The principal lessons, however, to be gleaned from the tables and diagram, Fig. 17, are: first, that the power consumption is independent of the tonnage of sands fed to the mills; and second, that the proportion of pulverized sands in the discharge is highest for lower tonnages, and decreases as the tonnage is increased. At low tonnages (20 tons per day), coarse sands are slimed in a single passage through the mill—the discharge containing more than 90 per cent. minus 200-mesh product. On the other hand, when tonnages in excess of 200 tons per day are fed to the mill, the discharge is only slightly finer than the feed. The third point is that when we consider tonnage, multiplying the tons passed through the mill by the grinding work done per ton, we see that without a single exception the total grinding work, or 'efficiency,' increases as the tonnage of feed is increased.

"A large feed has the same advantage over a small feed that a short tube mill has over a long one; *i.e.*, the pulp passes oftener through the classifiers, the pulverized sands being promptly removed from the circuit, and the work of the tube mill being expended more upon grinding coarse sands than upon grinding sands already sufficiently fine.

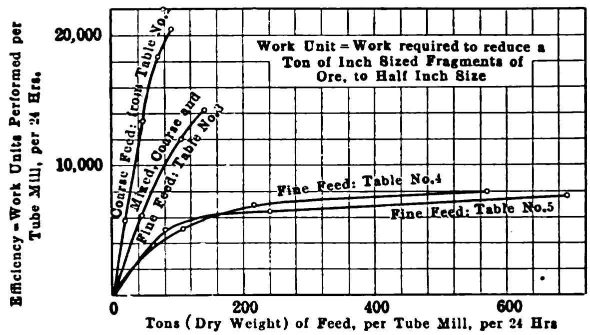


FIG. 17.—Relation of tube mill feed to efficiency.

"Extremely large tonnages, however, have two important drawbacks: first, the gain in efficiency is small; second, the proportion of coarse sands in the discharge is greatly increased. The small gain in efficiency for large tonnages is best shown by reference to the diagram: above a feed of 200 tons of dry sands per day, the curves flatten, indicating practically no gain in efficiency. The increase of coarse sands in the discharge is shown in Tables II, III, IV and V, and is also indicated in Table VI. This last table shows the increase of return feed when the number of tube mills in operation is reduced. As the tonnage is crowded into fewer mills, the amount of return feed increases out of all proportion to the tonnage fed to each mill. This excess of return feed must be elevated, concentrated, classified and returned to the tube mills. The extra cost of performing this work, when the tonnage of feed per mill exceeds 200 tons per day, is easily in excess of the slight reduction in cost due to increased grinding efficiency. Hence, an average daily feed of 200 tons of dry sands per mill has been selected as the tonnage giving best results. This 200-ton feed includes 50 tons per day of concentrator tailings ('initial feed') and 150 tons of sands returned from the classifiers after having passed one or more times through the tube mills.

"When operations were started at the Lucky Tiger plant, all five mills

TABLE VI.—RELATION OF FEED PER MILL TO TONNAGE OF RETURN FEED

Tube mills operating	Tons dry sands per day		
	Feed per mill	Total feed to all mills	Total return feed
5	40	200	50
4	80	320	170
3	200	600	450
2	600	1,200	1,050

NOTE.—In each case the total initial feed of concentrator tailings, passing to the tube mills, was 150 dry tons per 24 hr.

were run as much of the time as possible. It was soon noticed, however, that the mills grinding the return sands were receiving very little feed and one of these mills was stopped. Later, three mills were forced to do all the grinding when the feed was light, and a fourth mill run during heavy feeds. Recently (after completing the above tests), a greatly increased initial feed was crowded into three mills, the fourth mill being run only during exceptionally heavy feeds. By this means the total daily feed (initial and return feed) has been increased to 200 tons per

TABLE VII.—EFFECT UPON PEBBLE CONSUMPTION AND COST PER TON, OF INCREASING FEED TO TUBE MILLS

(From Records at Lucky Tiger Mill)

	Average tonnage of initial sands feed, per tube mill, per 24 hr.	Approximate tonnage of combined initial and return feed, per tube mill, per 24 hr.	Pounds of quarry rock per ton of initial feed	Pounds of imported pebbles per ton of initial feed	Cost U. S. currency per ton of initial sands feed, of tube milling, elevating and classifying.
First 3 months of operation (data not complete).........					
Next 6 months of operation..	26.1	35	10.0	17.9	$1.88
Next 9 months of operation..	40.3	80	none	15.8	1.01
Next 12 months of operation..	42.0	85	none	15.3	0.94
Last 3 months of operation....	50.0	200	none	8.2	0.68

mill, which is the tonnage shown by the tests to give maximum efficiency. This crowding of the feed through the tube mills has resulted in practically doubling the capacity per mill. When five mills were being run, the average daily tonnage of initial sand from the concentrating plant was barely 26 tons per tube mill. Now, with three mills in operation, the average tonnage of initial sands per mill is 50 tons per day. This increase in capacity has occasioned no decrease in the grinding work done, the screen analyses of the final overflow from the classifiers still averaging 80 per cent. through 200-mesh, as it did during the time five mills were used.

"The power required to grind a ton of sands has decreased in inverse ratio as the capacity of the mills has increased. It now requires 22 hp.-hr. to grind a ton of concentrator tailings to such a fineness that 80 per cent. passes a 200-mesh screen. This is half what was required when the mills were run on a light feed.

"The decrease in pebble consumption has been even more noticeable. With a feed of 26 tons of initial sands per day, it was necessary to charge 2,860 lb. of quarry rock and 467 lb. of imported pebbles per day into each mill. This is at the rate of 110 lb. of quarry rock and 17.9 lb. of imported pebbles per ton of sands ground. When the feed per mill was increased to 41 tons of initial sands per day (80 tons per day of combined initial and return feed), the imported pebbles charged to each mill amounted to 640 lb. per day, or 15.5 lb. per ton of initial coarse sands. No quarry rock was used. When the tonnage was further increased to 50 tons of initial sands, or 200 tons of combined initial and return sands per day, the daily consumption of imported pebbles dropped to 410 lb. for each mill, or 8.2 lb. per ton of initial sands fed."

V. The Amount of Moisture in Pulp

The amount of water in the pulp going to a tube mill depends upon the object of grinding. If we want a granular product for concentrating, more moisture is required than for the production of a slime.

While 38.5 to 40 per cent. of moisture has been considered correct for regrinding gold and silver ores treated by the cyanide process, we find in other cases the percentage of water to be as high as 300 per cent. But for the fact that the regrinding of the steam stamp product for copper concentration in the Lake Superior region (1914) by Hardinge mills required as high as 3 of water to 1 of ore, the limit of 300 per cent. would appear facetious, as the extreme percentage for sliming gold ores may be placed at 50 per cent. Walter Neal, using a 5-ft. by 24-ft. tube mill, found that 39 per cent. of water gave the best results for the ore he was treating, any variation from this percentage causing a falling off in the capacity of the mill. The Neal curve, Fig. 18, is here reproduced and shows very clearly the effect of increasing or decreasing the amount of moisture in the pulp from that which gives the best result.

"It is surprising to note what a marked difference in grinding, a comparatively small difference in dilution makes, and it immediately suggests that not enough attention has been paid to this point in the past. To bring the matter home in another way it may be stated that

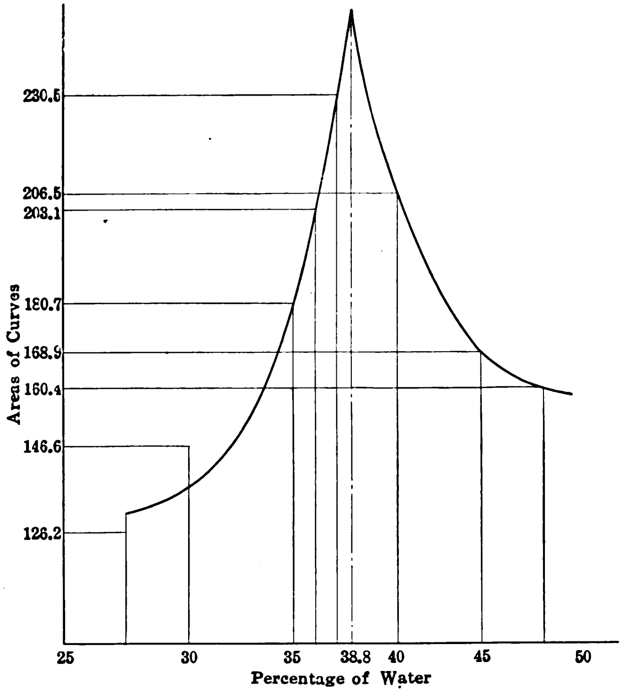


FIG. 18.—Test on best dilution of tube mill feed. (Neal.)

approximately three tube mills working on 39 per cent. pulp will do the work of four mills working on 36 or 48 per cent. pulp.

"It is probable that the 'critical dilution' would vary at different plants on account of different ores and various makes of tube mills, but certainly the matter is worth investigating. Undoubtedly it is more than a coincidence that V. B. Sherrod, experimenting with tube mills at the Guerrero mill, Real del Monte, finds that the grinding efficiency increases with the percentage of solids in the feed up to about 55 or 60 per cent."[1]

[1]*Mining and Scientific Press*, April 2, 1916.

Operators should not be lead into the belief that because Neal found 39 per cent. moisture to be the best dilution for the ore he has under treatment or Sherrod found 40 to 45 per cent. to be the best in his case, that this is the correct amount for any ore. The following table will give an idea of the amount of moisture contained in the pulp at various mills where a tube mill is used for sliming gold ores. It may be that those mills showing a relatively high degree of dilution are unable by reason of mechanical defects in the mill to obtain a thicker pulp. Some ores when ground to −200-mesh appear thick with 40 per cent. moisture while others containing 32 per cent. moisture are more fluid, so that as the pulp must be thick enough for the particles of ore to cling to the pebbles the per cent. of moisture necessary in the ore is a variable quantity according to the physical characteristics of the ore.

Mill	Per cent. of moisture
Techatticup, Nevada	31
Rochester, Nevada	33
Hollinger, Canada	33
West End Con., Nevada	35
Brakpan, S. Africa	35
Porcupine Crown	35
El Oro, Mexico	38–39
Motherlode, British Columbia	38–40
Nipissing Low-grade, Canada	40
Rainbow, Oregon	40
Liberty Bell, Colorado	40
Goldfield Con., Nevada	40
Nevada Hills	40
Corinthian North, W. A.	45
Waihi, New Zealand	50
Nipissing High-grade, Canada	55
Nevada-Colorado, Nevada	55
San Bafael, Mexico	60

The amount of ore fed to the mill influences to a certain extent the best moisture content, it being necessary to retain more moisture in an overfed mill than one that is underfed because in a mill taking a heavy feed the discharge is not so free, causing the mill to be overcrowded with a tendency for masses of pulp to be thrown over the center with the pebbles unless the pulp is thin enough to overcome this tendency. G. O. Smart experimenting with a 5½ by 22-ft. tube mill on the Rand found that 400 tons of sand containing 39 per cent. moisture by weight should be fed per 24 hr. With a small feed, 200 tons of sand per day, 27 per cent. moisture in the pulp was found to give the best grinding.

The most frequent cause of inefficient operation of a tube mill due to an excess of moisture is that the launder from the head of the classifier to the tube mill feed scoop is at such a low angle of inclination that a pulp of sufficient thickness cannot be maintained; the consequence is that the oversize from the classifier increases, requiring more water to wash it down the incline, increasing the trouble by the return of more and more oversize until a point is reached when the ore feed must be turned off until this excess of oversize is ground sufficiently fine to escape from the discharge end of the classifier. With a given inclination the launder will carry a pulp of a certain consistency and to tend to thicken it will result as outlined above. It is important, when designing a mill, that this particular should be given due consideration.

We may either have the oversize from a classifier feed the tube mill at the right consistency as when a wet-crushing machine precedes the classifier, or the oversize from the classifier mixed with dry ore, as for example when the tube mill is fed with the product of dry-crushing rolls. It is evident that in the first instance the pulp must be fed at the right consistency while in the second case the oversize may contain an excess of moisture which is brought to the right consistency by the addition of dry ore. The oversize from a Dorr classifier will contain from 28 to 30 per cent. moisture and this coarse material increased to 36 per cent. moisture will not flow unless the launder is at an inclination of at least 6 in. to 1 ft. It is obvious, then, that the classifier must be placed as close to the tube mill as possible and the feed scoop of the tube mill made of such radius that a launder at this inclination can be installed. If desired the classifier may be in any position near the tube mill and a spiral screw feeder used to convey the sand to the feed scoop of the tube mill. If, as in the second instance, the oversize is fed with dry ore, the inclination of the launder should be at least 4 in. to 1 ft.

We may illustrate this falling off in capacity of a tube mill due to a dilute pulp by approximate figures. If we suppose a mill is grinding 1 ton of ore per hour to −200-mesh with 65 per cent. moisture, it may be possible to increase this capacity 50 per cent. by adding to this feed enough dry ore to bring the percentage of moisture to 40 per cent. or under. If a granular product is desired, however, we must use a greater amount of moisture as this does not promote sliming, the pulp being too dilute to hold the finer particles of ore on the pebbles, so that only the coarser particles are ground.

If, when grinding to a slime, the pulp becomes too dilute and the feed launder is of insufficient inclination to carry the right consistency of pulp, the ore feed should be shut off until the quantity of oversize has been reduced to the point where it will flow to the feed scoop at the right consistency. As a last resort the mill may be stopped, the excess moisture allowed to run out and the mill started again with the pulp at the right degree of dilution. During the time the source of ore supply is cut out the tube mill is not doing its full share of work for there will be no coarse pieces to be ground with the medium-grade oversize, so that it is important at all times to maintain the right moisture content. Constructing engineers often overlook the necessity of placing this feed launder at the best inclination; in fact a great percentage of mills erected have this defect.

If we stand at the discharge end of a tube mill when it is doing effective work, we observe first that the pulp is at a higher temperature than the atmosphere, that the issuing pulp contains but little oversize and no particles have gone through the mill without some reduction and if we stick our fingers in the pulp it will form a smooth coating and will be of such consistency that a drop will hang at the end of the finger but will not be liquid enough to fall off. With a little practice dipping the finger in the pulp the right consistency can be maintained without the necessity of taking the specific gravity.

The right amount of moisture is not the least amount that can be used in the mill but the amount that is found by careful testing to be right, but practically it comes very close to being the least amount of moisture that will form a flowing pulp.

W. J. Pentland says in respect to the amount of moisture in tube mill grinding:

"We think that the pulp should be thick enough to permit the sand particles to cling to the pebbles as they arise with the mill. When the pulp is so thin that no particles cling to the pebbles, poor work is being done, and only the coarse sand is being satisfactorily ground."

It would appear, then, that we must proportion the amount of moisture so that the greatest number of particles of a size larger than that desired in the final product, will cling to the pebbles the pulp occupying the voids in the pebbles being composed of those particles already ground sufficiently to be discharged.

To compare the tonnages at different degrees of dilution we may vary the amount of solution added to the tube mill and take the time required to fill a tub of known capacity. Knowing the specific gravity of the dry pulp, the amount of dry pulp in the

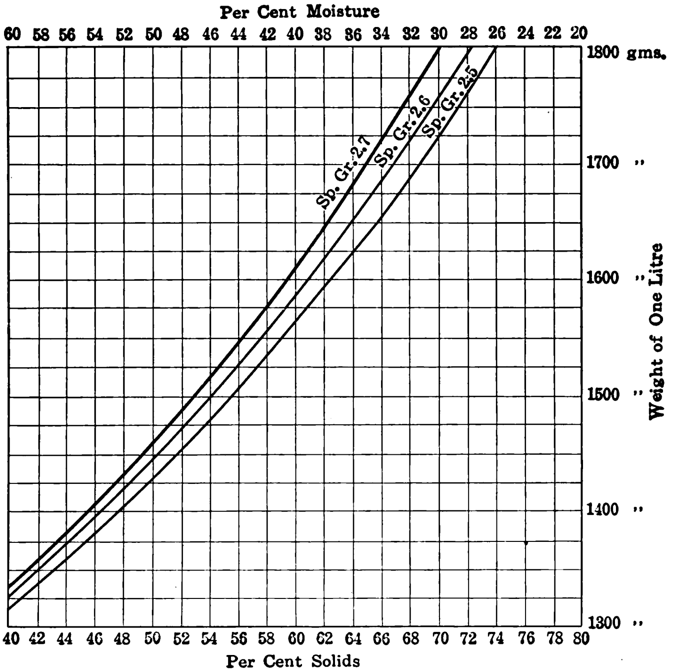


FIG. 19.—Specific-gravity curves.

discharge may be calculated with the help of a curve such as is shown in Fig. 19. To be accurate a sizing test of the dry pulp must be made, for the object is to compare the amount of −200 product with each variation of moisture. An example will suffice to show the method of calculating tonnages with the use of the specific-gravity curve. The specific gravity of the ore is 2.6. Compare the tonnages of −200-mesh product from a mill where it takes 75 sec. to fill a 20-gal. tub with pulp of 1.445 specific gravity and where it takes 100 sec. to fill the same tub with pulp of 1.585 specific gravity. The sizing tests on the two samples showed 75 per cent. −200-mesh for the first and 84 per cent. −200-mesh for the second.

A 20-gal. tub will hold 166 lb. water. If the pulp is 1.445 specific gravity the tub will hold 166 × 1.445 or 240 lb. pulp. Referring to the curve, we find that pulp of 1.445 specific gravity contains 50 per cent. dry pulp, so that in the tub there is 120 lb. dry pulp; 120 lb. in 75 sec. is 69 tons a day and of this 75 per cent. is −200 mesh, or 51 tons. Computing the second filling of the tub in the same way, we find it to be 56 tons a day, so that by decreasing the moisture from 50 to 40 per cent. with the same amount of dry pulp we gain 5 tons −200-mesh material per day.

Specific-gravity Formulas[1]

Let

a = specific gravity of wet pulp.
S = specific gravity of dry slime.
V = total volume of wet pulp.
m = total weight of dry slime in wet pulp.
c = volume of solution in wet pulp.
d = specific gravity of solution.
P = percentage of dry slime in wet pulp.

$$m = \frac{SV(a - d)}{(s - d)}$$

$$S = \frac{m}{(V - c)}$$

$$a = \frac{m + cd}{V}$$

$$P = \frac{100S(a - d)}{a(S - d)}.$$

An approximate value for P where the specific gravity of the solution is taken as 1 is:

$$P = \frac{100S(a - 1)}{a(S - 1)}.$$

From the formula $S = \frac{m}{(V - c)}$ we may obtain the specific gravity of the dry pulp. Take a beaker and balance it, then add

[1] "Pulp Constants," *Engineering and Mining Journal,* Dec. 19, 1914.

a known weight of dry pulp to which add a known weight of water and weigh the wet pulp and note the resultant volume of wet pulp; then by this formula the specific gravity is found. This is a good method of checking results found in the usual way.

VI. The Load of Pebbles

The weight of pebbles in a tube mill affects capacity by providing surfaces of contact on which the ore is ground. We must

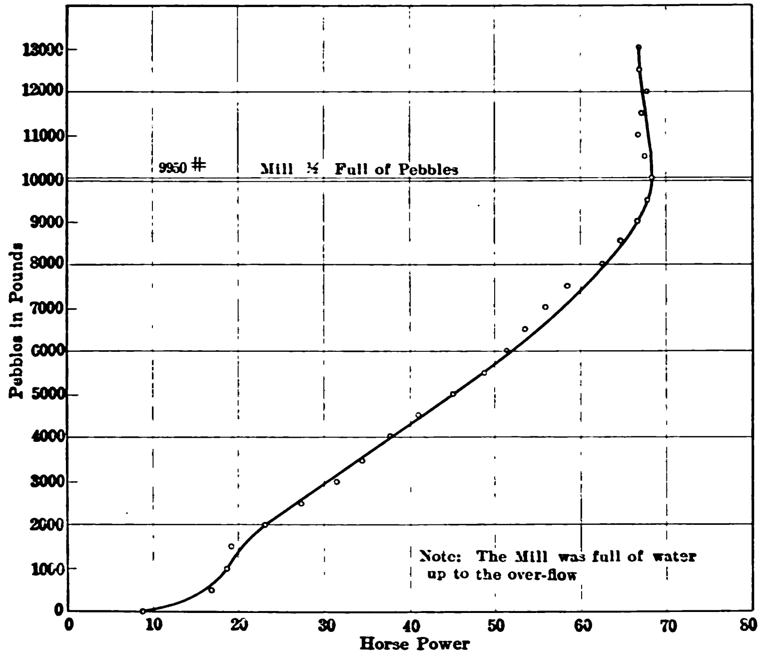


FIG. 20.—Power curve for tube mill. (Seeber.)

not only provide grinding surfaces but we must provide the right amount to obtain the maximum grinding effect with the minimum power. It takes the least amount of power to revolve an empty mill where there is no grinding; as the mill is filled with pebbles the consumption of power increases until the mill is half full; when the pebble load is carried over the center, the power required to revolve the mill increases but slightly until the mill is completely full when there is no grinding action. At some point between the center and full load there is the greatest amount of grinding for the amount of energy put into the mill. This point in the cylindrical tube is usually from 2 to 3 in. above the center although there are cases where the load has at times been kept 6 in. above the center.

H. H. Seeber shows, Fig. 20, that the power required to revolve a Hardinge conical mill increased with the amount of pebbles until the mill was half full with 9,950 lb. of pebbles and from this load up to 13,000 lb. the power required to revolve the mill fell off slightly.

H. A. Megraw[1] states that at the West End Con. mill, Tonopah, charging the mill 6 in. above the center, while the power required increased slightly, the tonnage increased from 100 to 150 tons a day, thereby allowing a coarser screen on the stamp batteries.

The following table will be of use for determining the weight of pebbles in a cylindrical mill. Divide the figure under the internal diameter of the mill by 22 and multiply by the length of the particular mill, in feet, to obtain the weight of pebbles in tons.

WEIGHT OF TUBE MILL PEBBLE LOADS[2]

Weight of Load in Tons of Pebbles (at 105 Lb. per Cubic Foot)

Pebble load in 22-ft. tube mill	Internal diameter of tube mill lining									
	54 in.	55 in.	56 in.	57 in.	58 in.	59 in.	60 in.	61 in.	62 in.	63 in.
6 in. above axis of mill	11.76	12.15	12.55	12.95	13.36	13.78	14.20	14.63	15.07	15.51
5 in. above axis of mill	11.33	11.72	12.11	12.50	12.90	13.31	13.73	14.15	14.58	15.01
4 in. above axis of mill	10.90	11.28	11.66	12.05	12.44	12.84	13.25	13.66	14.08	14.51
3 in. above axis of mill	10.47	10.79	11.21	11.59	11.98	12.37	12.77	13.17	13.59	14.00
2 in. above axis of mill	10.04	10.40	10.76	11.14	11.51	11.90	12.29	12.68	13.09	13.50
1 in. above axis of mill	9.60	9.96	10.31	10.68	11.05	11.42	11.80	12.19	12.59	12.99
Level with axis of mill (*i.e.*, half full)	9.18	9.53	9.88	10.23	10.60	10.96	11.34	11.72	12.11	12.50
1 in. below axis of mill	8.77	9.10	9.45	9.79	10.14	10.51	10.88	11.25	11.63	12.01
2 in. below axis of mill	8.33	8.66	9.00	9.33	9.69	10.03	10.39	10.76	11.13	11.50
3 in. below axis of mill	7.90	8.27	8.55	8.88	9.22	9.56	9.91	10 27	10.63	11.00

VII. SIZE AND CHARACTER OF THE PEBBLES

The size of the pebbles for use in a tube mill should bear some relation to the diameter of the mill, for presumably a mill of large diameter will be used for grinding large pieces of ore, and

[1] "Details of Cyanide Practice," H. A. MEGRAW, 1914.

[2] W. A. CALDECOTT, *Journal of the Chemical and Metallurgical Mining Society of South Africa.*

would then require large pebbles, the limit of size being a 7-in. pebble, while pulp coming from the average stamp mill with a 10- or 20-mesh screen would require a tube mill of medium diameter with 3 to 4-in. pebbles. While the maximum grinding effect is no doubt secured by small pebbles which give the most grinding surface per ton of pebbles, we must adapt the size to the character of the work. A soft ore or one easily reduced in its smaller sizes should be ground with pebbles 2½ to 3½ in. in diameter, while an ore that offers more resistance to being broken will take larger pebbles.

In the order of their efficiency as grinders in tube mills we have Danish, French and Newfoundland flint pebbles. In a class by themselves we have flints from Texas, pebbles from Manhattan, Nev., pebbles from the Pacific Coast beaches, quartz pebbles from mountain streams, rhyolite and basaltic lava blocks and mine rock.

The Danish pebble is dark gray in color, hard and free from fracture planes which makes it an ideal grinder; the French pebble is of lighter color and wears faster, while the Newfoundland pebble is more liable to break than either of the two mentioned due to minute cracks or planes of fracture. The dark gray pebbles appear to wear better than those of lighter hue, whether they be Danish or French.

Pebbles, to give the maximum efficiency, should be round, and where irregular pieces of local quartz or other rock is used in lieu of flint pebbles considerable work must be done on them in the mill before they become good grinders. The small pieces broken off in this process are too small for grinders and too large for being ground effectively, causing increased wear on the liners and increased power consumption. When local quartz is used, the sharp edges should be broken off or the quartz should be revolved in a roughing machine which then leaves the rock in better condition for grinding. While flint pebbles are used in most mills for the reasons stated, in other mills the use of mine rock has been found economical. It is more a question of freight rates than grinding efficiency. At the El Tigre mill, Mexico, the consumption of Danish pebbles was about one-tenth of the amount of mine rock, while the wear on the liners was less and a better product was obtained by the use of the former. At the Purisima Grande mill, Mexico, hard pieces from the mine about 4 in. in diameter were used consuming about 1½ tons a day.

At the Santa Gertrudis "flint pebbles were used for a considerable time, but their increasing cost led to the adoption of mine rock entirely to replace the pebbles. The mine-rock supply is obtained mechanically in the crushing plant and is sent separately over the regular conveyors to a compartment in the battery bin, from which it is transferred by chute to the primary tube mill floor, where it is distributed by car. Part of the rock is introduced into the tube mill through the feeder. As the trunnion opening of the mills is not as large as it should be, rocks over 5 in. size as well as occasionally large boulders, 12 to 15 in., required in the primary mills for efficient grinding, are loaded into the mills through the manholes once a day; 130 lb. of mine rock is required for each ton of ore milled and is credited to the total tonnage treated."[1]

The economical grinding point is taken at 75 per cent. through 200-mesh. The following table is interesting as the grinding is done with mine rock and at a plant with a capacity of 1000 tons daily.

FEED AND DISCHARGE OF PRIMARY AND SECONDARY TUBE MILLS

Mesh	Primary feed %	Tube mill discharge %	Secondary feed %	Tube mill discharge %
On 4	13.9	0.2	0.8	1.6
8	22.8	4.5	0.5	0.1
10	8.6	2.2	0.4	0.2
20	20.6	8.5	1.6	0.2
30	11.8	10.9	3.4	0.7
40	4.7	4.8	3.3	1.0
60	8.0	12.1	17.5	10.8
80	1.9	10.2	9.6	5.6
100	1.6	5.2	11.4	10.3
120	1.5	6.5	17.6	15.5
150	0.7	4.0	9.5	11.8
200	0.8	5.7	8.4	11.2
Through 200	2.5	25.0	15.6	30.7
	99.4	99.8	99.6	99.7

Moisture 35 to 40 per cent. Operating without return, 175 tons of ore pass through tube mill per 24 hr. Operating with return, in closed circuit, 200 tons of ore per 24 hr.

[1] *Engineering and Mining Journal*, Aug. 5, 1916.

At the Santa Gertrudis mill, Mexico, tests were recently made comparing the use of cast-iron balls in the tube mills with the mine rock generally used. "Results thus far obtained indicate a capacity increase of 33 per cent. with finer grinding. Power load shows an increase of 33 per cent., from 65 to 90 hp. per mill. Forged steel balls ordinarily used for such grinding were not obtainable, but it is probable that chilled cast-iron or semi-steel balls and liners will prove more economical taking into account the low cost of locally made castings (2.5 cents per lb.) as against high first cost plus importation expense of steel balls. Ball wear is 1.7 lb. per ton milled.[1]

H. E. West, in the *Mining Journal*, July 31, 1909, gives a series of figures showing the economy of using mine rock instead of imported pebbles in the tube mills at El Oro, Mexico. Using flint pebbles, the consumption was about 8 tons per month for Krupp No. 3, 14 tons for No. 4, and 20 tons for No. 5. With the nine mills in operation this is equivalent to 90 tons per month. At present, with mine rock 40 tons per day is consumed, or 1,200 tons per month (100 tons are maintained in the mills varying from 8 to 16 tons in quartz); in other words, the amount of mine rock used is 13 times by weight the amount of imported flints. This is roughly 5 per cent. of the monthly mill run. The pebbles cost 60 pesos per ton, or 5,400 pesos per month. The mine rock averages $6 gold per ton, the increase in the bullion return contributed by the crushed rock being about $6,000 gold per month. Allowing the cost of mining and handling the rock to equal the value of the gold extracted, it is apparent that the cost of the pebbles, or 5,400 pesos is saved per month.

Manganese or chrome-steel balls offer the best substitute for flint pebbles and from present indications the art of tube milling will in the future be more influenced by the use of this kind of grinder than from any other innovation. Steel being from two and one-half to three times the weight of flint, a mill, to hold the same load of balls, must be made of greater massiveness and strength to withstand this increased weight. If a 4-in. flint pebble with a 5-ft. diameter mill has been found to be an effective grinder for a given ore, we may with the same diameter mill use a 1-in. steel ball with the same effective pressure between the ball surfaces, with this difference that the larger surfaces will retain larger pieces of ore to be ground, so that theoretically a finer product must be

[1] *Engineering and Mining Journal*, Aug. 5, 1916.

fed the mill; this, however, is offset to a great extent by the increased amount of grinding surface by reason of the greater number of balls of lesser diameter. Just how far practice will agree with theory and at what point the compromise between diameter, area and weight will be reached can be indicated only very roughly at present. With the increased weight of tube mills due to the use of steel grinders we may expect a decreased diameter and length, that is, instead of our mills being increased in diameter to 8 or 10 ft. we may expect them to remain at 4 to 5 ft. That a mill of this character will be used to any extent, where the cyanide process is a necessary feature, is doubtful on account of the amount of undesirable fine iron or steel added to the pulp, but for other grinding schemes this objection may not exist. In several places this transition of pebble to ball mill is taking place. Mr. Seeber informs me that at the Winona mill, Michigan, by the use of manganoid-steel balls (and liners) in place of pebbles a greater power ratio has been obtained and the mill capacity increased. He uses a mixture of 7/8-in., 1 1/4-in., 2-in., 2 1/2-in. and 3-in. diameter balls. The increased capacity has amounted to approximately 15 per cent. with a consumption of 125 to 150 lb. of steel balls instead of 250 to 300 lb. of pebbles.

When it was decided to remodel the Anaconda concentrating plant, it was not known whether pebbles or steel balls should be used for grinding. To provide for this uncertainty a compromise was effected. The mills were made 10 by 4 ft. and built sufficiently strong for steel balls in case balls were used. Each mill was equipped with a 225-hp. motor directly connected through a flexible coupling. The mill filled with pebbles takes from 95 to 115 hp. to operate. In case steel balls were used it was planned to put in a false wood lining back of the steel lining in the cylindrical part of the mill to reduce the effective diameter of the mill.

This latter plan was finally adopted, and the Hardinge mills will be equipped with the false wood lining, 15 in. thick, in the cylindrical part of the mill, and a Cascade steel lining. With this form of lining, the mill is virtually 7 1/2 by 6 ft. and requires about 225 hp. when loaded with steel balls.

Fig. 21 gives the details of the lining. This lining was designed by the American Manganese Steel Co.

Jay A. Carpenter in describing the ore treatment at the West End mill, Tonopah, Nev.[1] says:

[1] *Mining and Scientific Press*, Aug. 5, 1916.

"Six tons of manganoid-steel balls were substituted for the 6-ton load of Danish pebbles in the 5 by 15-ft. tube mill. There was a sharp in-

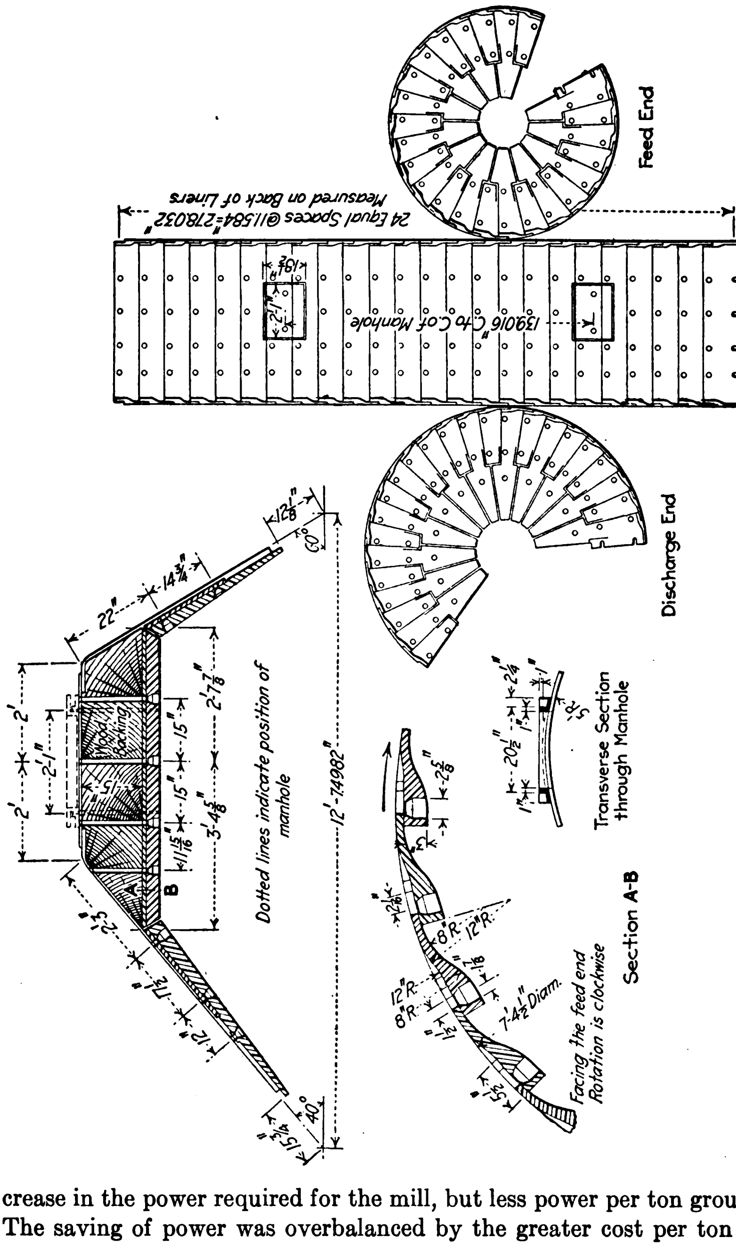


FIG. 21.—Details of wood backing and lining in the hardinge mills.

crease in the power required for the mill, but less power per ton ground. The saving of power was overbalanced by the greater cost per ton for the steel balls. Later, the mill was reduced to 3-ft. diameter, and charged with steel balls, resulting in a considerable increase of tonnage and decrease in the power required for the mill over the use of Danish pebbles in the 5-ft. mill. On account of a saving of 33 per cent. in power per ton ground, the test is being continued over a long period to determine the consumption of steel balls under the favorable conditions of the 3-ft. diameter mill."

With the increased use of steel balls in the tube mill inventors have given us every form of grinder that they could think of. We have dumb-bells and dumb-bells with rings around the shank; the concave balls already described; then we have short and long cylinders of steel; and next we will probably have the hollow steel ball offering less weight with greater grinding surface than the solid balls.

Mill	Grinders	Size	Pounds per ton ground	Cost per ton ground
Lluvia de Oro	Mine rock	6–10 in.		
Liberty Bell	Flint			
Cornecopia	Local quartzite	4 in.		
Yuanmi	Flint	.....	2.60	$0.0500
Dos Estrallas	Quartz	.....	15.00	
El Oro	Danish flint	.....	5.00	0.0920
Lucky Tiger	Danish flint	.....	2.20	0.0550
Hollinger	Danish flint	.....	2.00	
Nipissing low-grade	Flint	.....	3.82	0.0321
Rainbow	Mine rock	.....	30.00	
Motherlode	Danish flint	.....	4.48	0.0800
Gold Roads	Danish flint	.....	6.00	
Montana-Tonopah	Danish flint	.....	2.22	
Goldfield Con.	Danish flint	.....	5.39	0.0880
West End Con.	Danish flint	.....	4.40	0.1210
	Local, Manhattan	.....	7–8	
Nevada Hills	Danish flint	.....	3.00	0.0450
Nevada Wonder	Danish flint	.....	3.39	0.0800
Black Oak	Danish flint	.....	4.00	0.0780
North Star	Quartz	.....	40.00	
Alaska Treadwell	Danish flint	3 in.	3.00	0.1776
Big Pine, Nevada	Local, "Maris"	.....	2.50	
Broken Hills, N. S. W.	Flint	.....	1½–2	
Porcupine Crown	French flint	No. 4	6.00	
Nevada Packard	Danish flint	No. 4	2.50 + mine rock	
Techatticup	Danish flint	4 in.	3.20	

The preceding table will give an idea of the character and cost per ton of the grinding surfaces used at various mills.

VIII. Character of the Lining

The material of which a lining is made affects the capacity of a tube mill by retarding or promoting the cascade of the pebbles; it likewise determines, to some extent, the constituents of the ore that receive the most grinding by performing the greatest grinding effect at the bottom of the mill where sulphides accumulate or nearer the surface where the lighter material is found. The liner which promotes the tumbling over or cascading of the pebbles causes grinding by impact while a liner that allows the slipping and sliding action of the pebbles grinds to a greater extent by attrition. In the first class we have silex blocks, all forms of liners with lifting bars such as the Komata and liners with a pebble surface such as the El Oro; in the second class we have the smooth liners whether made of cast iron, wrought iron, soft steel or manganese steel.

If we are grinding an ore in which a great proportion of the metallic contents is in the heavy sulphides we desire to use a liner that will assist the grinding of these heavier portions. The smooth liners which allow the bottom layers of pebbles to slip on the lining will grind these heavy sulphides which naturally seek the lowest level in the mill. This at least is the theory prevailing in the Tonopah district and has influenced their choice of this class of liner in several instances. This particular ore may act in this way but with a pulp containing 32 to 35 per cent. moisture with other ores the sulphides do not separate out from the mass of material even if the pulp is at rest and when the pulp is thrown about in the mill there is even less chance of this separation. There can be no doubt that when capacity is desired the liner should be such that it will assist the lifting of the pebbles.

The silex lining is made with silex blocks laid in cement. The blocks are usually 4 in. thick laid on edge and last from 7 to 11 months. The advantage of the silex lining is that it does not introduce objectionable material into the cyanide process while the main objection to its use is that the mill must be idle for at least 6 days while a new liner is being cemented in the mill. Steel or iron liners can be replaced in less than 12 hr., so that the lost time is a serious item, particularly in small installations with but one tube mill. Many mills have changed or are being changed from silex to some form of steel or iron lining.

Silex blocks weigh approximately 46 lb. per square foot, the cost of lining a mill varying as follows:

When the Goldfield Con. used silex lining they lasted 7 months and were renewed with the following items of expense:

LABOR		
Removing and replacing manhole, removing end liners	$11.88	
Removing pebbles	3.75	
Removing old lining	11.25	
Relining	63.76	
Replacing pebbles	7.50	
Total labor		$98.14
SUPPLIES		
Cast end liners	$92.48	
Silex, 17,710 lb. at 2.634 cts. per pound	466.48	
31 sacks cement at $1.10 per hundredweight	34.10	
Total supplies		$593.06
Total cost		$691.20

The time involved is: Hours relining, 68; hours setting cement, 72; total hours lost, 140. While this is less time than is customary at other plants to allow the cement to set, there has never been any trouble on account of starting too soon.

The cost for lining seven 5 by 22-ft. tube mills with silex at the Montana-Tonopah mill, Nevada, is given by A. H. Jones[1] as follows:

Labor at $4.50 a day	$42.25
Cement, 28 sacks at $1.035	28.89
Silex, 6 tons at $42.80	256.80
	$327.94 for each mill

The average cost of renewing silex linings on the Rand is given at $525 and at the Nipissing mill at $625. The price of cast-steel or cast-iron linings for a 5 by 20-ft. mill will vary from $650 to $1,000,[2] so that as far as the cost of lining alone is

[1] "Details of Cyanide Practice," MEGRAW.

[2] War cost double this.

in question the silex is cheaper but as already mentioned the time lost during the relining must be taken into account.

The El Oro lining developed at the plants of the El Oro Mining and Railway Co., Mexico, consists of plates, usually of hard iron, having ribs lengthwise of the shell of such form as will cause pebbles of suitable size to become firmly wedged between them,

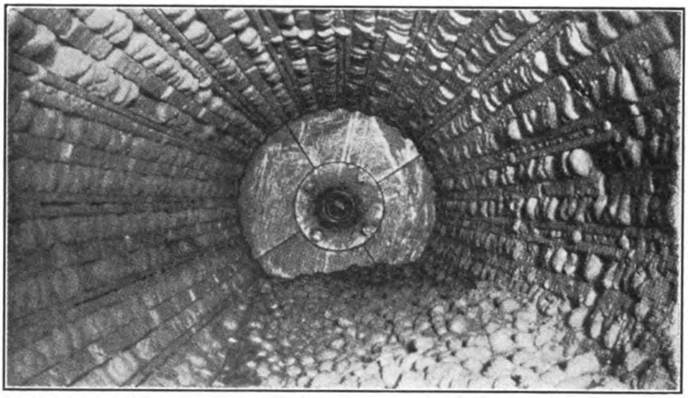


Fig. 22.—Interior of tube mill with El Oro lining.

thus throwing the wear on the pebbles. These liners in sections may be put in the mill without bolts and worn until the iron is so thin that it falls out. In 12 hr. a new lining can be put in place. In operation, when a pebble becomes fractured or worn and escapes from the corrugation, another will take its place and the effect is to maintain what is practically a flint lining to the mill. The rough surface presented entirely avoids the tendency of the whole charge to skid on the lining. Fig. 22 shows the inside of a tube mill with El Oro lining; while Fig. 23 shows the dimension of the lining at the Lluvia de Oro mill.

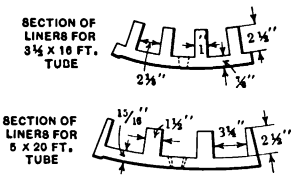


Fig. 23.—El Oro lining at the Lluvia de Oro mill.

The El Oro lining illustrated in Figs. 24 to 27 is made by the Stearns Roger Mfg. Co. and is held in place with keys without being bolted to the shell of the mill. The method of placing this liner is here described:

The liners are held in place by means of taper metal keys *D*. A number of different widths of keys are provided so that the one fitting the space may be selected. These keys must be driven home tight so that they will not shake loose after the mill goes into operation.

Commence lining at each end of the mill and work toward the center. Place the flange side of the lining next to the head so

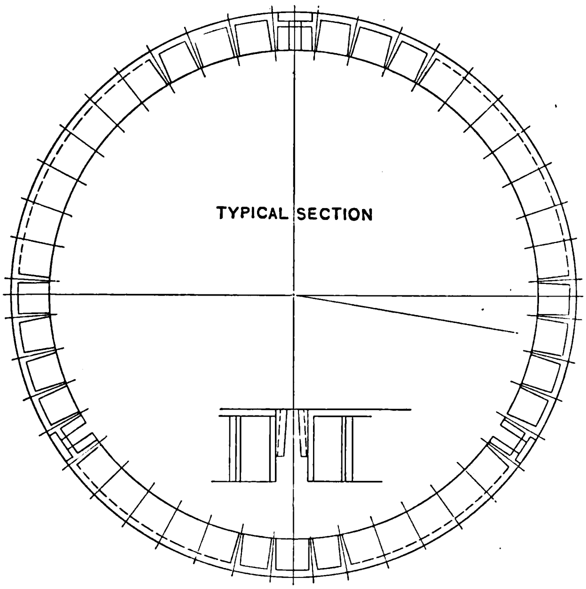


FIG. 24.—Typical section of El Oro lining wedged in mill without bolts through the shell. (S. R. Mfg. Co.)

that the teeth face toward the center of the mill. This will bring two sections near the center with the teeth facing each other.

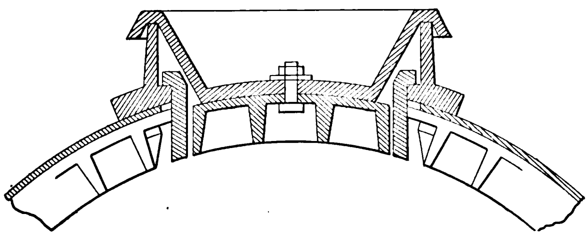


FIG. 25.—Section at manhole.

Each ring of lining is intended to occupy a space of 6 in. in the length of the mill. To allow for inequalities and roughness in the casting some clearance is allowed. This clearance should be distributed throughout the length of the mill so that when you come to the closing section the space will be correct. It will be convenient to divide the shell lengthwise into 6-in. spaces with a chalk and tape line.

The closing joint can be made near the center of the mill where the sections come with the teeth facing: See detail of "closing joint." The space *CC* admits short keys *A* and *B* which can be driven to place by means of a bar and sledge. After the keys *A* and *B* are driven, the space *C* is filled by filing one of the short key blocks to fit the place and driving it between the teeth. This will prevent the keys from backing out.

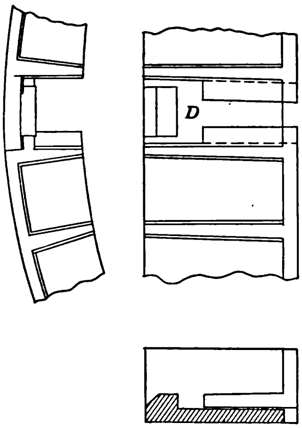


FIG. 26.—Details of joint.

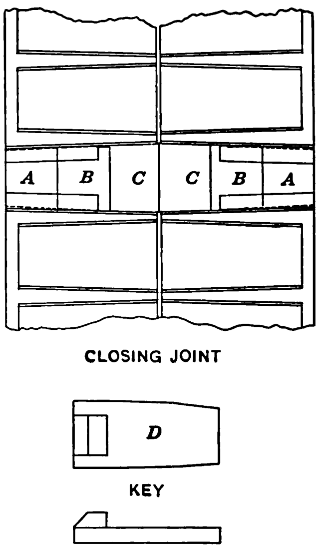


FIG. 27.—Details of closing joint with a key.

When starting the mill with the El Oro lining, a liberal load of assorted pebbles not larger than 3 in. should be used. The mill should be revolved for about 2 hr. with water and pebbles, using no ore so that the pebbles will have a chance to be imbedded in the meshes of the lining. The regular service is then started.

The Globe lining, Fig. 28, is an adoption of the El Oro idea with the ribs transversal instead of longitudinal and is used in several mills in Ontario, Canada.

The Osborne, liner Fig. 29, used extensively on the Rand, South Africa, is another modification of the El Oro lining.

"It consists essentially of two bars placed in such a manner that with the aid of cement concrete they lock themselves in the interior of the tube mill. The horizontal bar is about 2 in. long and about ¾ in. wide. The other bar, at right angles to the first, is practically of the same section as a grizzly bar, and is 4 in. long and ¾ in. in thickness at the end projecting into the mill, and 1½ in. in thickness at the base. Concrete is laid on the bottom and part of the sides of the tube mill, with the bars placed in position, as shown in the sketch."

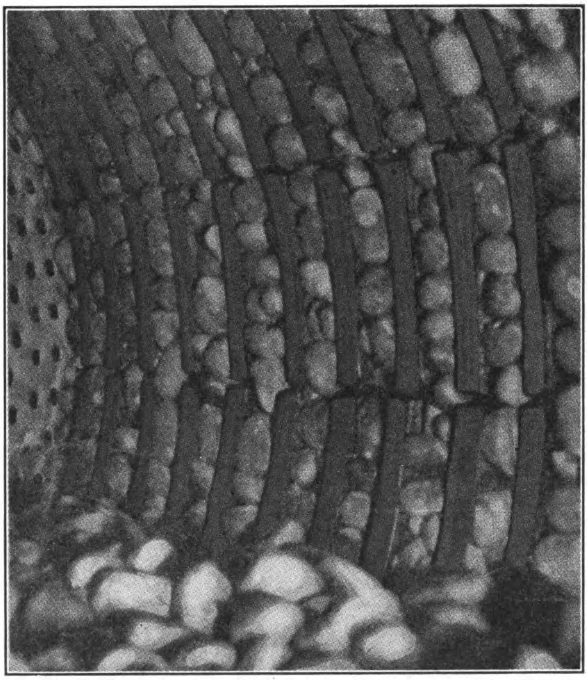


FIG. 28.—Globe liner.

At the El Tigre mill Forbes type of liners have been found to give better satisfaction than the El Oro liners. The former consists merely of grating plates of chilled iron or manganese steel, which are backed by mild steel plates 3/16 in. in thickness. "In this way the advantage of a lining in which the pebbles wedge is retained while the backing pieces do not have to be thrown away when the ribs are worn down, with the result that there is a better efficiency in the wear of the iron."

Closely allied to the El Oro lining is that used in the cylindrical portion of the Hardinge mills at Miami. It will be seen from the cut, Fig. 30, that this lining resembles the El Oro but in addition has a lifter bar, which projects about 2½ in. above the cast-iron ribs and takes much of the wear at the same time providing efficient means for lifting the pebbles without fear of the load sliding.

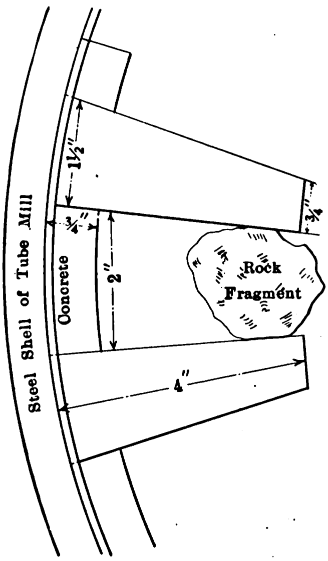


FIG. 29.—The Osborne liner.

The Komata lining, Fig. 31, has the advantage that it alters but slightly as regards thickness during its life, it is easily put in place and has shown in some mills a greater grinding efficiency than other liners used. F. C. Brown says:

"I lay the greatest stress upon the movement of the pebbles, my experience being the more movement the more grinding; hence the advantage of the Komata lining, as this permits of no 'dead' place in the mass of pebbles and ore."

From the cut it will be seen that the lining consists of longitudinal angle bars or ribs about 18 in. apart with plates between. The bars lift the pebbles with-

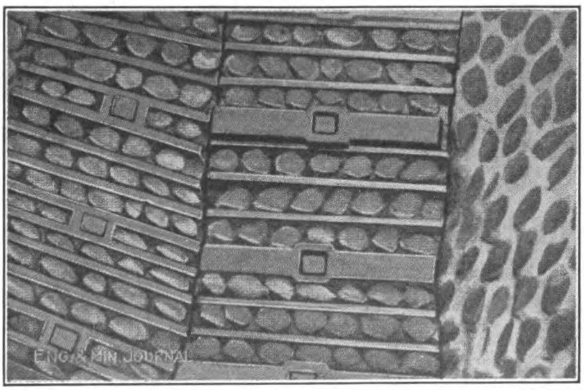


FIG. 30.—Maimi lining for Hardinge mills.

out any back-slip, thus avoiding waste of power. It can be made of cast iron, carbon steel or special alloy steel. At the Waihi Grand Junction mill in New Zealand a set of liner plates 1 in. thick at the center tapering to ⅜ in. at the edges lasted 75½ weeks, and the angle bars 60½ weeks, before requiring renewal, grinding 76 tons of ore per day. By reason of the decreased amount of space required for this liner the interior of the mill is of larger diameter than when silex is used, more pebbles may be used in the mill with greater grinding effect and the mill may be run slower to compensate for this increased diameter. There may be an increased power consumption even with the slower speed but the amount of grinding will more than compensate for this.

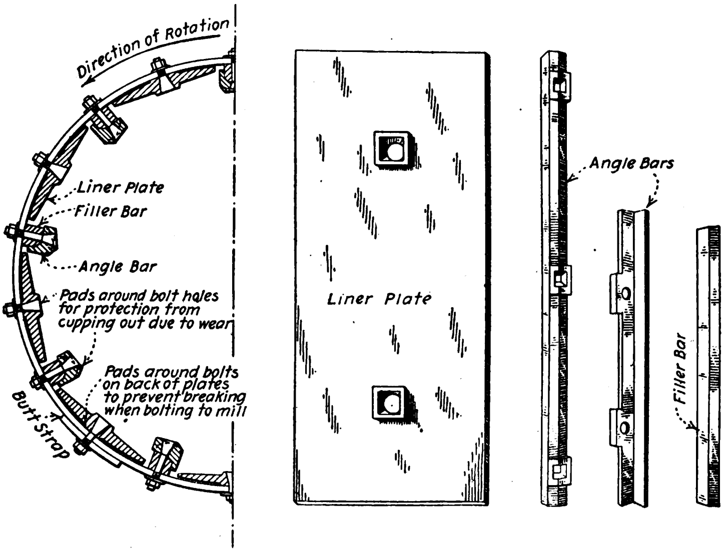


FIG. 31.—The Komata lining.

Smooth liners of carbon or manganese steel have found little favor except perhaps in the Tonopah district of Nevada where better results have been claimed for smooth liners on account of the advantage of being able to grind preferentially the heavy sulphides in the ore. At the Extension mill, one tube mill is fitted with ribbed liners for coarse grinding and another with smooth liners for fine grinding.

All-pebble-and-cement liners have been tried and are now used in Hardinge mills at the discharge and feed ends of the cones, Fig. 30. The disadvantage of this lining is that the cement wears away from the pebbles and when the pebbles are over half worn through they drop out requiring frequent renewals and consequent waste of time while the cement is setting. If this class of lining is used, the pebbles should be carefully laid in the cement to present a regular surface and not thrown in, as in concrete.

Fig. 32 shows a cemented flint lining which was finished in four shifts by one man with two helpers. The pebbles were carefully laid in cement mortar made of half sand and half cement with the long axis of the pebbles vertical. The illustration shows the work on the third day with the unfinished lower segment extending the whole length of the mill. After 10 hr. set the cemented flints clung to the shell of the mill without the need of braces of any sort. The mill was run 7 days after the

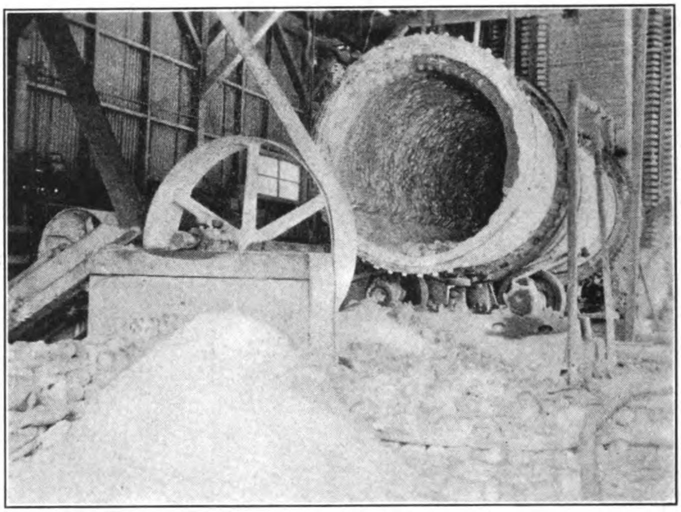


FIG. 32.—Interior of mill being lined with cemented-in pebbles.

cement work was started but could have been rotated a day earlier.

A pebble lining for a 5 by 22-ft. tube mill will cost about $200 and may last 7 to 8 months. It is the cheapest form of lining used, in first cost. To increase the life of this class of liner chilled iron bars may be cemented in with the pebbles, the bars being laid longitudinally with the mill and spaced 1 ft. apart. The iron now takes most of the wear and we have a lining retaining the advantages of lifting bars but with less weight and expense than the all-metal linings.

For those who desire to experiment with concrete pebble linings I have devised a method which I believe will overcome, at least in part, the objections put forward. The pebbles are laid in cement in cast-iron frames open at back and front. These frames are put in the mill like so many bricks or sections of El Oro lining and while laid in cement for greater safety the mill may be used at once as the frames are interlocked and form a complete circle inside the shell. The pebbles can be cemented in the frames any time before use. Fig. 33 illustrates the idea. A liner of this type was used by Barry in West Australia but was found to be of a temporary nature and was discarded in favor of metal liners.

"A new system of tube mill lining has been devised and put into use at the mill of the Liberty Bell Gold Mining Co., at Telluride, Colo. Hard cast-iron ribs are laid with the regular

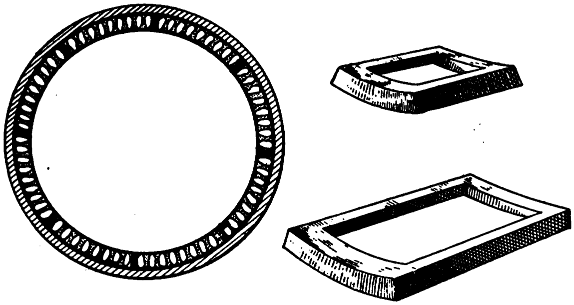


FIG. 33.—Lining made of pebbles cemented in cast-iron frames.

courses of silex blocks, greatly increasing the life of the lining as a whole. A lining in a 5 by 22-ft. mill, containing seven rows of ribs, or one rib for every four rows of blocks, gives a life of 2 years and 7 months of continuous operation. In all our 5 by 22-ft. mills, we have placed 10 rows of ribs, or one rib for every three rows of silex blocks, and to all appearances our expectance of a life of 3 years will be reached.

"The material used in lining a 5 by 22 -ft. mill includes 900 silex blocks averaging 4½ by 5 by 8½ in. and weighing approximately 13 lb.; also, 19 cu. ft. of Portland cement, 19 cu. ft. of clean sand and 10 rows, or 50 pieces, of hard cast-iron ribs. These ribs are given a taper, measure 1¾ by 2 by 4½ in., and are 47½ in. long. They weigh approximately 110 lb. each. They are cast in Telluride, since the local foundry provides the best hard iron that we have been able to obtain.

"Two picked men line the mill in three shifts. The lower half is lined in one shift. Live steam is then turned in for 8 hr., and the mill is allowed to cool by draft for 8 hr. more. Next the mill

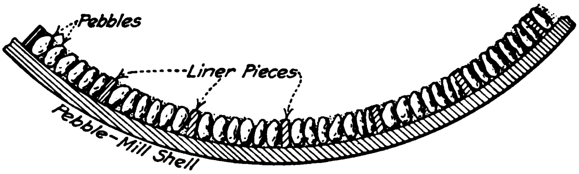


FIG. 34.—Pebble and iron bar lining.

is given a quarter-turn, and one-quarter more of its lining is put in, followed by the same heating and cooling. On the third

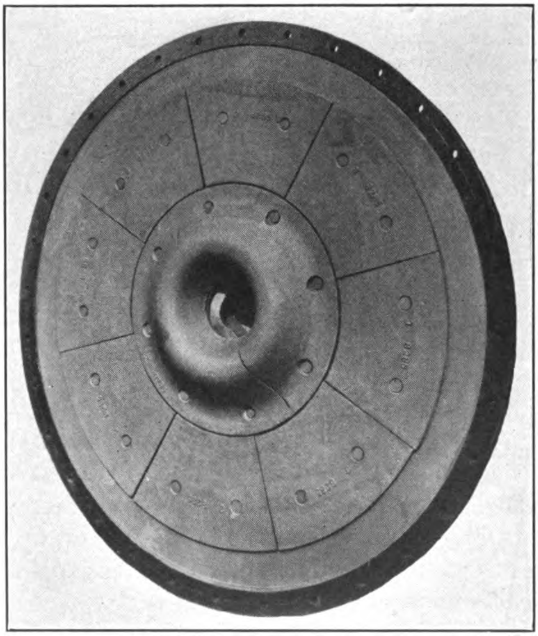


FIG. 35.—Feed head lining of 5 by 8-ft. tube mill at the United Gold Mines Co. mill, Oregon.

day the lining is completed, and it is afterward steamed for 12 hr. loaded with pebbles and then cut into service.

"Where the cement is uncertain or slow setting, or where it is necessary to use it on minor repairs, soda ash is added to the mortar. This cement mortar sets rapidly and allows the mill to be put into use in 8 or 10 hr. The ribs are laid in mortar only, and no bolts are used in the lining. Experience has not developed any difficulties in holding the ribs or the lining by this method. No timbering is used in putting in the lining.[1]"

We find a general tendency to reline mills with discarded steel or iron from mine or mill such as old rails, rock-drill bits, stamp-mill liners, etc. The illustration, Fig. 34, shows how semi-steel battery liners are being utilized at the Plymouth Con. mill.[2]

The plates were broken into pieces roughly 4 in. wide and 12 to 36 in. long. These pieces were set in cement with the pebbles, the liner pieces being set lengthwise of the mill and from 8 to 12 in. apart. They were, on the whole, about flush with the pebbles, though some points projected as much as ¾ in. above the general pebble surface.

The lining as described, and as shown in the sketch, was installed in an 8-ft. by 22-in. Hardinge pebble mill and has every appearance of wearing well, although it has not been in use long enough to wear it out.

The Britannia lining, so-called because it is used at the Britannia mill, B. C., is made with rail-sections 5 in. long set on ends laid in neat cement. The grouting between the rails is made by embedding short pieces of drill steel in the cement. These pieces of rails present a rough surface and are said to give good service. Such a lining used on the Rand, South Africa, was found to add as much as 400 to 500 oz. amalgam to the output by the recovery of the amalgam collected between the pieces of rails.

Mill	Material	Pounds per ton ground	Cost per ton ground	Life, months
Waihi	Corrugated-ribbed cast iron	.....	.....	18
El Oro	El Oro	1.000	$0.0410	
Nipissing	4 in. silex	1.378	0.0342	11
Goldfield Con.	4 in. silex	0.270	0.0150	7
West End Con.	Smooth cast iron	1.250	0.0450	
Nevada Wonder	El Oro	0.200	0.0220	15
Black Oak, California	Ribbed	6.000	0.0266	
Nevada Packard	Forbes			
Porcupine Crown	Globe			

[1] *Engineering and Mining Journal,* July 1, 1916.

[2] *Engineering and Mining Journal.*

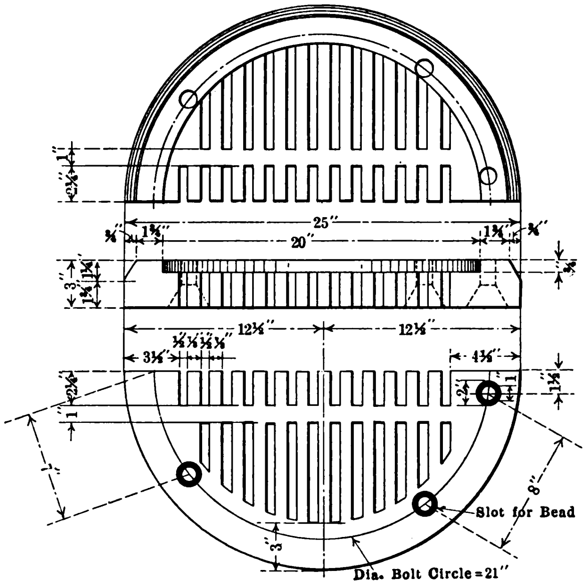
The ends of cylindrical tube mills are lined with cast-iron plates, preferably chilled. Fig. 35 illustrates such a lining. Manganese steel has been used for end plates with decreased cost per ton of ore ground.

The preceding table will give an idea of the character and cost of liners in tube mills and while accurate as to figures at the time, some of the mills have since changed from silex to metal liners.

IX. Character of the Discharge Opening

The diameter of the discharge opening affects the capacity of a tube mill by limiting the amount of pulp in the mill to the lower level of the opening and by limiting the load of pebbles to the same level unless a screen or grating is used at the discharge end to keep the pebbles in the mill. The position of the discharge opening or the particular point in the discharge end at which the ore is allowed to escape from the mill affects the capacity by fixing the amount of pulp in the mill and the speed with which it runs through the mill. Suppose the mill has a 6-in. discharge pipe at the center of the discharge end, the amount of pulp leaving the mill will be determined by the difference in level of the pulp at the feed and discharge ends and its fluidity. Now suppose we put a grating at the discharge end (closed at the center if desired) and discharge at the periphery of the mill, the "head" of pulp in the mill will be increased and the discharge will be at a maximum. Suppose now we retain the grating but enclose the discharge end leaving a 6-in. pipe in the center for the discharge and arranging a series of lifters or elevators between the grating and the end, the discharge will still be at a maximum with the advantage that any degree of dilution can be maintained in the mill. This idea is incorporated in some of the latest mills. It was thought at one time that the ore should be allowed to remain in the mill until finally ground to the desired mesh and a high pulp level was maintained. It is now considered advisable to pass the ore through the mill as quickly as possible by keeping a low pulp level and return the oversize for further grinding. The low pulp level assists grinding by allowing the pebbles or balls to fall with less resistance from the pulp and therefore the force of the blow delivered is greater.

Gratings are used in tube mills at or near the discharge end so that the pebbles may be carried above the center thus giving increased grinding efficiency. The mesh varies from ½ to ¾ in., which allows pebbles smaller than this size to escape from the mill. As a matter of experience when pebbles have been worn



2 Castings like this completes
Screen

FIG. 36.—Cast-iron screen for discharge end of tube mill.

to 1 in. in diameter, more or less, they suddenly disappear, being ground so as to be undistinguishable from the ore. When a tube mill is emptied of its contents, one seldom sees small pebbles.

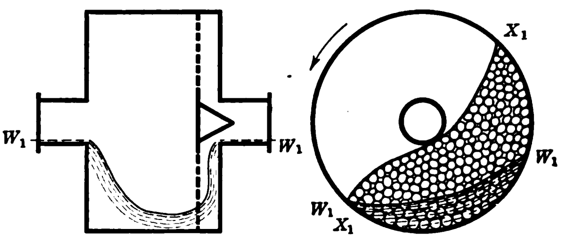


FIG. 37.—Marcy mill periphery discharge.

Fig. 36 illustrates the grating used at the discharge ends of the tube mills at the Goldfield Consolidated mill made of cast iron, but preferably made of manganese steel.

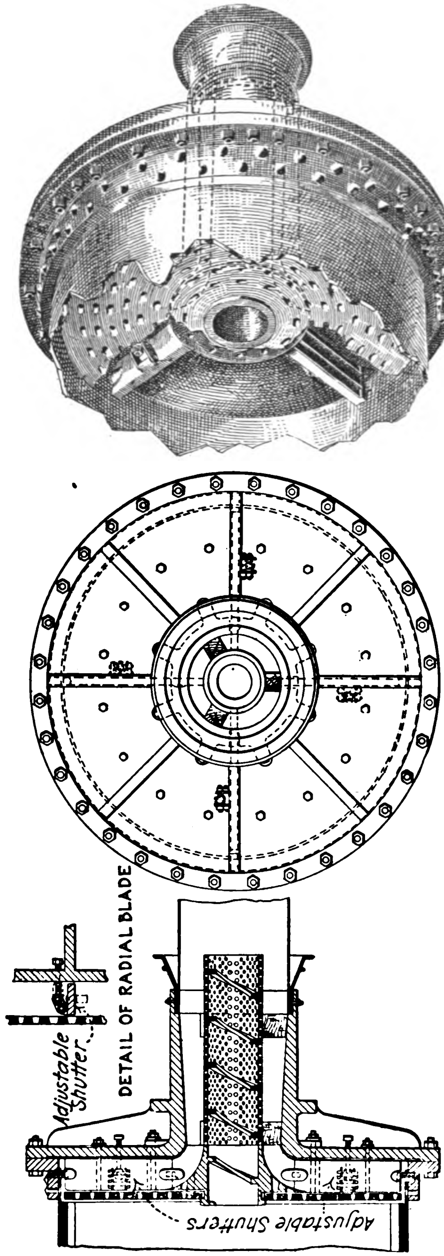


FIG. 38.—Quick discharge arrangement for tube mills.

The peripheral discharge is incorporated in the Marcy mill, Fig. 37, which shows the passage of the pulp through the mill and the level at which the pulp remains by reason of the radial ribs or lifters which removes the pulp as soon as it enters the chamber at the end of the mill.

Chalmers and Williams have added to their tube mills an adjustable quick discharge device, Fig. 38, which consists of a perforated plate, interposed between the end of the tube mill and the body of the mill. Between the perforated plate and the end of the mill are radial blades which are used practically as elevators. The blades are arranged on hinges, so that by changing their inclination with respect to the tube mill end, the space between the perforated plate and the mill end may be more or less closed and the amount of pulp elevated greater or less.

The following tests at the Gold Hunter mill, Idaho, show the value of the adjustable quick discharge fitted to a 5 by 14-ft. tube mill; test No. 1 being for the mill without the discharge in use and test No. 2 being for the mill using the adjustable quick discharge.

	Test No. 1	Test No. 2
Capacity, tons per day	112.06	141.600
Cost power per ton, cents	5.95	5.100
Cost pebbles per ton, cents	1.50	0.875
Total cost power and pebbles per ton, cents	7.45	5.975
Weight of pebbles used per ton, pounds	1.2	0.700

The addition of the adjustable quick discharge lowered the cost of tube mill regrinding in the above instance by approximately 1½ cts. per ton.

In the Allis-Chalmers adjustable ball mills the

"lining is of manganese steel, with stepped corrugations. The mechanism of the machine includes a diaphragm near the discharge end, having a number of concentric holes, which may be closed by wooden plugs. With all the holes open the mill will discharge within 3 or 4 in. of the periphery, and by closing the holes the discharge is raised accordingly. By this means the pulp level is regulated to suit the material crushed, and to effect the ratio of fineness of product that may be required." [1]

In mills where the load of pebbles is kept near the center line

[1] W. A. Scott, *Mining and Engineering World*, June 24, 1916, p. 1165.

the hollow trunnions are lined with an internal screw or helix which, when the mill revolves, causes the coarse material or pebbles to progress into the mill and provides means for feeding pebbles at this end. Fig. 39 illustrates the idea. In combination with the helix we may have a conical lining to the trunnion which somewhat answers the purpose of the conical end of the Hardinge

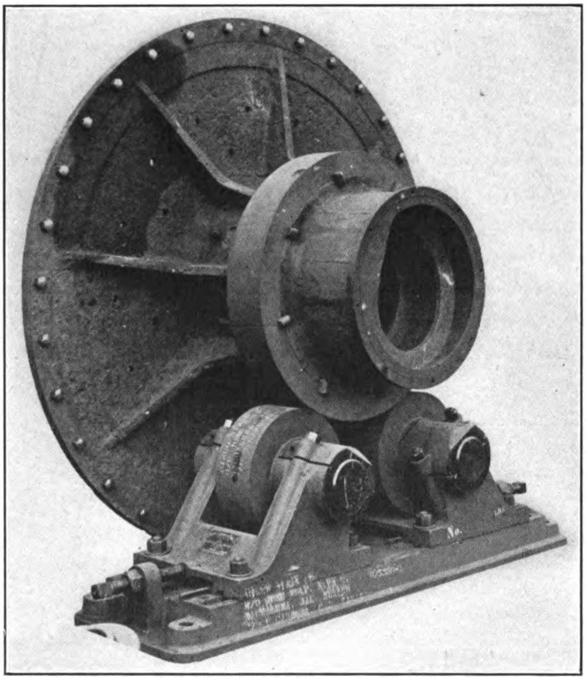


FIG. 39.—Discharge end of tube mill showing helix.

mill, presumably acting as a classifier, of limited extent and utility.

A device, the object of which is to increase the capacity of ball or tube mills, was recently tried out at the Tough-Oakes mill, Kirkland Lake,[1] on a 6-ft. Hardinge conical ball mill that was installed last year to handle 100 tons of ore per day.

"The ore, generally speaking, is a hard and rather tenacious reddish porphyry with gold disseminated throughout in minute joints and small quartz stringers. This ore proved comparatively refractory to disintegration by ball mill treatment, for after a thorough test extending over

[1] C. SPEARMAN, *Engineering and Mining Journal*, April 15, 1916.

a period of 3 months it was found that the ball mill had an average daily capacity of only 82.68 tons for total time and a maximum capacity of only 90.62 tons for actual running time, instead of a 100 tons daily. Many experiments were conducted by ball mill experts to increase the capacity of this mill during the period mentioned, including increase and decrease of revolutions, load, etc., but the capacity failed to reach the prescribed figure.

"Finally, the idea was conceived of a less obstructed discharge by permitting the whole product, oversize and undersize which was inclined to accumulate in the conical section of the mill, to pass through the conical revolving screen inserted and fastened in the discharge, as shown in Fig. 40, thereby classifying the product and returning the oversize to the feed end of the mill by means of a conveyor. This arrangement, as will be seen, permitted a much heavier load to be carried and developed consequently a greater discharge area. Later there was installed

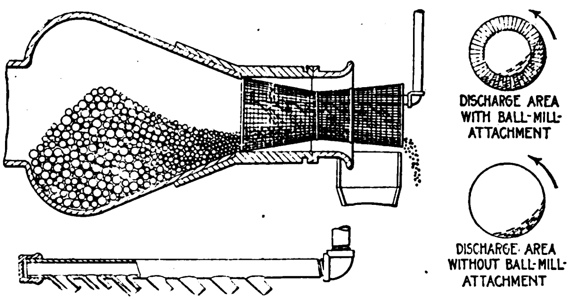


FIG. 40.—Rotary screen applied to conical mill.

a stationary water pipe with a series of water jets arranged so as to assist the discharge of both the oversize and undersize.

"After a 5-month trial of the device it was found that the capacity of the mill had been increased so as to average for that period 102.27 tons per day and for actual running time 110.41 tons per day, thus giving an increase in tonnage of 23.7 per cent. for total running time and 21.8 per cent. for actual time. The power required to operate the mill before the installation of the device was about 40 hp.; after the installation it was 35 hp., showing an actual decrease of 12.5 per cent. By taking into consideration the increased tonnage for the actual running time, the actual decrease in power consumption per ton of ore was about 28.1 per cent.

"The device makes it possible to raise the level of the load in the ball mill without discharging the grinding medium and thus increase the load from 8,500 lb. to 11,500 lb., or about 35.3 per cent. This feature also decreased the ball consumption from 2.45 lb. per ton to 1.8 lb. per ton. Attached to the tube mills the device permitted ready discharge and classification of oversize ore particles and undersize pebbles, which tend to accumulate and thus congest the tube mill circuit. By its use about 2 tons of oversize is removed daily from the tube mill circuit and returned to the ball mill feed. Partly by substituting ore masses and partly by the use of the device, the pebble consumption was reduced from 16.1 lb. to 3.9 lb. per ton. The accompanying table is based on averages over a period of 3 months without and 5 months with the device."

COMPARISON OF BALL MILL RESULTS

	Without device	With device	Increase, per cent.	Decrease, per cent.
Average mill tonnage per day, total time	82.68	102.27	23.7	
Actual horsepower required	40.00	35.00	....	12.5
Average mill tonnage per day, actual time	90.62	110.41	21.8	
Actual kilowatt-hour per ton per day	7.94	5.71	....	28.1
Ball consumption per ton of ore	2.45	1.80	....	26.5
Pebble consumption per ton of ore	16.15	3.94		

"From this test the following advantages are shown for the device: It permits an increase in load of grinding medium of 35 per cent. more than could be retained with the old open-mouth discharge under normal working conditions, thus subjecting a greater area of ore to attrition; it increases the effective discharge area of the outflowing pulp, as shown in the illustration; it permits free discharge of the product from the conical section of the ball mill where the relatively refractory oversize particles of ore are inclined to accumulate, due to the shorter radius of action, and thus retard the discharge of the fine product; it permits a continuous classification during discharge, separating the oversize, which is returned to the feed end or zone of maximum disintegration of the ball mill (about 10 per cent. of the discharge product is classified as oversize and thus returned to the zone of greatest efficiency for further disintegration); it also decreased the ball and pebble consumption, and the cost, including maintenance, is low—about 0.00125 cts. per ton treated."

We have all noticed in the problem of settling sand or slime that a thick pulp such as that containing 32 to 38 per cent. moisture settles as a homogeneous mass with no segregation of heavy sulphides or sand, in fact at times the sulphides remain at or near the top, but that when this pulp is diluted with solution or water the sand and heavy minerals settle out leaving the slime alone in suspension. It is for this reason that we are able to agitate sand and slime if we keep the pulp thick, but immediately we thin out the pulp the sand settles and gives trouble. Taking for granted this fact, which I believe is beyond dispute, it occurred to me that advantage might be taken of this phenomena, at the discharge end of a tube mill to classify the pulp before it is discharged from the mill. If we diluted the pulp at the discharge end of the mill we might be able to cause a quick discharge of slime and the retention within the mill of the sand which on dilution of the thick pulp would sink and be further ground, its place being taken by the easily floated slime. This was tried with success, the preliminary experiments being in the nature of a trial but with no exact data for comparison. A further application of a stream of water or solution within a ball or tube mill is that the character and amount of grinding can be varied to suit the conditions of the after-treatment of the pulp. We know that economical grinding requires a thick pulp and also that a thick pulp promotes sliming so that when we require capacity and a granular product, which is obtained by using a dilute pulp we have two contrary conditions. This may be overcome in part, at least, by diluting the pulp within the mill at the point where sliming must be stopped. The idea is so easily carried out, simply by inserting a pipe in the discharge end of the mill at any distance within the mill, that all who read may try the experiment with little or no expense. The amount and character of the discharge is not governed entirely by the position of the water jet, but also by its volume, so that we have two adjustments to make, the position and amount of water added to the pulp within the mill.

It will be noticed in the Kirkland Lake experiment, Fig. 40, that there is a water jet inside the screen, this being the first printed application of the idea but the full significance of adding the water inside the mill is not indicated.

THE FEEDING DEVICE

The feeding device on a tube mill is used for supplying a continuous stream of pulp to the interior and to provide means whereby pebbles may be taken into the mill.

The usual form of feeding device for pulp now in use is the feed scoop, bolted to the trunnion, which may have as many as three openings. Fig. 41 is a single scoop while Fig. 42 has three openings. This scoop delivers the pulp to a spiral in the trunnion which as the mill revolves conveys it to the interior of the

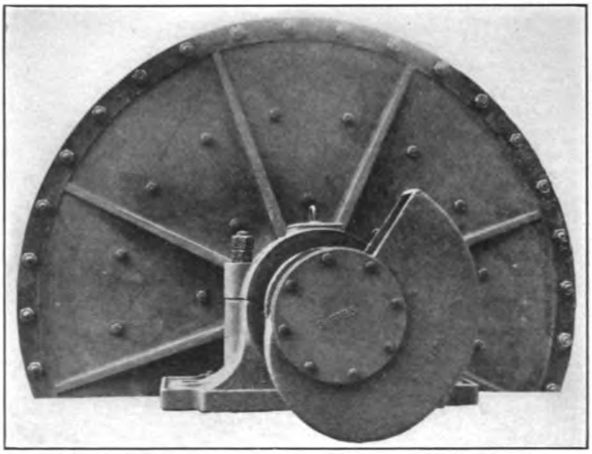


FIG. 41.—Single-feed scoop on the 5 by 8-ft. tube mills at the Cougar mill, Oregon.

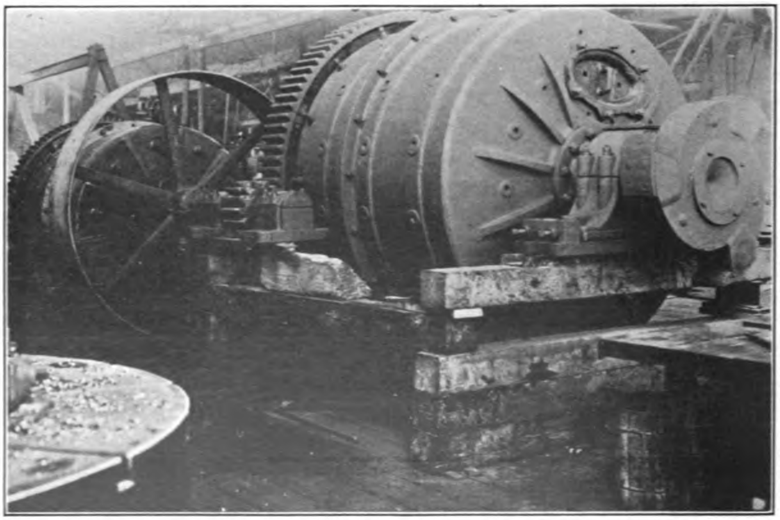


FIG. 42.—Marcy mill with triple-feed-scoops.

mill. The ore may be fed through a pipe entering the trunnion with a water-tight joint, as shown in Fig. 43, being the device used at the Waihi Grand Junction mill.[1] In the smaller mills

[1] *Mining and Scientific Press*, June 19, 1915.

the feed nozzle is 1¼ in. in diameter. A clearance space of 18 in. is left around the feed pipe; hence the pulp discharging into the mill causes a current of air to enter between the two pipes, thus preventing any chance of overflow of pulp at the inlet end.

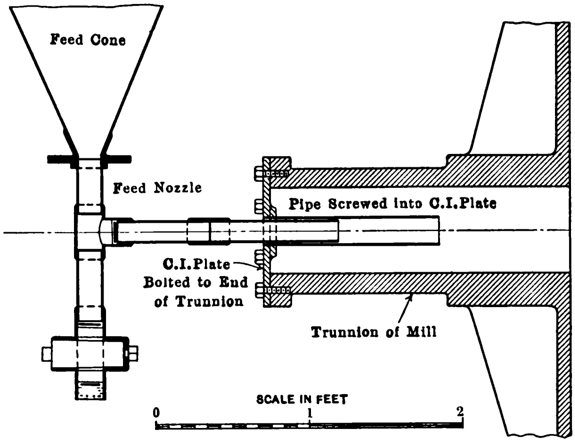


FIG. 43.—Injector for feeding pulp to tube mill.

The Graham feeder, Fig. 44, shows the same idea applied to the pulp feed, with the addition that the pebbles are fed through the same pipe, the pulp being injected with the pebbles.

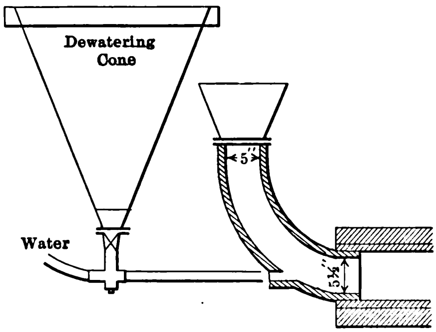


FIG. 44.—The Graham feeder. (Smidt.)

Usually the discharge trunnion is fitted with a reverse spiral which conveys the pebbles into the mill against the stream of pulp. Pebbles may be fed to the scoop on the feed end of the mill, but the process is slow when feeding by hand and it is impracticable to enclose the feed scoop in a box and depend upon the scoop to pick up the pebbles without abnormal wear on the lip of the scoop or injury to the box. Hand feeding is practical only at the discharge end where, as previously explained, there

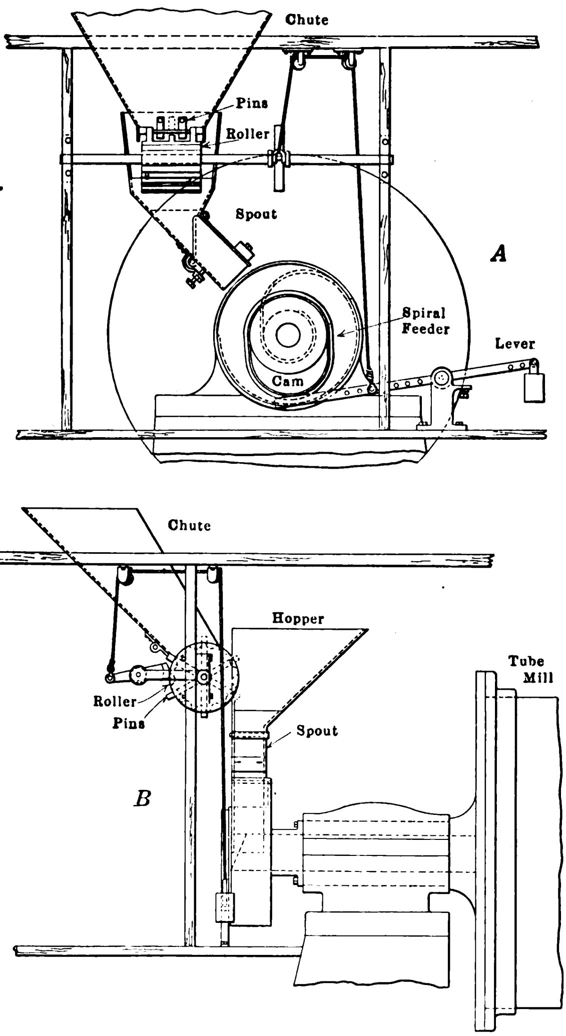


FIG. 45.—The Thomas automatic pebble feeder.

is a reverse spiral in the trunnion. Various devices have been employed to overcome this difficulty and feed the mill with pebbles by automatic feeders. That shown in Fig. 45, known as the Thomas feeder U. S. Patent No. 1045342 and described as follows, offers a very good solution of the problem:

The above device comprises a chute which leads the pebbles up to a hopper from which they are discharged into the spiral feeder of the mill. A fluted or corrugated roller delivers the pebbles from the chute into the hopper supplying the mill, and it is provided with projecting pins which protrude through corresponding slots in the hinged bottom of the chute and prevent choking or jamming of the pebbles in the chute. Movement of the shaft upon which the roller is mounted is provided by a friction wheel on the shaft moved by a rope from a lever actuated by means of a cam or eccentric on the casing of the spiral intake. The movement of the lever can be regulated and so made to control the movement of the fluted roll governing the feeding of the pebbles.

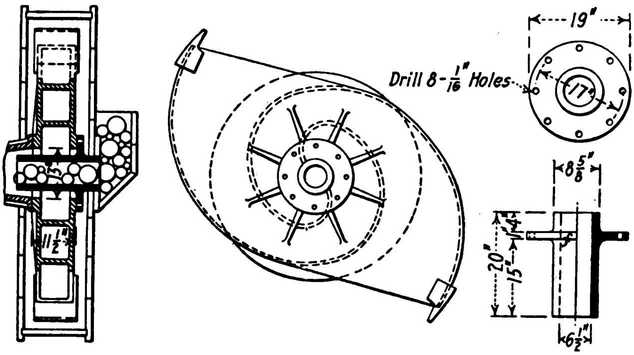


Fig. 46.—The Anaconda feeder.

The pebbles, after entering the hopper, are conveyed through an inclined spout which delivers them directly above the entrance opening of the spiral feeder, into which they fall and are conveyed inside the tube. The movement is so set that the pebbles will be dropped into the intake of the spiral feeder at the proper point of the revolution. By means of this device the chute can be maintained free of pebbles, which will automatically be fed into the tube mill without especial care on the part of the operator. The feeding being continuous, the replacement by new pebbles more nearly approximates the natural rate of wear and avoids the fluctuating efficiency caused by allowing great wear to take place and suddenly replacing it at one time.[1]

At the Anaconda concentrating mill the device, Fig. 46, was successfully adopted for feeding pebbles when it was found impracticable to feed the pebbles in the scoop box. The difficulty

[1] *Engineering and Mining Journal*, April 19, 1913.

was complicated on account of the feed scoop being 7 ft. in diameter in order to lift back into the mill the sand discharged from the Dorr classifier and to retain the necessary grade from the classifier to the tube mill scoop.

POWER REQUIREMENTS

The power required to rotate a tube mill depends upon the speed of rotation, the weight of the mill, the character of the lining, the load of pebbles, the consistency of the pulp and the character of the power transmission. The following table will serve as a guide, and although not complete in details it may be used for making estimates.

Size of tube mill	Hp. to rotate	Speed, r.p.m.	Lining	Load of pebbles	Tonnage per 24 hr.
Ft. Ft.					
$3\frac{2}{3} \times 13\frac{7}{12}$	17.0	....	....	....	37.50
$3\frac{11}{12} \times 19\frac{1}{2}$	48.0	31	El Oro	....	100.00
4×12	14.0	26	Silex	....	24.00
4×16	16.0	26	....	....	30.00
4×16	30.0	29	....	....	38.00
4×16	30–35	....	Mn steel		
4×20	60.0	31	Forbes	....	75.00
$4\frac{1}{2} \times 19\frac{1}{2}$	50.0	30	El Oro	....	100.00
$4\frac{1}{2} \times 20$	16.0	20	....	....	30.00
$4\frac{1}{2} \times 20$	43.0	30	Silex	....	75.00
$4\frac{1}{2} \times 20$	16.0	20	Chilled iron	....	30.00
$4\frac{3}{4} \times 18$	55.0	$25\frac{1}{2}$	Ribbed cast iron	....	77.00
$4\frac{11}{12} \times 23$	80.0	$27\frac{1}{2}$	El Oro	....	140.00
$4\frac{11}{12} \times 26$	87.0	$28\frac{1}{2}$	El Oro	....	165.00
5×16	78.0	29	Globe	....	108.00
5×16	60.0	26	Smooth cast iron		
5×18	48.0	26	Ribbed cast iron		
5×18	60–62	26	Smooth cast iron		
5×20	40.0	26	El Oro		
5×20	50.0	26	El Oro	....	50.00
5×22	42.5	27	El Oro	....	52.00
5×22	42.5	27	Smooth cast iron	....	55.00
5×22	45–48	....	4 in. silex	....	73.40
5×22	50.0	24	Soft steel	....	125.00
5×22	54.0	28	El Oro	Above center	69.40
5×22	59.0	27	....	6 in. above center	88.75
5×22	85.0	32	4 in. silex	....	95.00[1]
5×22	80–90	32	El Oro	....	90.00[1]
5×24	55.0	26	El Oro	....	121.00
6×20	60–68	25	4 in. silex	3 in. above center	244.00

[1] Estimated.

The list of 5 by 22 ft. is the most interesting as it shows that the increased tonnage obtained by carrying the load above the center involves but a slight increase of power: it also shows the increase of power required when the mill is revolved at 32 r.p.m. To compensate in power for rotating the mill at 32 instead of 27 r.p.m. the tonnage should be at least double that at the latter speed.

The mistake is too often made of underestimating the amount of power required to start and run a tube mill. The mill must be started from rest with the pebbles imbedded in mud and even if the mill is rocked back and forth with a clutch the power required to start the mill is far above that required to keep it running once

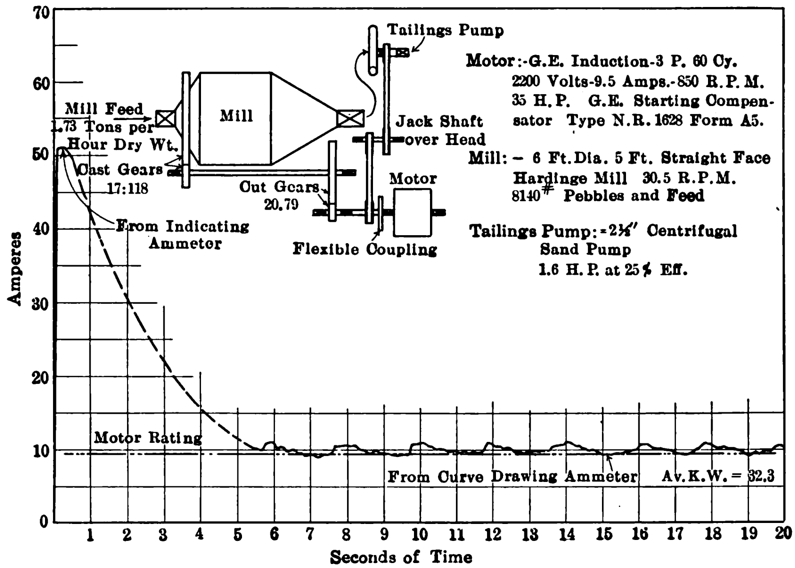


FIG. 47.—Curve showing ampheres required to start and run No. 3 Hardinge mill. Winona stamp mill.

it has attained its maximum velocity. The diagrams, Figs. 47 and 48, given to me by H. H. Seeber of the Winona Copper Co. show the starting current of the motors used to start two sizes of Hardinge conical mills. It will be observed from the curve that for the first 5 sec. the motors must bear a great overload. At the Liberty Bell mill a 75-hp. motor is used to start the 5 by 22-ft. mill while the running load varies from 45 to 48 hp. Probably the 75-hp. motor takes at least a 25 per cent. overload to start the mill from rest. At the Montana-Tonopah a 5 by 20-ft. mill requires 60 hp. to start and 42½ hp. to keep running.

We may consider that a tube mill requires twice the amount of power to start, that it requires to keep it running.

If the source of power is an electric motor this overload must be provided for in the rated capacity of the motor and in its guaranteed overload for a given time; if run by an explosive gas engine the flywheels must be of sufficient weight to store up the momentum necessary to start the mill without killing the engine. The momentary overload capacity of an electric motor being high and that of an internal combustion engine being low, it is

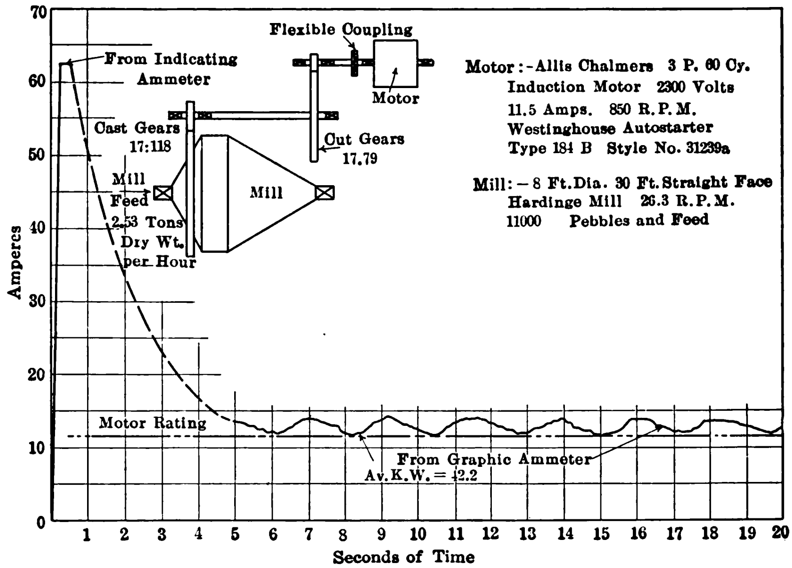


FIG. 48.—Curve showing amperes required to start and run No. 1 Hardinge mill. Winona stamp mill.

not necessary to have a flywheel with the former, as it is essential with the latter.

A. M. Merton[1] gives the following formula for calculating the power required to revolve a tube mill when 60 per cent. of the volume of the mill is occupied with pebbles, moisture in feed 38 to 40 per cent. and the peripheral velocity of the mill 400 ft. per minute.

Horsepower = 0.25 × cu. contents in feet (of pebbles).

This is a near approximation and figures out, for a 5 by 22-ft. mill, 52 hp.

[1] *Mining and Engineering World,* June 14, 1913.

When stopping a tube mill it is well to thin out the pulp, as this will help to start the mill from rest, and when starting the mill it should be flooded with water or solution, as when at rest, the solution runs out of the mill leaving a thick mud which, with the pebbles, forms an almost solid load.

APPLICATION OF POWER

The means by which a tube mill is made to revolve as well as the position of the driving mechanism in respect to the feed

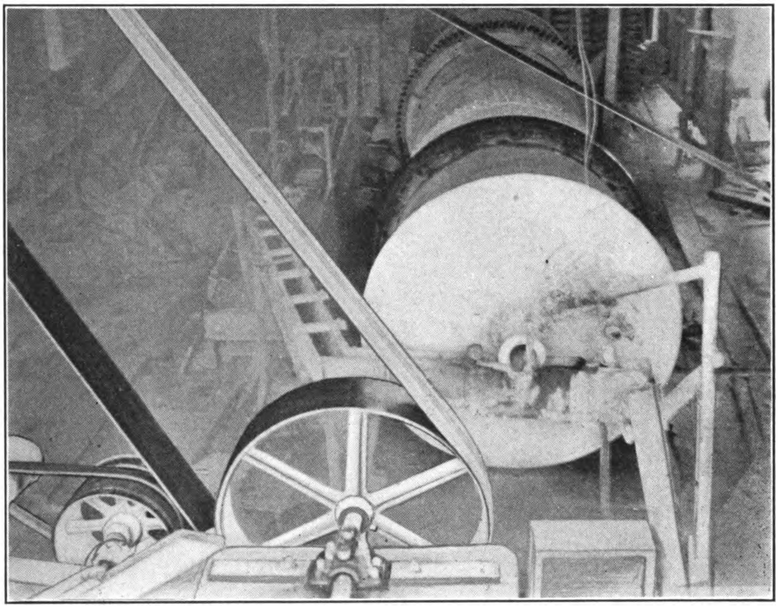


FIG. 49.—Belt-driven 5 by 22-ft. tube mill at the Techatticup mill.

and discharge openings appears to be a matter of individual preference, as shown by the pinion shaft driven by belts, Fig. 49, silent chains, Figs. 50 and 51, and motors direct-connected to the pinion shaft, Fig. 52. Some operators have changed from belts to gears ostensibly because the belts have given trouble by slipping but more likely due to not having the belt wide enough; some have changed from gears to belts because the gears have given poor service. We see some machines driven from the feed end and some from the discharge end of the mill. For convenience in being out of the way of possible splash and dirt the latter method is preferred. Both the Hardinge mill illustrated, Fig. 52 and the cylindrical mill, Fig. 11, are driven from the discharge

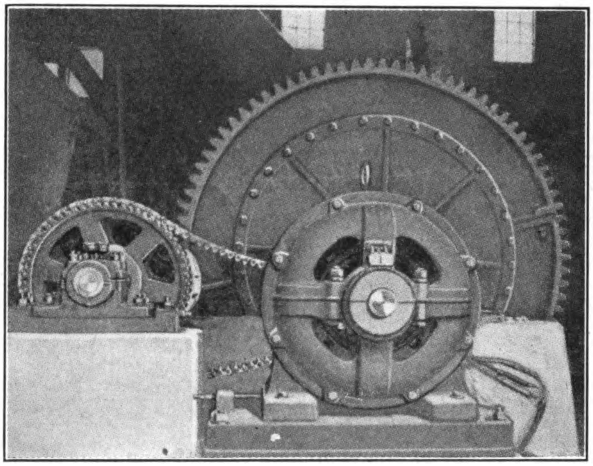


FIG. 50.—75 Hp. chain driving tube mill at the Mexican Gold and Silver Mining Co. mill, Virginia City, Nev.

end. In some cases mills have been driven by two parallel belts from both ends. All the up-to-date mills are driven from the

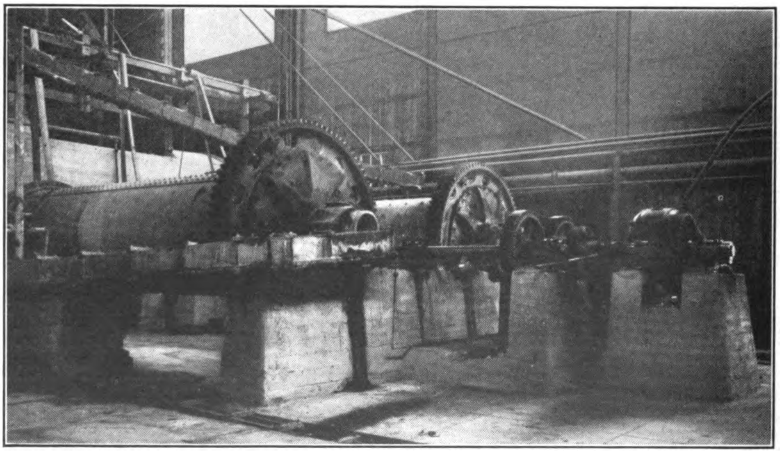


FIG. 51.—Morse chain drive at the Belmont mill, Tonopah, Nev.

pinion shaft by herringbone gears which in the case of the Hardinge mills is said to effect a saving of at least 15 per cent. in power. The advantage of this type of gear is not only that it saves power but not being subject to the amount of vibration in the ordinary tooth gear, it allows a motor to be direct-connected to the pinion shaft. Care should be taken that the gear is prop-

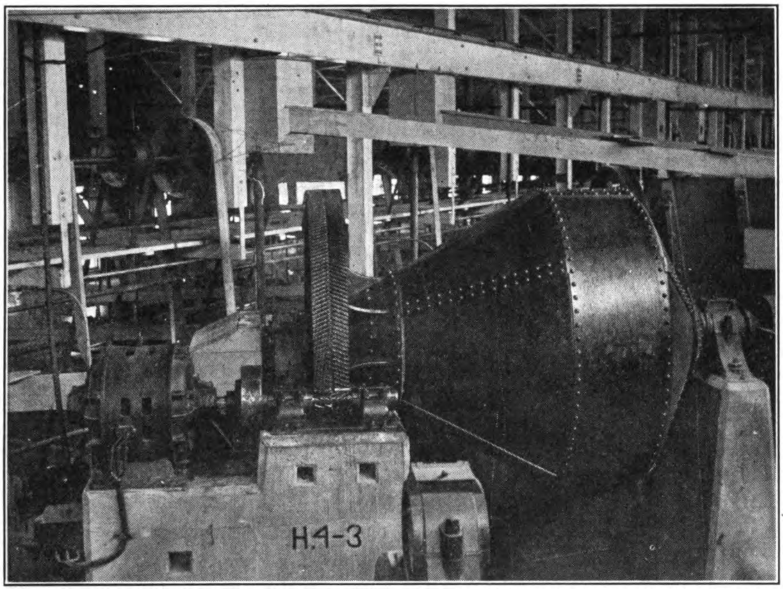


FIG. 52.—Direct-connected Hardinge mill driven by herringbone gears. Baltic mill, Michigan.

erly installed in the first place or the advantages claimed will not materialize.

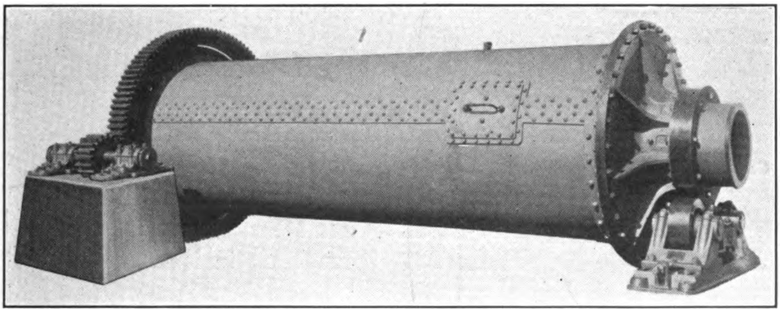


FIG. 53.—Tire and trunnion tube mill.

Mistakes are frequently made of using belts on the pinion shaft, which are entirely too narrow to pull the mill without stretching the belt to the point of ruin, for it must be remembered that the belt must be wide enough to start the mill from rest, an effort which, as we have seen, requires about twice the power necessary to keep it going. A 5 by 22-ft. mill with a 54-in. pulley on the pinion shaft requires a 14-in. belt, but designers will persist in recommending 10 and 12-in. belts for this work, and then blame the belt for slipping. There is no economy in narrow belts for this heavy work even if doubled; it is better to have a wide belt in the beginning with the pulleys of proper proportions and save trouble.

A tube mill must be started with a clutch on the countershaft, pinion shaft or engine. If the engine runs nothing but the tube

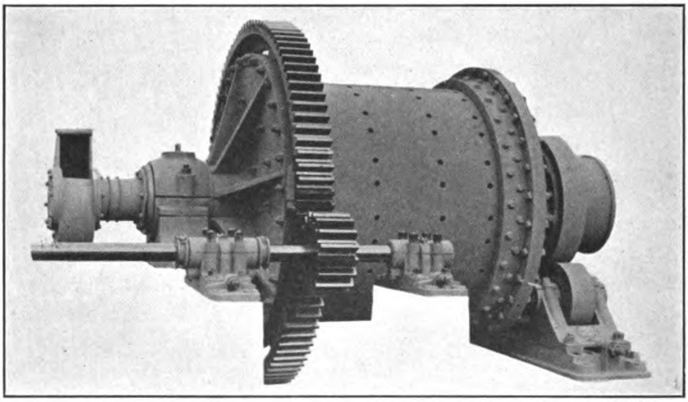


FIG. 54.—Tire-and-trunnion mounted ball mill.

mill, the engine shaft is a good place for the clutch; otherwise, it is better on the pinion shaft or countershaft. Wherever it is placed it should be of sufficient strength to start the mill from rest which, in the case of a 5 by 22-ft. mill, means a clutch rated at 100 hp. or over. Any clutch below this capacity for this size of mill will last but a short time.

The mill may be revolved on tires or on trunnions or both. For a long mill such as a 22-ft. mill, tires are preferred because the weight can be better distributed, resulting in less thickness of shell, and there being less friction on a 6-ft. diameter tire running on rollers than on trunnions, less power is required to rotate the mill. For a medium-size mill, such as those 18 ft. long, a combination of tire and trunnion illustrated in Fig. 53 may be used or a ball mill, Fig. 54 may be mounted in the same way.

If a tire-mounted mill is ordered, it is essential to specify that the tires and rollers be made of steel either rolled or cast and not of cast iron. The inner ring should be rivetted, not bolted and there should be no sign of a leak, for the least bit of grit on the rollers soon ruins them. The roller shafts should be lubricated

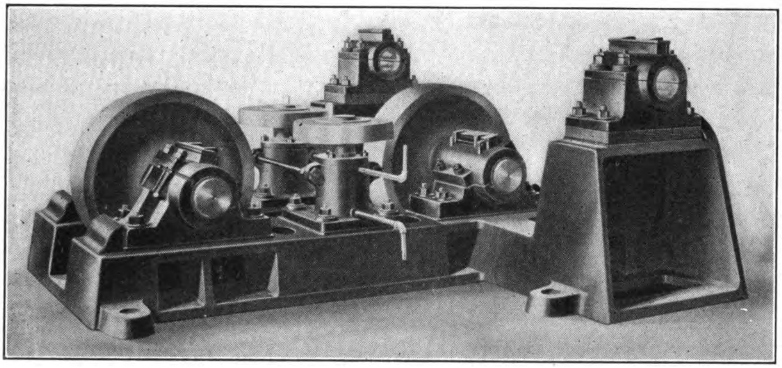


FIG. 55.—Abbé bed plate for tire-mounted tube mill.

with a good grade of grease but the surface of the tires and rollers should be oiled with heavy black oil. Fig. 55 shows the Abbé style of tube mill bed plate with supporting rollers, thrust rollers and countershaft bearings for the drive end of the mill. The thrust rollers are for the purpose of preventing the mill from sliding off the supporting rollers, but a well-balanced mill should float on the rollers and not touch the thrust rollers at all. Remembering that the mill will travel toward the rollers that are spread apart the mill may be made to travel either way on the rollers by turning an adjusting screw a fraction of a turn. The rollers should be placed at an angle of 120°, as shown in Fig. 56.

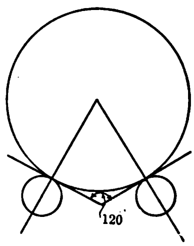


FIG. 56.—Proper angles for rollers on tire-mounted tube mill.

Fig. 57 illustrates a mill made by the Stearns Roger Co., the same being in use at the Liberty Bell mill, Colorado. In this mill the thrust rollers are absent and in their place we have a rim on the tires about 1 in. deep made of forged steel. This is an improvement because, if the shaft of a thrust bearing should break, there is nothing to prevent the mill from running off the tires, and such accidents may happen at any time.

With any type of bearing the mill should be revolved in the direction that will lift the gearing on the mill and press down that of the pinion shaft in order to avoid strains on the foundation bolts of the boxes.

Roller bearings have not generally been applied to tube mills because the usual mill is too heavy for this class of bearing but there is no reason why a medium-size mill run on tires should not be so provided. At the St. John del Rey mill, Brazil roller bearings were put on the tube mills with an estimated saving of 7 hp. This is quite an item and it would be well for manufactures to pay more attention to this point.

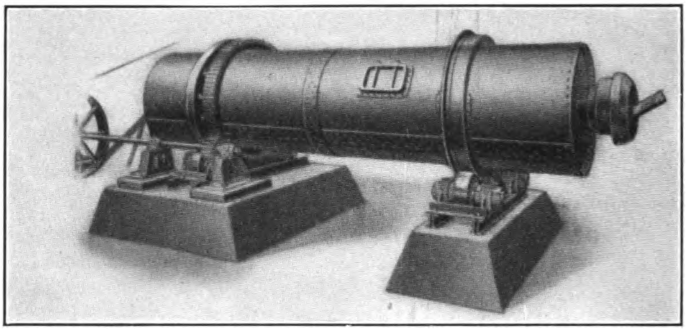


FIG. 57.—Tube mill with rim on tires.

COST OF OPERATING

The cost of grinding ores with a tube mill includes the three usual items of power, material and labor. The power as a rule being the greatest item of expense, every precaution should be taken to proportion the factors already discussed to obtain the maximum result with the minimum expenditure of fuel, and the type of transmission of power to the driving gears should be selected for efficiency with the first cost a secondary consideration. The cost for material should be only for the lining and pebbles, but renewal of tires and gears are often required due more to carelessness in operating than to actual wear of material. Clutches are often used for short periods and discarded being too weak to stand the starting strain of the mill, or belts are used too narrow for the work required and must be abused by stretching or "doping" until unfit for further work. A well-designed and well-operated tube mill will require but little expense outside of the pebbles and lining, but one that is neglected or abused will need frequent repairs and renewals of all the parts. The cost for labor outside of that required for relining the mill is a negligible quantity as the mill is usually run in connection with other crushers and the same labor may look after the tube mill, for outside of oiling and charging the new pebbles there is little work required until the mill is ready to be relined. At a certain mill in Nevada the labor cost amounted to over 30 cts. per ton due to poor operating and poor construction. The cost per ton of grinding in this mill is so abnormal that I have not included it in the list which follows.

The cost of grinding ores by the tube mill may be gaged by the following list of mills grinding gold ores to practically −200-mesh screen, except in the case of the Homestake where −100 is sought.

Mill	Cost per ton of ore ground	Remarks, costs per ton
Homestake, South Dakota	$0.3285	Power cost 7.59 cts.
Oroya Brownhill, W. A.	0.4210	Power cost 9.9 cts.
Hannas Star, W. A.	0.4360	
Yuanmi, W. A.	0.3600	Power cost 6 cts.
Waihi Junction, N. Z.	0.2800	Power cost 12 cts. (1907).
Dos Estrallas, Mexico	0.1560	
El Oro, Mexico	0.2630	
Lucky Tiger, Mexico	0.3580	
Hollinger, Canada	0.2770	Silent chain drive.
Dome, Canada	0.5000	
Liberty Bell, Colorado	0.0769	Power not included.
Gold Roads, Arizona	0.5000	
Montana-Tonopah, Nevada	0.6500	
Goldfield Con., Nevada	0.2080	
West End Con., Nevada	0.4920	
Mac Namara, Nevada	0.5230	
Nevada Hills, Nevada	0.2930	
Nevada Wonder, Nevada	0.2120	
Alaska Treadwell	0.4300	Grinding concentrates, power cost 12.79 cts.
Commonwealth, Arizona	0.3390	
Black Oak, California	0.1380	
Gold Hunter Mill, Idaho	0.0730	Feed 94 per cent. +35-mesh, discharge 30 per cent. −200 (1915).

As this last item shows an unusually low cost for tube mill grinding even with 30 per cent. −200-mesh, a further analysis may be of interest.

Nine months operation until lining was changed, 5 by 14-ft. tube mill

Pebbles per ton of ore crushed	$0.0120
Lining per ton of ore crushed (El Oro)	0.0197
Repair labor per ton of ore crushed	0.0003
Power per ton of ore crushed	0.0410
Total	$0.0730

Man running tube mill also looks after 12 vanners, two Wilfleys, one pump, two sets of rolls and two elevators. The mill grinds approximately to 30-mesh with feed 17.5 per cent. on 6-mesh for an average of 30 hp.

The cost sheet of the regrinding plant at the Homestake mine, South Dakota, is here given as being of particular interest. The feed contains 25 per cent. on 50-mesh screen and is ground without classification.

Regrinding Plant Operating Costs for 1914

	Cost per ton fed to tube mills	Cost per ton reduced to pass 100-mesh sieve
Labor	$0.0496	$0.1289
Pebbles and liners	0.0259	0.0673
Renewals, mills and cones	0.0074	0.0193
Machine-shop service	0.0037	0.0096
Silver-plating	0.0077	0.0200
Sundries	0.0029	0.0075
Power	0.0292	0.0759
Total	$0.1264	$0.3285

Regrinding the ⅝-mesh steam stamp product at the Winona mill, Michigan, the Hardinge conical mills in 1914 showed the following results:

Costs	Cents per ton ground
Power at 1.054 cts. per kilowatt-hour	13.64
Labor	0.65
Pebbles at 0.8 ct. per pound	2.08
Silex lining at 1.436 cts. per pound	0.09
Steel lining at 5.242 cts. per pound	0.33
Sundry charges	0.43
Total	17.22

It is possible to reduce the cost of tube mill grinding at the expense of the primary crusher, but the careful millman will so proportion the work of his various machines that the whole cost of reduction will be at the lowest compatible with economical extraction of metal.

The output of a tube mill will average about 10 tons of ground sand per ton of pebble charge per day, so that knowing the interior dimensions of the mill and the line at which the load of pebbles is kept we may find the weight of pebbles in the mill by the table already given and by multiplying by 10 we arrive at an approximation of the amount of sand ground to say 85 per cent. –200-mesh, which is an average screen analysis.

The output per horsepower will vary with the size of feed and discharge. Grinding to 90-mesh as practiced in South Africa the capacity is commonly reckoned at 1½ tons per horsepower, while the El Oro output at 150 to –200-mesh will average 1.2 tons per horsepower. The following list, taken from data at hand, is approximate.

Mill	Tonnage per horsepower	Mill	Tonnage per horsepower
Oroya Brownhill	2.0	Liberty Bell	1.5
Hannas Star	1.2	Montana-Tonopah	1.3
Waihi	1.4	Nevada Wonder	1.3
Dos Estrallas	2.2	Alaska Treadwell	1.5
Guanajuanto	1.7	Standard	2.5
Conecopia	1.0	North Star	1.8

For an average quartz ore grinding to the usual mesh for cyaniding we would consider 1.4 tons per horsepower to be a fair average.

The following table shows the cost of tube mill grinding compared with the cost of crushing with primary and intermediate machinery. This table may be useful when it is desired to have a means of comparing the cost of grinding with a tube mill in place of stamps.

Cost Per Ton

In this table a few figures will be found showing the cost per ton of stamping in comparison with coarse crushing in a rock breaker and fine grinding in tube mills.

Plant	Rock-breaker	Stamps	Tube mill	Total mill expense	Remarks
Homestake	$0.0600	$0.3460	$0.2570		
Goldfield Con.	0.0380	0.1330	0.1040	$1.652	1914.
West End, Tonopah	0.1030	0.2480	0.4920		
Hollinger	0.0730	0.1810	0.2770	1.493	
Wonder, Nevada	0.1210	0.2950	0.4180	3.760	Chilian 0.20.
Liberty Bell	0.0725	0.1908	0.0769		$0.077 amalgamation.
Lake View, W. A.	0.0500	0.3500	0.3500	2.430	
McNamara, Tonopah	0.1760	0.3400	0.5230	3.299	
Alaska Treadwell	0.0210	0.1380		0.230	
Yuani, W. A.	0.1060	0.3130	0.8070	1.524	
Belmont, Tonopah	0.1120	0.3500	0.4390	3.328	
Black Oak, California	0.0121	0.1126	0.0533	1.729	Power not included.
Nipissing L.-g., Ontario	0.1350	0.2380	0.5000		
Nevada Hills, Nevada	0.0520	0.2400	0.2930	2.622	
Pittsburg Silver Peak, Nevada	....	0.3140	....	1.210	$0.044 amalgamation.
Motherlode, B. C.	0.0160	0.1130	0.1180	1.398	$0.08 amalgamation, 1915.

COST OF INSTALLING TUBE MILLS

The following table written for the *Engineering and Mining Journal* by Percy E. Barbour

"gives details of tube mill specifications and quotations made to a large mining and milling company in Pachuca some time since. The prices are in U. S. currency, and have been reduced to prices per pound of tube mill complete, f.o.b. factory, and prices per cubic foot of charge when full to 2 in. above the center line. This gives two interesting methods of comparing the quotations. Mill *D* was selected, partly on account of price, although it was not the lowest in cost, and partly on account of superiority in certain mechanical features and construction."

TUBE MILL QUOTATIONS AND SPECIFICATIONS

	A	*B*	*C*	*D*	*E*	*F*	*G*
Size	4′ 7½″×19′ 8″	4′ 6″×20′ 0″	5′×20′	5′×20′	5′×20′	4′ 6″×20′	5′×18′ 3″
Thickness of shell	Welded about 13/32″	¾″	⅝″	7/16″	½″	½″	
Material in gear and pinion	....	Cast iron	Cast iron	Cast iron, cut	Cast iron, steel pin	Semisteel	
Gears—face and pitch	....	....	8″F., 2″P.	12″F., 1¼″P.	11″F., 3½″P.	8″F., 1⅞″P.	
Shaft dia. and rev. per min	125 r.p.m.	4 7/16″ − 142	4 7/16″ − 200	4 9/16″	4 7/16″ − 168	4 7/16″ − 200	− 160
Clutch pulley	94½″×10¼″	60″×14½″	36″×20″	78″×16″	78″×18″	60″×20″	74¾″×10¼″
Area of belt contact per min	2,524 sq. ft.	2,603 sq. ft.	4,549 sq. ft.	3,829 sq. ft.	4,805 sq. ft.	4,976 sq. ft.	
Size of trunnion	....	15″×15″	16″×18″	15″×18″	15″×18″	16″×18″	
Trunnion bearing pres. per sq. in.	....	115 lb.	118 lb.	122 lb.	131 lb.	142 lb.	
Vol. of charge 2″ above C.L. 3½″ lining	136 cu. ft.	132 cu. ft.	158 cu. ft.	161 cu. ft.	165 cu. ft.	133 cu. ft.	144 cu. ft.
Weight of charge at 135 lb. per cu. ft.	18,300 lb.	17,800 lb.	21,300 lb.	21,700 lb.	22,300 lb.	18,000 lb.	19,400 lb.
Net weight of unlined mill	25,580 lb.						
Gross weight of unlined mill	27,120 lb.	24,050 lb.	36,600 lb.	27,625 lb.	30,000 lb.	35,000 lb.	36,960 lb.
Weight of cast-iron ribbed lining	20,290 lb.	15,000 lb.	23,500 lb.	25,000 lb.	25,000 lb.	25,000 lb.	21,200 lb.
Weight of cast-steel ribbed lining	....	....	19,000 lb.	19,500 lb.	17,600 lb.	14,500 lb.	
Net weight of unlined shell	15,430 lb.	....	23,500 lb.	....	24,000 lb.	28,500 lb.	
Gross weight mill with c.i. lining	47,410 lb.	39,050 lb.	60,100 lb.	52,625 lb.	55,000 lb.	60,000 lb.	58,160 lb.
Price unlined mill, f.o.b. factory	$2,385	$1,775	$2,566	$1,900	$1,500	$2,750	$2,342½
Price unlined mill per pound	8.8 cts.	7.4 cts.	7.0 cts.	6.9 cts.	5.0 cts.	7.9 cts.	6.3 cts.
Price cast-iron lining	$1,095	$407.50	$799	$960	$750	$687.50	Mex. $1,178
Price cast-iron lining per pound	5.4 cts.	2.7 cts.	3.4 cts.	3.8 cts.	3.0 cts.	2.75 cts.	5.6 cts.
Price cast-steel lining	....	....	$860	$1,015	$1,056	So. B. $440	
Price cast-steel lining per pound	....	....	4.5 cts.	5.2 cts.	6.0 cts.	3⅜ cts.	
Price of mill with cast-iron lining	$3,480	$2,182.50	$3,365	$2,860	$2,250	S.F. $3,437.50	$3,520.50
Price of mill with c.i. lining per pound	7.3 cts.	5.6 cts.	5.6 cts.	5.4 cts.	4.1 cts.	5.7 cts.	6.1 cts.
Freight, fees, etc., but not duty	....	$437	$703	$547	$665	$636	
Price f.o.b. Pachuca	....	$2,619	$4,068	$3,407	$2,915	$4,073	About $3,720
Price f.o.b. Pachuca per pound	....	6.7 cts.	6.8 cts.	6.3 cts.	5.3 cts.	6.8 cts.	6.4 cts.
Price per cu. ft. charge, 2″ above C.L.	....	$19.80	$25.70	$21.20	$17.70	$30.60	$25.80
Delivery at factory	60 days	45 days	90–105 days	45 days	....	30 days	

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The cost of erecting a 5 by 18-ft. Gates tube mill at the Goldfield Consolidated is given as follows:

Weight of mill 25,287 lb. unlined, without motor.

Invoice (including chain drive)	$1,910.00	
Freight	650.73	
Labor (erection)	297.55	
Supplies (erection)	114.13	
Labor (foundation)	298.45	
Supplies (foundation)	258.94	
Labor (lining)	111.35	
Supplies (lining)	301.64	
		$3,942.79

Cost of erecting two tube mills 5 by 16 ft. at Portland mill, Colorado.

Invoice	$2,170.00	
Labor	841.26	
Lumber	63.39	
Rods and bolts	42.78	
Cement	190.00	
Electric supplies	88.20	
Motor	806.46	
		$4,107.73

Cost of erecting 6 by 20-ft. tube mill at the Nipissing mill, $3,023.00. Weight 45,375 lb.

TUBE MILL FOUNDATIONS

In placing of tube mill foundations the engineer must consider stresses due not only to static forces but also to dynamic forces, as the dynamic forces or those due to impact and vibration are more important than those due to steady compression or tension. The dynamic stresses being far in excess of those due to the weight of the machinery alone, instead of allowing a factor of safety of 3 to 6 as in ordinary structural work we must increase this many times to obtain a foundation that will stand the vibratory movements due to the shifting of the load within the mill.

The greatest compressive stress on the base of a tube mill of average weight is not over 10 lb. per square inch, while the usual mixture of concrete used in such construction will stand a compressive stress of 2,400 lb. per square inch or 240 times that due to the weight of the mill alone. This may in a sense be considered a factor of safety of 240. The Boston building laws allow pressures on concrete structures of not more than 55 lb. per square inch while the City of New York allows 200 lb. per square inch. If we take the Boston limit we have then a factor of safety of 5½ above that allowed in that city. Kent says that when the stresses are of a complex character and of uncertain amount, a very high factor is necessary, possibly even as high as 40. He gives us to understand that unless the strength of the material is known and the forces it must withstand are known, whatever factor of safety is assumed is a "factor of ignorance." So with concrete mixtures that must withstand dynamic stresses our factor of safety is a factor of ignorance because we have no standard tests for concrete, reinforced or otherwise, that will give an idea of the capabilities of the material for withstanding vibratory forces.

Without the advantage of dynamic tests on concrete specimens I would not apply any rigid rule, but it appears reasonable from the nature of the movements within a tube mill to place this factor at 100, for the concrete itself, but a foundation may fail from not being anchored to the bedrock or from not having the base cover enough area to prevent the structure from tumbling over or "creeping." Fig. 58 illustrates three forms of concrete foundation for a 5 by 22-ft. tire-mounted tube mill. *A* is a tube-mill foundation erected in 1915 which was most unsatisfactory, the whole pillar moving and crumbling from the vibrations communicated to it. The designer, probably to save a little concrete, lessened the base area with a hole through the center. What was required was an extended base area with the pillar firmly anchored to the bedrock as in illustration *B*. The repair actually made was as shown in *C*, the concrete being extended from pillar to pillar, forming one solid base and well anchored with numerous rods. If the foundation had been made as shown in *B*, it is probable that no movement would have taken place, the joining of the two pillars into one block being unnecessary.

Concrete plays such an important part in tube mill work that a few words on the subject may not be out of place, for whether we have a foundation to erect or silex blocks or pebbles to cement in a mill the nature of cement and its mixture with rock and sand to form a concrete should be understood.

In considering concrete foundations we must recognize that the human factor plays a greater part than where steel or timber is used, for these products come to us with known qualities while concrete has but one ingredient (cement) of the three that is known and we must depend upon a proper selection of sand and rock and the method of mixing and placing it to the judgment of some one who may or may not know the experiences of others. The failures in concrete structures are seldom due to miscalculations of compressive stresses but to the results of vibratory forces and it is imperative that the human factor be elert in taking every precaution to make a good concrete and to form strong

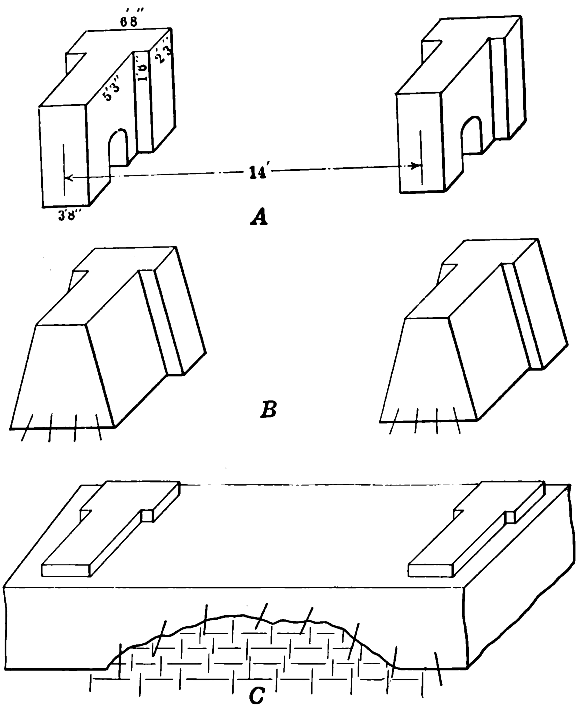

FIG. 58.—Tube mill foundations.

bonds between the machinery and the concrete and the concrete and the bedrock. Some foundations fail from lack of care in selecting sand and rock. To insure against this contingency blocks of concrete should be made of the materials it is intended to use and subjected to physical tests.

Small sizes of concrete mixers run by gasolene or steam engines are so readily obtained at small cost that it will pay to use a mixer even for a small job, but no doubt many millwrights will prefer to mix by hand where but a small amount of concrete is to be placed.

For convenience the cement in concrete is often spoken of as the matrix while the sand, gravel or crushed stone is designed the aggregate. The cement should be kept in dampproof storage until ready to be used, and then used immediately, as rain or dampness causes it to set prematurely and renders it, if not useless, at least undesirable for good work.

Concrete should always be mixed fresh; that which has been mixed over ½ hr. has an initial set and should be rejected. The sand should be hard, and angular and when it contains over 5 per cent. fine dirt is should be washed or screened. The broken stone should be hard and sound and should be washed before using to remove all dirt or dust, so that the cement may adhere to the surfaces. If a particularly rich mixture is required for finishing the top of a structure the cement should be mixed with sand in the proportion of 2 or 3 of sand to 1 of cement.

Cement will not readily take up any more water than is necessary for mixing (16 per cent. by weight is necessary for thorough combination); the excess will come to the top, and does no harm unless carried to the extreme, causing a separation of the cement from the aggregate, thereby weakening the mixture. A good way to judge of the right proportion of water is to fill a mold with the mixed concrete, when, upon striking the surface of the mass with the flat of a shovel, it will quake like jelly and water will appear at the point struck. Wet concrete takes longer to set and eventually becomes harder than when the water is only in quantity sufficient to create adhesion. Concrete of this latter description should be well rammed until water comes to the surface. While setting, concrete should be kept wet, for this retards the setting and, as stated in regard to wet concrete it produces a more lasting structure. It is not necessary for any piece of work to be of the same proportional composition throughout, for the richness of a concrete should be proportioned to the strains it must bear; for example, the base of a foundation being usually larger than the top may be made of a poorer mixture.

If concrete must be placed in freezing weather, it is better to keep the structure above the freezing point with fires than to lower the freezing point with salt. If the sand and rock are frozen, hot water may be poured upon them before mixing. Concrete may even be made with hot water, a safe limit being 150°F. but hot cement sets quickly and is not so strong as a slow setting concrete made with cold water.

To make a concrete of maximum strength it is necessary to know the amount of air space or "voids" in the sand and rock for the spaces in the sand must be filled with cement and those in the rock with the cement and sand mortar. To find the amount of voids in the rock or sand weigh a bucketful and then the amount of water to cover when levelled off. Knowing the specific gravity of the material we may find the amount of voids by the following formula:

$$\frac{W}{\frac{R}{SG} + W}$$

where W is the weight of the water, R the weight of the rock or sand and SG its specific gravity. Another method, avoiding the necessity of knowing the specific gravity, may be devised as follows: Suppose we fill a bucket with water and weigh the water in the bucket, which will be the weight of the bucket and water less the weight of the bucket. Now fill the bucket with rock or sand; catch and weigh the water overflowing which will be the amount of water displaced by the rock. The amount of water remaining in the bucket will be that occupying the air spaces. To find the proportion of voids use this formula:

$$\frac{W - w}{W}$$

where W is the weight of the bucketful of water and w the weight of the water displaced.

To illustrate we will consider that the weight of the rock is 80 lb., the amount of water needed to cover this is 30 lb. and the specific gravity of the rock 2.50. Using the first formula we have

$$\frac{30}{\frac{80}{2.5} + 30} = \frac{15}{31} \text{ or 48.3 per cent. voids.}$$

The sand by the same process has shown 40 per cent. voids; therefore the concrete will require for each 100 parts of rock by volume, 48.3 parts of sand and 19.3 parts of cement, this being 40 per cent. of 48.3. These ingredients will represent a mixture of 1 cement : 2.5 sand and 5.1 rock or familiarly 1 : 2.5 : 5.1. This mixture will stand a compressive strain of about 2,000 lb. to the square inch.

To obtain maximum strength in concrete the voids in the rock and sand must be filled and likewise enough of the cementing material to completely envelop both rock and sand. Engineers usually allow from 5 to 15 per cent. for voids over that found by experiment to cover this condition. If we allow an increase of 5 per cent. to cover this cementing material, we then have a mixture containing 100 parts rock, 53.3 sand and 24.5 parts cement or a 1 :2.2 :4.1 mixture. This will stand a compressive strain of 2,400 lb. per square inch, or a tensile strain of 240 lb. per square inch, taking the extreme ratio. If we consider the strains due to vibrations to be six times those due to tension, the compressive strains on our foundations should not be over 40 lb. to the square inch. As we have but a fourth of this we may consider the mixture sufficiently strong for our purpose. As the base of the block covers more area than the top, the lower portion may be made of a leaner mixture either by increasing the proportion of rock and sand or by ramming in large pieces of rock as the foundation is built up.

At times it is necessary to build a foundation on a clay or shale base. It will then be advisable to proportion the base larger than usual and to anchor the block to the bedrock by drilling or boring a number of holes, "fanning" them out to cover as much area as possible. The holes should be expanded with powder and 1-in. rods that have been upset or barbed at the end cemented in the holes and left projecting into the cement superstructure. Even should the formation be solid rock, the block should be anchored by the same method, but with fewer rods put in more perpendicularly. The surface on which the concrete rests should be uneven to afford a better grip and should be wetted before pouring the concrete. Having excavated a hole about 2 ft. larger each way than the dimensions of the block, build the form of 2-in. planks and hang the bolts or the pipes which will receive the bolts, from cross-beams in the exact places they will occupy. Reinforcement of old steel cables or barbed wire may be used to strengthen the structure. The pipes for the bolts must be wired firmly in place and the form must be well braced to prevent movement while tamping the concrete.

As the concrete is mixed and shovelled or dumped into the form it should be rammed with a heavy pounding iron or a 4 by 4 scantling so shaped that it can be easily handled. If spaces must be filled with boards to give the form the right shape, these blocks must be made so that they can be easily taken out when the concrete has set.

The top of the block should be finished with a couple of inches of cement mortar containing 1 part of cement to 2 of sand and, before the final set the surface, should be levelled. This plan is better than grouting concrete under the casting after it has been put in place; in fact many concrete structures have failed because of this latter method of finishing, because the concrete rammed under the casting makes a poor bond with the concrete block. Bearing in mind the fact that a slow setting concrete is finally stronger than one that sets rapidly, the structure should be covered with canvas and kept wet until the final set has taken place, ordinarily about 7 days. For patching worn spots in a tube mill with pebbles cemented in a matrix of sand and cement 36 hr. is sufficient time to allow for the set, but for any extensive repairs of this character at least 3 days should be allowed.

Cement Data

Portland cement weighs per barrel, net	376 lb.
Portland cement weighs per bag, net	94 lb.
Natural cement weighs per barrel, net	282 lb.
Natural cement weighs per bag, net	94 lb.
Cement barrel weighs from 15 to 30 lb., averaging about	20 lb.
Portland cement is assumed in standard proportions to weigh per cubic foot	100 lb.
Packed Portland cement, as in barrels, averages per cubic foot about	115 lb.
Packed Portland cement based on a barrel holding 3.5 cu. ft., weighs per cubic foot	108½ lb.
Loose Portland cement averages per cubic foot about	92 lb.
Volume of cement barrel, if cement is assumed to weigh 100 lb. per cubic foot	3.8 cu. ft.
American Portland cement barrel averages between heads about	3.5 cu. ft.
Foreign Portland cement barrel averages between heads about	3.25 cu. ft.
Natural cement barrel averages between heads about	3.75 cu. ft.
Weight of paste of neat Portland cement averages per cubic foot about	137 lb.
Volume of paste made from 100 lb. of neat Portland cement averages about	0.86 cu. ft.
Volume of paste made from 1 bbl. of neat Portland cement averages about	3.2 cu. ft.
Weight of Portland cement mortar in proportions 1:2½ averages per cubic foot	135 lb.

Weight of concrete and mortar varies with the proportions as well as with the materials of which it is composed.

Weight of Portland cement concrete per cubic foot after setting:

Cinder concrete average	112 lb.
Conglomerate concrete average	150 lb.
Gravel concrete average	150 lb.
Limestone concrete averages	148 lb.
Sandstone concrete average	143 lb.
Trap concrete average	155 lb.

Loose unrammed concrete is 5 to 25 per cent. lighter than concrete in place, varying with the consistency.

Barrels of cement in a cubic yard of concrete. Divide 10.5 by the sum of the ingredients. For example, a 1:2.3 mixture will contain $\frac{10.5}{6} =$ 1.7 bbl. per cubic yard.

CHAPTER II

AMALGAMATING WITH TUBE MILL

The tailing from a stamp mill or other intermediate crusher often contains considerable oversize, which holds enough metal to pay the cost of regrinding to a finer mesh and extracting the metal therefrom by amalgamation. For this purpose we may use chilian mills, pans or tube mills. The tube mill requires less attention, can be adjusted to produce a granulated or slimed product by altering the feed of ore and water, and has a greater capacity for fine grinding than other machines, and has therefore been chosen to follow stamps with the plates taken away from the battery and put in the tube mill circuit. The plates then receive no coarse material, and the maximum amount of metal is recovered by amalgamation without requiring the stamps to break the ore to a fineness not warranted by their weight.

As this regrinding is not often required finer than −100-mesh the Hardinge conical mill is equal to other grinding machines on the market for this class of work. As a grinder to −200, I have disqualified the Hardinge mill but for this particular purpose it may be equal to the cylindrical tube mill. What follows in respect to amalgamating with tube mills must therefore pertain particularly to the Hardinge conical mill.

Ores may be reground in water or in cyanide solution and amalgamated inside the tube mill or after the pulp has been discharged. For convenience the subject will be treated as follows:

I. General hints on amalgamation.
II. Amalgamating in cyanide solution.
III. Amalgamating inside the tube mill.
IV. Amalgamating in the tube mill circuit.

I. General Hints on Amalgamation

The first requisite of the successful amalgamator is clean mercury for that which has been fouled by grease or impurities will not combine readily with metallic particles. If the mercury is newly bought, or has previously been used, it should be digested with weak nitric acid and stirred frequently to dissolve the impurities. As mercury likewise dissolves in nitric acid an iron nail will cause its precipitation, the iron taking the place of the mercury. The iron salts may be washed off with water and should grease still appear on the surface a little metallic sodium will get rid of this, but do not add enough sodium to cause an iron nail to amalgamate.

If the mercury must be retorted, conduct the process at the lowest heat possible and keep the mercury coming off last for future distillation, as it contains most of the volatile impurities. If the distilled mercury is condensed in a vessel with dilute sulphuric acid, the impurities coming over with the mercury will be dissolved and the latter left bright and clean.

Whether amalgamating on plain copper plates or on those that have been silver plated it must be kept in mind that an amalgam retains metallic particles better than mercury alone, the practical application of which is that all amalgamated surfaces should be covered with a layer of amalgam of the right consistency to catch and hold the metals amalgamated. When the plates are cleaned or amalgam taken off, leave enough on to form a good surface for future work. If bare copper plates are used, the amount of amalgam left to cover the plate must be greater than when the plate is electroplated with silver, because copper is more readily oxidized and the oxidized compounds of copper form a scum on the surface of plates which prevent amalgamation.

An excellent rule to follow is to avoid the use of chemicals on plates. There are a few compounds, such as soap (mineral) and water, sal ammoniac and weak washes of soda, that might be of benefit and are harmless, but the majority of "dopes" used on plates are of temporary benefit only and when their effect wears off the plates are in worse condition than formerly. Most chemicals and acids, while not particularly affecting copper or mercury, may affect the constituents of the ore and with them forms compounds that are far from desirable.

Verdigris is a salt of copper which appears on copper plates when the covering of silver or amalgam is too thin to protect the plate from corrosion. It may be due partly to the effect of sulphates in the ore or to the corrosion of copper in the amalgam. The remedy is to clean off the coating with chamois leather, rub in amalgam frequently until the spot is thickly coated and let this amalgam remain on the plate, taking off the excess only, every day. If the verdigris still persists in coming to the surface, follow the procedure outlined above and in addition let water flow over the plate immediately after the plate is brushed to prevent the air from reaching the surface. If gold amalgam is not available prepare silver amalgam by dissolving silver in nitric acid and after evaporating to dryness add mercury and a few bright nails and stir to a smooth paste. If the amalgam is prepared from a copper coin when the coin is dissolved in nitric acid, evaporate and fuse to render the copper oxide insoluble, then dissolve the nitrate of silver in water and proceed as in the first instance.

The inclination of copper plates used for amalgamating should be such that the pulp flows gently over the surface in a thin layer. There must never be a bounding stream nor should the plates be exposed to the air. If the pulp is thick, the grade should be greater than for thin pulp, the extremes being 2½ and 1½ in. to the foot.

The amount of surface used for amalgamating is determined to some extent by the process used for gold recovery. If the cyanide process follows the plates, it is necessary only to extract the gold not easily dissolved in cyanide, that is the coarse particles which may have been flattened and floated in the tube mill. If amalgamation is the sole method of gold recovery, the plate area must be such that all the gold is amalgamated.

Plates must be dressed whenever the surface shows any foulness, which may be twice a day or oftener, once to take off the excess of amalgam and at other times to clean the surface. In tube mill amalgamation, when a plate is being cleaned, the pulp must be switched to another plate and the process conducted as in stamp mill amalgamation, except that the plate must be better protected with amalgam. If the cyanide process is used, the plates are kept wetter by reason of the greater amount of mercury used in the mill. The plate is sprinkled with mercury and well rubbed with a whisk broom. A rubber scraper is then used to gather the amalgam by first scraping down the sides, then toward the center and then toward the head of the plates where the amalgam is taken off in a scoop. The plate must not be robbed entirely but enough amalgam left to form a smooth plastic coating. Plates 16 ft. long may be cleaned and brushed at the rate of 7 min. a plate, but if the ore is particularly refractory the time required may be twice as great, as the plate will probably be covered with compounds which require careful removal.

If chemicals are used on plates, let it be sparingly for any ores that require chemicals to promote amalgamation contain substances that form compounds with the chemicals that prevent amalgamation.

II. Amalgamating in Cyanide Solution

The application of the cyanide process to the extraction of the precious metals from ores has greatly limited the usefulness of amalgamation, for the reason that no amalgamation process alone thus far devised will save a commercial percentage of the total content of metal in the more refractory ores, while a cyanide process will. The practice, therefore, tends to discard the amalgamation process, unless there is coarse gold in the ore, which can be caught with mercury when ground to a slime, but which will not readily dissolve in cyanide. If we use the cyanide process entirely, we are able to grind faster at less expense for it is not necessary to arrange the intermediate crusher, such as stamps, for both crushing and amalgamating, but for crushing only.

There is a diversity of opinion as to the effect of cyanide of potassium on amalgamated copper plates when crushing in cyanide solution, due no doubt to the diverse character of the ores treated. If the plates are protected by a thick coating of amalgam, cyanide solution may be used with little fear of the plates being corroded, but a greater percentage of gold will be amalgamated without its use. In a cyanide process the loss is more than compensated by the gain in extraction and the simplification of the process due to adding the chemical in the mill water. There are many mills crushing in cyanide solution with nothing extraordinary to note, but some show conditions that are of general interest. A few examples will suffice.

C. J. Stone says in relation to the Free Gold mill that

> "The amalgamation in the cyanide solution is unsatisfactory, only about 10 per cent. of the total gold being recovered on the plates. The solution hardens the plates and renders them almost of no value for amalgamating after 3 or 4 months' run."

Another authority says in respect to the Liberty Bell mill:

> "The recovery by amalgamation is materially smaller than in previous years of water-amalgamation; the process is more expensive both in labor and material, and requires more skill; and the consumption of cyanide is considerable. . . . The plates are kept rather wet and any drip is caught in a trap."

The following quotation is from another authority:

"The solvent action on amalgamating plates greatly depends upon the amount of amalgamable gold in the ore. If the ore contains much gold and the plates are not cleaned too closely, they will last indefinitely, being protected by a coating of amalgam. If the ore be low-grade, it is next to impossible to keep the plates in good "catching" condition; they are rapidly eaten up by the solution. . . . I find no advantage whatever in using silver-plated copper when crushing in cyanide. This is contrary to water practice. The plain copper after the first careful dressing catches and builds amalgam splendidly, and does not harden so much as silvered plates."

At the Alaska Treadwell[1] concentrate plant:

"When grinding in cyanide solution stronger than 1 lb. per ton, followed by amalgamation, it was difficult to keep the plates bright, due to a dull white surface deposit, which if allowed to remain turned to a dull gray. A muntz metal plate was substituted for the copper plate, but as all the plates were silver-coated no variation in the result was noted."

John Gross says:

"It sometimes becomes advisable to attempt the saving of a portion of the precious metals in ores that are too coarse for the ordinary system of cyanidation, owing to the high strength of cyanide solution carried, or an excess of lime in the solution. I have seen plates almost entirely eaten through in the course of several months, using a battery solution containing 2 lb. of KCN and carrying a protective alkalinity of about 1 lb. of lime."

III. Amalgamating in Tube Mills

The most conspicuous example of amalgamating in a tube mill is no doubt the unique process of amalgamating the native silver ores at the Nipissing High-Grade mill, Ontario, Canada. While we have used an amalgam barrel for many years to amalgamate sand and scrap in a stamp mill the Nipissing mill has carried the idea to an extreme that the old amalgamators never dreamed of. The tube mill used is 3 ft. 11 in. by 19 ft. 8 in. running at 37 r.p.m. with 55 per cent. moisture and practically 100 per cent. -200-mesh in the product. A 5 per cent. solution of cyanide is used and 8,500 lb. of mercury to 6,500 lb. ore. The ore treated contains a high percentage of arsenic which under ordinary

[1] "The Cyanide Plant of the Alaska Treadwell Mines," *Transactions* American Institute Mining Engineers, October, 1911.

amalgamating conditions would be a difficult ore to treat. H. A. Megraw says:[1]

"The tube mill has its axial entrances sealed except for a small compressed-air pipe which enters it at one end and a corresponding air exit at the other end. . . . Here then is a tube mill used as an amalgamating barrel, but under conditions which were probably never before sought for amalgamating purposes. The combination of an extremely high cyanide solution, a complex ore which contains all sorts of elements, and mercury all ground violently in a pebble mill, would seem to indicate the reverse of satisfactory. . . . The amount (of cyanide) originally added to the tube mill is sufficient to produce a very strong solution, which is capable of partially outlasting the destroying effect encountered within the mill. The agitation is continued for 9 hr., after which time 97 per cent. of the contained silver has been extracted from the ore."

At the Plymouth Con., California, gold is amalgamated inside a Hardinge mill and on plates.

"The discharge from 30 stamps is classified and the coarse material ground in a Hardinge mill, the discharge from the Hardinge mill passing over two 5 by 8-ft. copper plates, at the top and the bottom of which are traps or wells of about 4 in. in depth. The upper traps catch as much as 70 per cent. of the gold without the use of quicksilver, it being used solely on the plates, which are cleaned once a day. After passing the first pair of plates the ore passes over a second set of ten 6 by 12-ft. plates, where a further recovery is made. The present system of amalgamation is proving both simple and efficient."[2]

The following description of the amalgamation of gold inside the Hardinge mill at the Plymouth Con. mill, by the millman who actually attended to the feeding of the mercury, is particularly interesting as first-hand evidence of what actually took place.[3]

"When I arrived at the plant referred to, amalgamation was done on the plates only. Shortly after, this was changed to feeding quicksilver to the Hardinge mill, with a marked improvement in results. Later, this was again changed to feeding quicksilver to the batteries, with a further improvement in results, as we were informed. During the first part of the campaign the amalgam on the plates was maintained very hard, being removed by scrapers, one of the plates being cleaned up each 24-hr. period. The Hardinge mill was cleaned up at the end of the month, by having the manhole cover removed and the screen adjusted,

[1] *Engineering and Mining Journal*, Dec. 14, 1912.

[2] *Mining and Scientific Press*, Editorial, Feb. 27, 1915.

[3] FRANCIS O'BOYLE, *Mining and Scientific Press*, March 20, 1915.

and the mill was revolved and all sand and possible amalgam was allowed to fall out. The screen was then removed, and a man entered the mill to see what was inside. Considerable amalgam was found in the spaces between the silex bricks, but by far the largest amount of amalgam found in the Hardinge was stuck fast inside the throat of the mill at the points marked *A* and *B* on the diagram, Fig. 59, and was very hard, a hammer and a cold chisel being necessary to remove it. Considerable amalgam was also found in the throat, both at feed and discharge ends, and also in the feed scoop, stuck tight, and I recall that we removed the scoop to get at it better. No hard balls of amalgam were found in or around the Hardinge mill, but we did get the hard balls in the batteries around the dies. Also, the small cones, feed, and discharge boxes of the Hardinge mill and the distributing box above the plates seemed favorite places; hard chunks would build up in most un-

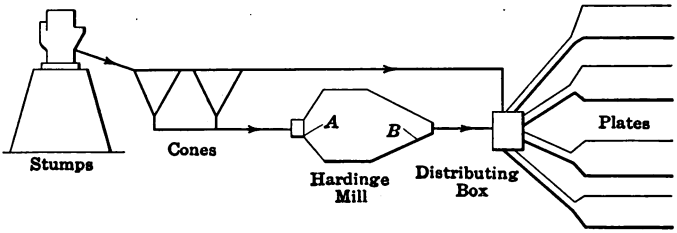

FIG. 59.—Flowsheet at the Plymouth Consolidated Mill.

expected places. Nail heads in launders and boxes were always covered with amalgam. I do not recall that we got anything from the sand and loose material inside the Hardinge mill.

"Later on, the plates were kept softer and the amalgam more pasty; scrapers were scraped, and the rubber used. Amalgam built up faster on the plates, one being cleaned every day, or more than one if it seemed advisable. At the monthly clean-up the Hardinge mill was a disappointment, also the mortar boxes, and the largest quantity of amalgam on this run was obtained from the plates; we were told that the extraction had reached expectations. I have no means of knowing the mercury losses, but do not believe they were excessive. The ore was principally limonite, some quartz and clay. A Pierce amalgamator placed below the plates never got anything but dirt, although dressed daily once each 8-hr. shift."

It must have been after the above description was printed that the Hardinge amalgamators were attached to the discharge ends of the mill for O'Boyle makes no mention of them. Fig. 60 shows the discharge end of the Hardinge mill which is practically the same as is in use at the Amador Con. mill. Small copper balls are fed into the amalgamator which build up with amalgam as may be seen in Fig. 61. For comparison of size the balls are arranged alongside a 3-ft. rule. It will be seen that eight of them occupy a space of 9 in. Fig. 62 is a photograph of the chunks of amalgam found in the throat of the amalgamator and stuck thereto. A 3-ft. rule is photographed with the pan for comparison of size.

It may be well to remark that an amalgamator such as shown can be attached to any tube mill.

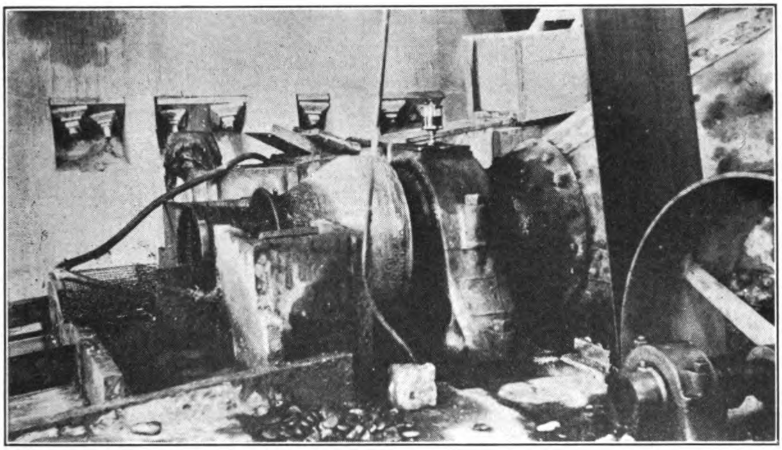

FIG. 60.—Hardinge conical mill with amalgamator attached.

Mr. W. H. Hardinge says[1] that the mill superintendent stated that the loss of quicksilver was 0.025 oz. per ton of ore ground.

At the Amador Con. milling plant, California, the discharge from the Hardinge mills goes to Hardinge amalgamators, which are attached to and revolve with the mill. Mercury is fed into these amalgamators as required by the condition of the outside plates, in the same manner as feeding it into a stamp mortar. About a dozen small copper balls, made from ¼- or ⅜-in. wire, are kept in the amalgamator. As these balls roll around they build up, forming good-sized amalgam balls which are removed daily while the mill is in operation; after removing the accumulated amalgam, the balls are returned to repeat the operation.[2]

We are indebted to H. W. Hardinge for the conical mill and

[1] *Mining and Scientific Press*, April 17, 1915.

[2] T. S. O'BRIEN, *Engineering and Mining Journal*, Aug. 14, 1915.

for some useful information contained in an article in *Mining Press*, Feb. 13, 1915, from which we quote the following:

"This matter of inside amalgamation was suggested by me to J. M. Elmer, in 1909, when he was in charge of a Mother Lode mine in Tuolumne, Cal. He then tried it out with highly satisfactory results. Mr. Elmer later reported that the amalgamation recovery advanced from 65 to between 80 and 85 per cent., and a total of over 90 per cent. was made with subsequent concentration. From information later gained, however, we must give A. D. Foote, of the North Star mine at Grass Valley, credit for a still earlier use of quicksilver in pebble mills for inside amalgamation. Some months after Mr. Elmer's resignation from the

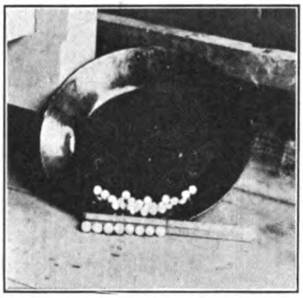

FIG. 61.—Balls of amalgam from Hardinge mill.

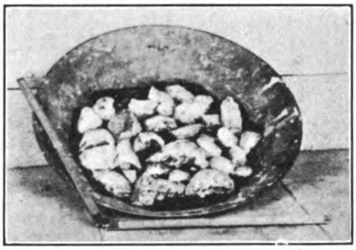

FIG. 62.—Chunks of amalgam from Hardinge amalgamator.

Tuolumne mine, we received a letter from Charles Maass, Mr. Elmer's successor, and from his letter I quote the following:

"'The objection usually urged against inside amalgamation, where fine grinding is done, is that the mercury will granulate and thus become useless for amalgamating purposes. This is due to the mercury becoming foul through taking up soluble mineral salts. To prevent this, it is necessary to introduce some agent which will cause the liberation of hydrogen gas from the liquid in the mill, thus deoxidizing the salts when they no longer have an affinity for the mercury. In other words, it is simply necessary to keep the mercury as nearly chemically pure as possible. To accomplish this we have found that, in the case of the silex-lined mill, metallic sodium gives excellent results. In our iron-lined mill the results were not as good, as the action of the metallic sodium was so violent that it caused the amalgam to adhere to such parts of the iron lining as had become brightened through wear. In this case we used an amalgam made up of zinc and aluminum. This also has the effect of liberating hydrogen gas, though the action is slower and will not cause the amalgam to adhere to the polished iron. The objection to the use of this combination lies in the fact that it cannot be made up in quantities to keep, and as a consequence must be made up freshly whenever necessary to add any to the mill.

"'To sum up: For successful inside amalgamation in your mills it is necessary only to keep the mercury as nearly chemically pure as possible, and this can be accomplished by the introduction of any agent which will produce nascent hydrogen. As to quantities and proportions, this will vary according to conditions and the nature of the ore treated and is a matter which can be easily determined by any experienced amalgamator. Mercury should be fed to the mill as freely as possible but not in such quantity as to cause it to run on the copper plates. It is not our object to catch the amalgam inside the mills, as its recovery there is an expensive matter, but to prepare the amalgam so that it will readily adhere to the copper plates when it leaves the mill.'

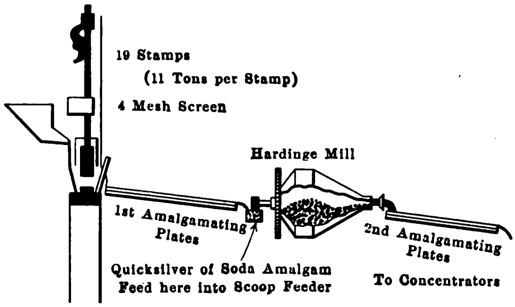

FIG. 63.—Plan of work at Tuolumne. (Hardinge.)

"In a plant at Jarbridge, Nev., where excessive losses between heads and tails could not be accounted for, an expert was brought in to check up results, who later reported that all he did was to open the mill, dump the contents, retort three separate balls of 50 per cent. amalgam, and hand the manager $16,000 worth of bullion.

"Here it was explained that the cause of the trouble was in too small a quantity of quicksilver being added to the charge in proportion to the gold content. As soon as a quantity of quicksilver sufficient to make a fairly liquid amalgam was introduced into the mills, the amalgam issued with the charge and was recovered on the plates."

The system as practised at Tuolumne is outlined in the sketch, Fig. 63.

At the Incaoro mill, Bolivia, mercury is fed into the tube mill every hour, the mill being lined with flint pebbles 6 in. in diameter set in cement and lumps of hard quartz from the mine are used instead of pebbles for grinding. The clean-up takes place every 15 days, the amalgam being found distributed from the ball mill to the pebble mill but no loose balls of amalgam are found inside the mills. Three-quarters of the amalgam is collected from inside the mills and one-quarter from the plates.[1]

These few examples will serve to indicate that we may advantageously amalgamate gold ores inside a ball or tube mill, either with coarse or fine grinding and that the success thus far attained opens a new field for the amalgamator whose prestige of late has been overshadowed by the cyanide chemist.

J. W. Pinder[2] states the four factors governing the success of amalgamation in tube mills or other mills that violently grind gold ores in the presence of mercury to be: The proper consistency of the pulp, the proper quantity of mercury, a subsequent mercury bath and the proper capacity of the machine.

He says:

"1. The pulp should be thick enough to hold the fine globules of mercury in thorough suspension as it revolves throughout the tube. About 75 per cent. (by volume) of water would, in most cases, result in the best consistence, although that must be determined by conditions and the character of the ore. When too thin, the pulp cannot be well amalgamated, and the mercury is likely to be whipped into a sickened condition. The consistence of soft mortar is therefore the best, and the safest. While grinding, the pulp should not be flooded for any purpose.

"2. The quantity of mercury to be used should be determined entirely by the content of the ore and consistence of the pulp. According to the facility with which the gold is amalgamated, the quantity of mercury should be regulated. If the ores contain zinc, lead, arsenic, antimony, talc, or graphite, or any other substance tending to affect the activity of the mercury, more mercury should be added to lessen such bad effects, in proportion to the detrimental material contained in the ores. In such cases it may become necessary to use as much as can possibly be carried, in fine globules, by the stiff pulp, in proper suspension without running together to form larger bodies in the mill. But with free ores, or comparatively so, half the quantity would suffice.

"3. The subsequent mercury bath is all important to the success of this operation. The lesson may be taken from the performance of the settler, in pan work, for collecting the amalgam and preventing the loss of mercury. This bath should be ample in capacity, and permanent, changed when showing signs of requiring quickening; and the proper dilution of the pulp as it enters the bath.

"4. The quantity of ore treated, within economical reason, should depend upon the quality of the work. The capacity of the machine is a

[1] GEORGE R. PRINGLE, *Mining and Scientific Press*, May 29, 1915.

[2] *Mining Press*, Feb. 27, 1915.

matter of judgment, to be based upon results obtained. Amalgamation is a delicate and an important operation in itself. To do good work one should not try to make a tonnage record for grinding and expect to get the best extraction at the same time."

IV. Amalgamating in the Tube Mill Circuit

Amalgamating gold ores on plates in the tube mill or classifier circuit has, in cyanide plants, succeeded the time-honored position of the plates in front of the battery, especially in large mills where the amalgamating may be carried on in a separate department. If the process involves the use of plates after the classifiers, the pulp will be dilute and the plates will have a low inclination but if the plates are placed immediately at the tube mill discharge before dilution, the plates must be steep to take off the thick pulp.

The effect of taking plates away from the battery and putting them in the tube mill circuit is indicated by the following experience in South Africa:

Mill	Per cent. amalgamated	Area of plates	Total extraction
Simmer Deep:			
Before........	56.80	5,276	93.40
After.........	57.70	1,700	93.50
Randfontein:			
Before........	54.46	.....	92.06
After.........	48.60	.....	92.94

The net result was that the amount of gold amalgamated remained the same, the area of the plates was greatly reduced and the total extraction slightly increased. The pulp in the tube mill circuit contained 55 per cent. moisture and the plates have a grade of 18 per cent. or 2.16 in. to the foot.

At the Robinson Deep mill, South Africa, where amalgamation is practised in both circuits of stamps and tube mills, the percentage of total recovery is as follows: From battery plates, 47 per cent.; from tube mill plates, 23 per cent.; total extraction, 70 per cent.; and recovery by cyaniding, 30 per cent. From these figures it must not be assumed that the recovery from amalgamation has been increased 23 per cent. from what could be obtained from battery plates alone, for by the use of the tube mill the ore is crushed to a coarser mesh in the stamps, and the tube mill continuing the process releases gold that could be less advantageously recovered by the stamps alone.

At the Rainbow mill, Oregon, the tube mill discharge with 78 per cent. -100-mesh is diluted to 60 per cent. moisture and run over plates with a inclination of 2¼ in. per foot. The extraction was increased about 10 per cent. by regrinding in the tube mill and the trouble due to lime blackening the plates, when amalgamation followed the batteries, was rectified by the finer grinding. The mill grinds in cyanide solution averaging 1 lb. KCN and 0.6 lb. protective alkalinity. Plates are dressed twice a shift, using fiber brushes, whisk brooms and scrapers cut from rubber belting. The plates are not allowed to build up with hard amalgam for when this was allowed the extraction fell off considerably. Plate consumption is rather high, due to the dissolving action of the solution—a plate lasting from 3 to 4 months—and for this reason silver-plated plates are not used. There is no noticeable difference in plate extraction on this account. The presence of lime in the mill solution caused the plates to blacken and rendered them unfit for amalgamation when recovery was entirely on the battery plates, but when the ore was reground and amalgamated after leaving the tube mill this difficulty disappeared.[1]

At the Big Pine mill, Nevada, a 5 by 20-ft. tube mill at 28 r.p.m. is used as intermediate and final grinder taking 1-in. mesh product from a 10 by 16-in. Blake. Amalgamation takes place on plates 12 ft. long; two of them are 4 ft. 9½ in. wide and one is 4 ft. 6 in. wide. They have a grade of 1¾ in. per foot. After leaving the plates the pulp is elevated to a classifier, the oversize going to the tube mill. The mill has a big capacity, due no doubt to the soft nature of the ore for with this coarse feed it is said to grind 130 tons a day, 85 per cent. -200-mesh. Care is taken that no mercury gets into the tube mill.

At the Dome mill, Ontario, instead of the plates being in the tube mill circuit they are placed after the classifiers, there being four classifiers and four copper plates, each 108 by 144 in. with 1½ in. per foot grade. Although with a 16-mesh screen on the stamp batteries the percentage caught by amalgamation was 78 per cent. and with a 10-mesh screen 46 to 50 per cent., it was found that the coarse-mesh screen was necessary to attain capacity and as the sand scoured the battery plates abnormally with

[1] M. W. Dake, *Engineering and Mining Journal,* June 26, 1915.

this screen on the battery, the plates were taken from in front of the battery and put after the classifiers. While the percentage of gold caught by amalgamation was less than formerly, all the gold, which might have caused trouble in the cyanide department, was caught on the plates. The stamp mill crushes in water with 7½ water to 1 of ore, lime being added in the battery water at the rate of 3.2 lb. per ton of ore.

CHAPTER III

GRINDING ORES WITH THE TUBE MILL FOR FLOTATION

The tube mill or the ball mill is an ideal grinder for preparing ores for oil flotation because not only is it an excellent grinding machine but likewise it is a good agitator, thus combining the two features in one machine, and when the oil is added to the mill with the mill-feed no other method of mixing is required.

The degree of fineness required to save the highest economical percentage of the metallic contents of an ore will in great part determine the class of machine used for grinding. When a 60- or a 100-mesh product is required some type of ball mill or the Hardinge conical mill will no doubt be the favorite, the short cylindrical tube mill of big diameter and short length being a close competitor; when fine grinding is required the cylindrical tube mill from 12 to 18-ft. long will no doubt be preferred.

Most of the flotation plants are now concentrating base metals where as a rule fine grinding is not required but when this method of concentrating becomes common, for precious metal-bearing ores, finer grinding will be the rule. Tests made at a mill in Mexico indicated "that fine grinding was necessary for good results in flotation." When the mill heading was crushed to 60-mesh, the tailing from flotation assayed 0.08 oz. gold and 11 oz. silver; when the same ore was crushed to 100-mesh, the tailing assayed 0.04 oz. gold and 5 oz. silver; when the crushing was carried to 200-mesh, the tailing assayed 0.02 oz. gold and 3.75 oz. silver.[1] This experience will no doubt be duplicated in many localities so that the long cylindrical tube mill now used for preparing ores for the cyanide process can be used for grinding ores for flotation. Fig. 50 illustrates in detail the class of mill that will probably be used for grinding gold ores for flotation in closed circuit with a classifier.

A few examples of the degree of grinding and the machines used at various plants will serve to indicate present-day practice, at least for the flotation of base-metal concentrates.

[1] *Mining and Scientific Press*, July 24, 1916, 123.

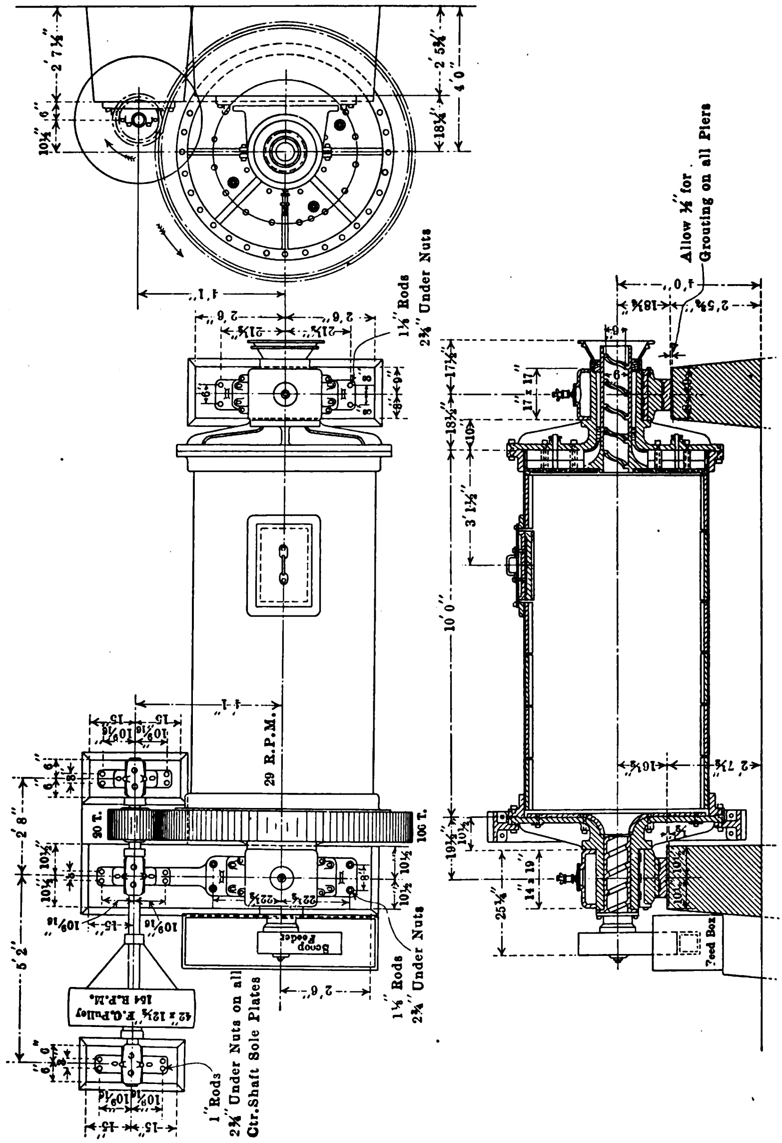

FIG. 64.—Details of a 5 by 10-ft. adjustable discharge tube mill.

At the Braden Copper Co. the ore is agitated with pine-tar oil in Hardinge conical mills. These mills are all run in closed circuits with the object of producing the least possible + 60-mesh solid in the finished pulp. The following screen analysis will show the class of work done by the Hardinge mills:

	Screen analysis, feed, per cent.	Discharge, per cent.
On 20-mesh	24.3	
On 40-mesh	54.5	12.0
On 60-mesh	15.8	11.2
On 80-mesh	5.3	8.8
On 100-mesh	3.1	9.2
Through 100-mesh	6.0	
On 200-mesh	.....	10.4
Through 200-mesh	.....	48.4
	100.0	100.0

Ground per 24 hr., tons	86.4
Pebble load, kilogram	5,200.0
Horsepower	51.9
Water-solid ratio pulp	0.48–1.0

At the reconstructed Anaconda mill the ore is ground in 10 by 4-ft. Hardinge mills and 8 by 12-ft. tube mills in closed circuit with classifiers. The following screen analysis is typical of the flotation feed:

Screen size		Cumulative, per cent. solids
Square mesh	Aperture, sq. mm.	
+ 16	1.180	0.3
+ 24	0.730	1.3
+ 40	0.430	3.0
+ 60	0.260	5.8
+ 80	0.210	12.8
+ 110	0.130	38.5
+ 130	0.110	42.3
+ 160	0.085	54.8
+ 200	0.076	59.3
+ 240	0.063	62.8
− 240	0.063	37.2

This is practically a 60-mesh product. In this case the oil is added ahead of the grinders and agitated in the tube mill.

At the Lewis mill, Battle Mountain, the grinder is of the Marcy type of mill, diameter 6 ft. and 3 ft. long lined with manganese-steel step liners. The capacity of the mill is 100 tons a day to pass 100-mesh screen. The oil mixture is fed into the mill with the ore.

At the Engels Copper Mining Co., Plumas Co., California, the rolls deliver a ½-in. product to 6 by 5-ft. ball mills using forged-steel balls 2 to 5 in. in diameter. The overflow from the classifier passes direct to the flotation machines while the oversize from the ball mills passes to a 7 by 10-ft. tube mill working in a closed circuit with a classifier. A screen analysis shows that 5 per cent. will remain on 100-mesh and 65 per cent. will pass 150-mesh screen. Half the oil is added in the tube mill. The circulating load in the tube mill circuit varies from 400 to 500 tons a day.

At the Inspiration Copper Co. the ore is ground with oil and water in 8 by 6-ft. Marcy mills to about 40-mesh screen, tonnage 425 tons per 24 hr. per ball mill. Steel ball (chrone) consumption 1.79 lb. per ton ground. Power consumption 9.86 kilowatt-hr. per ton.

At Maimi, Hardinge mills are used: at the Brittania copper mine a tube mill 7 by 12 is used while at the Gold Hunter mill the grinding is done in a 5 by 14-ft. tube mill.

At the Magma Copper Co.'s plant, Arizona, a Symons 24-in. disc crusher is used for intermediate crushing. The fine grinding is accomplished by a Marcy chrome-steel ball mill and a Chalmers and Williams tube mill, the former reducing to 12-mesh and the latter to − 60. The Marcy mill product goes to the tube mill to be reground, thence to the flotation system.

These few notes on oil flotation are sufficient to indicate the growing importance of the tube mill in the metallurgical treatment of base as well as precious metal-ores. It may be that the tube mill will in future take the place of the intermediate crushers as well as hold the place it fills so well at the present time, that of taking the product from the intermediate crushers and finishing it for flotation, amalgamation or cyanidation. The use of chrome- or manganese-steel balls will make this possibility an accomplishment, judging by present indications.

CHAPTER IV

CRUSHING EFFICIENCIES

It is necessary to have a means of comparing the work of a tube mill or other crushing machines grinding to various sizes so that the most effective range may be determined and the successive stages of crushing may be proportioned to give the maximum efficiency for each machine. For this purpose we must use a mathematical formula which will contain the necessary elements and which is consistent with practical experience. While the problem appears at first sight an easy one, the more it is studied the more complex it becomes, by reason of certain factors which cannot be determined with accuracy.

The problem is to determine the amount of work done in crushing ore, work being the overcoming of resistance through space, represented by the formula, work = distance × resistance.

The work done in crushing particles of rock to smaller dimensions varies in proportion to the surface exposed in crushing or to the ratio of the original to the final diameters. This is known as the Rittinger theory.

Opposed to this we have what is known as Kick's law which states that "the energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

While personally I favor the Rittinger hypothesis, both theories have their advocates. In most of the cases I have worked out the two methods give nearly the same results. I quote from a recent article by O. A. Gates[1] who is the originator of the crushing surface diagram method of computing crushing efficiencies based on the Rittinger theory:

"Some 2 years ago there was published in your columns a note on the rock-crushing tests then being conducted at McGill University, the preliminary results of which indicated that Stadler's theory of Kick's law was correct. Recently the secretary of the Canadian Mining Institute called my attention to an editorial in their October *Bulletin* referring to my paper presented in the September *Bulletin* of the American Institute of Mining Engineers, but more particularly giving preliminary conclu-

[1] *Mining and Scientific Press*, March 11, 1916, p. 365.

sions on the work that has been done at McGill up to date. The editorial concludes thus:

'Nearly 200 tests have been made at McGill during the past 2 years and we understand that each series will demonstrate convincingly the fallaciousness of expressing power in terms of "energy units," as proposed by Mr. Stadler in accordance with Kick's law. Rittinger's hypothesis, on the other hand, is supported in so satisfactory a degree by the results obtained by actual experiment as to appear quite dependable. It seems probable, therefore, that the investigations made at McGill and Purdue universities will result in terminating the long standing controversy between the supporters of respectively Rittinger and Stadler; while if, as now seems likely, it will conduce to the definite establishment of a correct basis for calculating the efficiency of rock-crushing machines, a work of great utility, by reason of its practical value to millmen, will have been accomplished.'"

The following table shows the mechanical values for the Tyler standard screens for various formulas used in calculating crushing efficiencies taking the -200-mesh arbitrarily at 0.002 in.

Constants for Tyler Standard Screen Sizes

Mesh	Size of opening, inches	Stadler, $-10 \log S$	Log D/d	D/d (Rittinger)
1.0	0.7420	1.2	0.11	1.3
1.5	0.5250	2.8	0.27	1.9
2.0	0.3710	4.3	0.43	2.7
3.0	0.2630	5.8	0.58	3.8
4.0	0.1850	7.3	0.73	5.4
6.0	0.1310	8.8	0.88	7.6
8.0	0.0930	10.3	1.02	10.7
10.0	0.0650	11.8	1.19	15.5
14.0	0.0460	13.3	1.33	21.7
20.0	0.0328	14.8	1.48	30.5
28.0	0.0232	16.3	1.63	43.1
35.0	0.0164	17.8	1.78	60.9
48.0	0.0116	19.2	1.92	86.2
65.0	0.0082	20.8	2.08	122.0
100.0	0.0058	22.3	2.23	172.0
150.0	0.0041	23.8	2.38	244.0
200.0	0.0029	25.3	2.58	344.8
-200.0	0.0020	27.0	2.70	500.0

Under the heading of "Volume of Feed" R. T. Misler makes a comparison of the work of a tube mill with varying amounts of feed in which he uses constants based on the Rittinger theory. A perusal of his work will give an idea of the method of computation.

The method employed is first to make a screen analysis of the ore entering and leaving the mill. The percentage of each grade is multiplied by its energy number and the difference of the sum of all the energy units in the feed and discharge represents the number of energy units used in crushing. This figure is divided by the number of tons crushed per unit power. By this method the work done by two mills operating under like conditions or one mill operating under unlike conditions can be compared with a fair degree of accuracy.

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In all crushing problems the mill product must be screened before and after going through the machine, and when we state that a certain per cent. of the output will pass a certain screen it becomes of prime importance that the size of these grains be fixed beyond dispute. In the following pages this subject is briefly treated. It will be good policy in future to either use the Institute Mining and Metallurgy screen scale or that known as the Tyler standard screen scale, the latter being used in the United States while the former is used mainly in South Africa.

Gage.—The (U. S.) Steel Wire gage[1] or Washburn and Moen gage ("W & M") is the standard for iron, steel, and tinned wire cloth, the old English gage for brass, copper and bronze wire cloth and the size of needle punched and slotted steel or iron plates.

The diameter of wire used in screens should be measured with a micrometer as shown in the illustration, Fig. 65, and recorded in decimals of an inch. This method is far better than with a disc

[1] This name is recommended by the U. S. Bureau of Standards. It was also known as the American Steel and Wire Co. Gage.

gage because it shows decimal not vulgar fractional sizes and because fine wires are liable to stretch when forcing the wire in the notch of the disc.

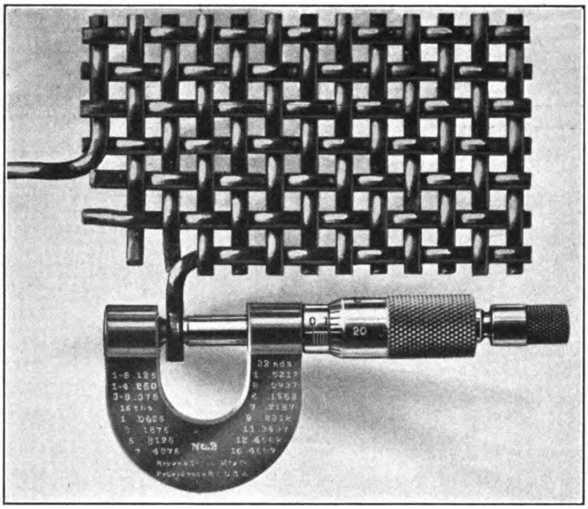

FIG. 65.—Micrometer used for measured wire.

Mesh.—The work "mesh" in wire cloth means the number of openings per lineal inch, measured from center of wire, Fig. 66, and does not indicate the size of the opening or "spaces" unless

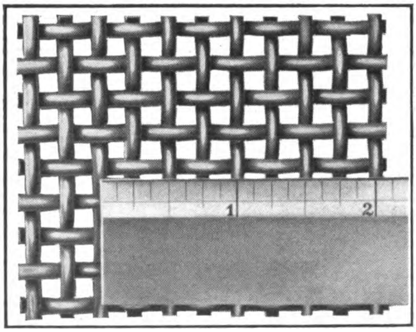

FIG. 66.—Mesh of wire screen.

the size of the wire is also known. For example, a 30-mesh steel wire cloth may be made of wires between No. 27 and 36 giving size of openings between 0.0163 and 0.0243 in., or a 24-

mesh screen No. 24 wire will give about the same size opening as a 35-mesh cloth with No. 34 wire.

In 1907 the Institute of Mining and Metallurgy after exhaustive study adopted screen sizes shown in the following table where the size of the openings is the same as the size of the wire, giving a constant ratio of unity between the area of open space and of metal.

Aperture, inch	Mesh per linear inch	Size of wire, inch
0.0500	10	0.0400
0.0250	20	0.0250
0.0166	30	0.0166
0.0125	40	0.0125
0.0100	50	0.0100
0.0083	60	0.0083
0.0071	70	0.0071
0.0062	80	0.0062
0.0055	90	0.0055
0.0050	100	0.0050
0.0042	120	0.0042
0.0033	150	0.0033
0.0025	200	0.0025

Makers of screens have such different ideas of mesh values that it is impossible to order screens without stating the size of aperture required. The table below will show how makers gage their screens.

DIAMETERS OF SCREEN OPENINGS

Number meshes per inch	Needle size	Commonly accepted values	Tyler standard wire	Tyler mining wire	Johnson and Chapman steel	Ludlow Saylor wire mining
12	1	0.0580	....	....	0.058	0.0420
14	2	0.0490	0.0460	....	0.049	0.0360
16	3	0.0420	....	....	0.042	0.0340
18	4	0.0350	....	....	0.035	0.0300
20	5	0.0290	0.0328	0.0340	0.029	0.0270
24	6	0.0270	....	....	.....	0.0230
25	7	0.0240	....	....	0.027	
26	8	0.0220	....	....	.....	0.0210
28	9	0.0200	0.0322	....	.....	0.0180
30	10	0.0180	....	0.0198	0.024	0.0170
35	11	0.0165	0.0164	0.0170	0.022	0.0140
40	12	0.0150	....	0.0150	0.020	0.0115

For laboratory work we must use screens that have fixed mesh values and ones that can always be duplicated in any locality. As all screening work on samples is done in the laboratory and not in the mill, when we say that a certain proportion of the mill pulp will go through an 80-mesh screen or as expressed by metallurgists −80-mesh, we mean that in the assay laboratory using a standard set of screens a certain per cent. will go through that screen; therefore, every screen should have a fixed standard value.

THE TYLER STANDARD SCREEN SCALE SIEVES

Various bases for the starting point in screen scales have been proposed. As the United States Government has standardized the 200-mesh sieve made from 0.0021-in. wire, having an opening of 0.0029 in., this sieve has been adopted as the base of the Tyler standard screen scale.

The 100-mesh and the 20-mesh sieves in this scale also come within specifications adopted by the Bureau of Standards, so that there are three sieves in the series which have been standardized by the Bureau.

The ratio between different sizes of the screen scale has been taken as 1.414 or the square root of 2, as recommended by Rittinger in his work on ore dressing. The niceness of this will be apparent from the following: Taking 0.0029 in. or 0.074 mm., the opening in the 200-mesh sieve as the base or starting point, the diameter of each successive opening is exactly 1.414 times the opening in the previous sieve. It also makes the area of surface of each successive opening in the scale just double that of the next finer or half that of the next coarser sieve. In other words, the diameters of the successive sizes have a constant ratio of 1.414 while the areas of the successive openings have a constant ratio of 2.

This constant ratio in the openings is shown drawn to scale, Fig. 67. To illustrate: the opening 0.093 in. in the (8-mesh) sieve is 1.414 times the opening in the preceding sieve 0.065 in. (10-mesh). The area of the opening in 0.093 in. (8-mesh) sieve is twice that of 0.065-in. (10-mesh) and just half the area of the opening in the 0.131-in. (6-mesh) sieve.

Another advantage in this selection of ratio is that by skipping every other screen, you have a ratio of diameter of 2 to 1, by skipping two sizes you have a ratio of 3 to 1 (approximately), and by skipping three sizes, you get a ratio of 4 to 1, so that in selecting a screen scale for concentrating work, for instance, you

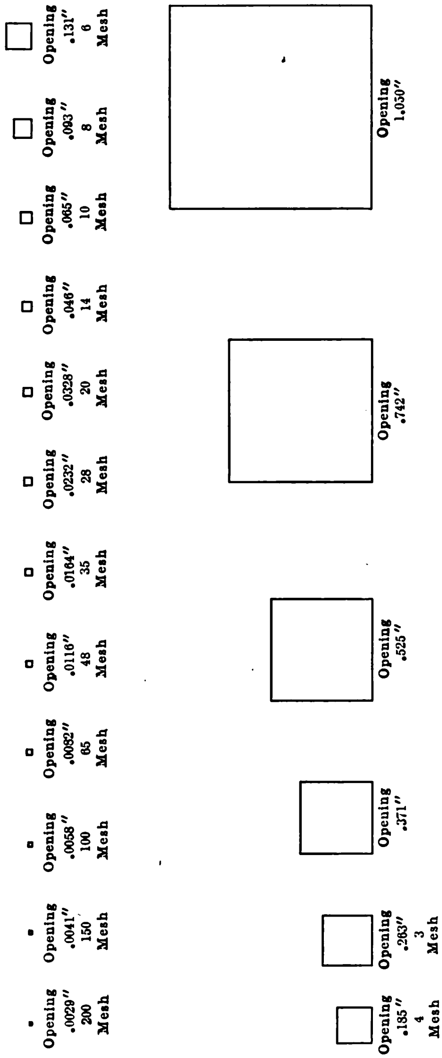

FIG. 67.—Actual size of screen openings.

can pick out from the table without any calculation a 1.414, 2, 3, or 4 to 1 ratio of opening.

TYLER STANDARD SCREENS

Mesh per linear inch	Opening, inches	Opening, millimeters	Diameter of wire, inches	Diameter of wire, millimeters	Area of openings, square inches
	4.2000	106.600	0.3750	9.52	17.64
	2.9700	75.390	0.2070	5.26	8.82
	2.1000	53.330	0.1920	4.88	4.41
	1.4900	37.730	0.1490	3.78	2.20
	1.0500	26.670	0.1490	3.78	1.10
	0.7420	18.850	0.1350	3.43	0.551
	0.5250	13.330	0.1050	2.67	0.276
	0.3710	9.423	0.0920	2.34	0.138
3	0.2630	6.680	0.0700	1.78	0.069
4	0.1850	4.699	0.0650	1.65	0.034
6	0.1310	3.327	0.0360	0.91	0.017
8	0.0930	2.362	0.0320	0.81	0.0086
10	0.0650	1.651	0.0350	0.89	0.0042
14	0.0460	1.168	0.0250	0.64	0.0021
20	0.0328	0.833	0.0172	0.44	0.00108
28	0.0232	0.589	0.0125	0.32	0.00054
35	0.0164	0.417	0.0122	0.31	0.00027
48	0.0116	0.295	0.0092	0.23	0.000135
65	0.0082	0.208	0.0072	0.18	0.0000672
100	0.0058	0.147	0.0042	0.11	0.0000336
150	0.0041	0.104	0.0026	0.07	0.0000168
200	0.0029	0.074	0.0021	0.05	0.0000084

The screen scale in the following table is based on an opening of 0.0029 in. and increases in series in the ratio of the fourth root of 2 or 1,189, the factor recommended by Prof. Richards in his work on ore dressing.

FOR CLOSER SIZING—65- TO 200-MESH

Opening in inches, ratio or 1.189	Opening in millimeters	Mesh	Diameter of wire, decimal of an inch
0.0082	0.208	65	0.0072
0.0069	0.175	80	0.0056
0.0058	0.147	100	0.0042
0.0049	0.124	115	0.0038
0.0041	0.104	150	0.0026
0.0035	0.088	170	0.0024
0.0029	0.074	200	0.0021

CHAPTER V

THE USE OF WROUGHT IRON AND ALLOY STEELS

I have indicated in a foregoing chapter the growing use of manganese and chrome steel for grinding balls in tube mills as well as their use for linings. It is therefore important that the student or operator should be conversant with the qualities imparted to steel by the addition of quantities of the various substances now used in "alloy steels."

If we consider the various stresses to which crushing machines are subjected we readily appreciate the fact that some attention should be given to the study of the various alloy steels now coming into use. We should not be led to believe that their use is a remedy for all evils for both wrought iron and carbon steel are more suitable for some classes of work than any mixtures of steel with manganese, nickel, chrome, tungsten or vanadium.

Most steels contain a small percentage of manganese by reasons of the methods of manufacture, so that this element is a basic constituent of steel and until this amounts to over 2 per cent., the product is still carbon steel. Over this amount we have manganese steel, but the real manganese steel contains over 6 per cent. manganese for between 2 and 6 per cent. the alloy is so brittle as to be worthless from an economic point of view.

Owing to the ease with which some of the elements enter the slags, it is difficult to keep to a fixed percentage of the element within narrow limits. In the manganese alloys containing 10 to 15 per cent., when made in large quantities by the openhearth process, the manufacturers require a limit of at least 1½ per cent., up and down. On account of the difficulty of decarburizing the raw products containing the alloy element, it is not commercially profitable to fix the carbon content, this being approximately regulated by the amount of carbon in the steel before the addition of the alloy element, and by that contained in the alloy iron. For example, ferromanganese containing from 60 to 80 per cent. manganese, has a carbon content of 6 to 6.50 per cent., and while the carbon content in the steel, before adding the ferromanganese, may be lowered to under 0.10 per cent. carbon, yet it must take in the carbon of the ferromanganese when this is added. The above inference pertains to steels made by the openhearth or Bessemer processes; those made by the crucible and electro-smelting processes are more even in composition, and can be duplicated within very narrow limits.

It appears that steel is a nicely balanced product, in which any increase in any one physical quality is compensated by a decrease in some other quality. For example, when we increase its tenacity by the addition of some element such as carbon, we decrease some other quality, such as its ductility or elasticity. With plain carbon steel, an increase of carbon up to a certain maximum, increases the tensile strength, likewise the hardness, but at the same time it decreases the ductility until this becomes *nil.* This may be seen by consulting the graphs.

The curious feature in respect to the alloy steels is that the increase and decrease of the physical qualities follows no definite rule, so one cannot predetermine qualities of steel containing a certain per cent. of the alloy element, from a knowledge of what the qualities were when a different proportion of this element is used. It is therefore a question for experiment to determine the physical qualities of alloy steels, with different percentages and combinations of the alloy.

CHOICE OF MATERIAL FOR THE WORK

When choosing iron or steel for any particular work, we should consider tests made on material undergoing similar stresses to those in actual practice, and on test pieces as near as possible to the actual size of the pieces actually used. For example, the usual physical tests recorded are: the tenacity or tensile strength, which is the force with which a body resists being pulled asunder, stated in pounds or tons per square inch; the limit of elasticity, or the point at which a body is permanently deformed when subjected to a stretching strain, reckoned in pounds or tons per square inch, sometimes confused with the yield point, which is near to, but not the same as the elastic limit; the ductility, which is the extent to which a body elongates before breaking, measured as a percentage of the length; the reduction in area, or the amount a test piece is reduced in size at the point of fracture;[1] the com-

[1] A coarse-grained, overheated steel will show a low percentage of reduction of area and is therefore to be avoided. A well-hammered or rolled or heat-treated piece will show a high percentage of reduction as a result of a very fine grain.

pressive strength, which is the pressure required to produce fracture; and the hardness, which is measured by the amount of indentation produced by a weighted steel ball 10 mm. diameter with a pressure of 3,000 kg. The hardness number is obtained by dividing the area of the spherical indentation into the pressure. Fig. 68 shows the hardnesses of some alloy steels. These are also hot and cold bending tests required for some material. For example, in specifying for wrought iron it is general to require the specimen to be bent hot and cold through an angle of 180° without signs of breaking.

These tests are evidently useful only for choosing a material that has to withstand static stresses, and are useless when we have to consider the effect of dynamic forces, such as forces due to

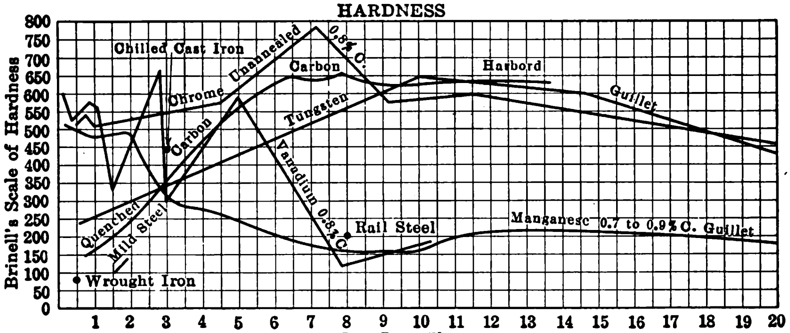

FIG. 68.—Graph of hardness of various alloy steels.

impact, rotary vibrations or combinations of many dynamic forces, acting in union or intermittently. The point is to test the proposed material under as nearly the practical conditions as possible, for while we may have a small test piece with high static properties, we have no indication of how large forgings will act under dynamic stresses, as the difficulties encountered in the heat treatment of large pieces of metal are absent when small pieces are similarly treated.

Some authoritative figures will illustrate the point of view. First is a table showing a comparison between the physical qualities of mild steel and wrought iron.[1]

Suppose that we require a material in which ductility is the most important quality necessary. We see that the mild steel

[1] ROE, *Transactions* American Institute Mining Engineers; xxxvi, p. 210.

	Tensile strength, lb. per sq. in.	Elastic limit, lb. per sq. in.	Elongation, per cent.		Reduction of area, per cent.
			12 in.	18 ft.	
Mild steel........	59,260	33,150	39	14.4	51.50
Wrought iron.....	48,560	31,550	23	15.0	28.30

is superior to the wrought iron in tensile strength and elastic limit, and that the elongation is greater on a 12-in. test piece, but on an 18-ft. test, the reverse is true. We also find the reduction of area to be greater on mild steel. Now suppose we are considering the material for a camshaft which is going to be 14 ft. long; evidently we will then favor the test of a length somewhat similar, and dynamic tests on the material proving equal, we may with good judgment select the wrought iron. The elongation and reduction of area are the effect of that quality of wrought iron, by which it elongates over a much greater length, owing to its fibrous nature, than does steel.

Considering the dynamic properties, the available figures are rather scant, but enough has been done to direct present-day practice.

Two sets of comparative figures are here given. The sets themselves must not be compared, as it is not certain whether they were taken under identical conditions.

The Society of Automobile Engineers use what they call a *quality figure*, represented by a number which expresses the relative power of metals to resist dynamic stresses, often referred to as "fatigue." "This figure has been taken arbitrarily as consisting of the following components: (1) Elastic limit, showing the range of load. (2) Reduction of area (static ductility). (3) Alternation (dynamic figure), being a measure of the fatigue resisting power. The product of these components should give a measure of both static and dynamic strength of the steel:"[1]

	Alternations	Quality figure
Carbon steel, cold-rolled, not treated...........	129,600	225
Chrome-nickel steel-annealed..................	319,200	375
Carbon steel, hot-rolled, heat-treated...........	412,200	596
Chrome-nickel, heat-treated....................	670,800	2,400

[1] *Transactions* Society Automobile Engineers, 1912.

$$\text{Qual. fig.} = \frac{\text{Elastic limit} \times \text{reduction area} \times \text{dynamic fig.}}{10^9}$$

"The dynamic figure is obtained by using a test piece approximately 0.174 by 0.500 by 6 in., clamped in one end of a vise. A tool-steel head, with a 0.5-in. slot, is placed over the other end, the distance from the striking head to the vise line being 4 in. A crank and connecting rod operated by a 0.5-in. eccentric furnishes the reciprocating motion for the head, causing a rapid vibration of the test piece. In addition to this alternating flexure, the test piece is also subjected at each reversal to an impact, due to the slot in the reciprocating head." By this method annealed chrome-nickel steel is nearly 2.5 times stronger dynamically than the carbon steel, both having the same elastic limit.

The next table shows some comparative figures obtained on different materials by static and dynamic tests:[1]

Results of Mechanical Tests of Typical Vanadium and Other Steels

	No. 1 carbon "axle" steel	No. 2 nickel "axle" steel	No. 3 vanadium "axle" steel, type A No. 1	No. 4 vanadium crank shaft, type A No. 2	No. 5 vanadium gear steel continual mesh, type A No. 3	Nature of tests
Yield point pounds per square inch	41,320	49,270	63,570	110,100	224,000	Static.
Ultimate stress tensile strength, pounds per square inch	65,840	87,360	96,080	127,800	232,750	Static.
Ratio, per cent	62	56	66	87	96	Static.
Elongation on 2 in., per cent	42	34	33	20	11	Static.
Contraction of area, per cent	61	58	61	58	39	Static.
Torsional twists	2.6	3.2	4.2	2.5	1.8	Static.
Alternating bends	10	12	18	10	6	Intermediate.
Pendulum impact foot-pounds	12.3	14.0	16.5	12.0	6.0	Dynamic.
Alternating impact number of stresses	960	800	2,700	1,850	800	Dynamic.
Falling weight on notched bar number of blows	25	35	69	76	....	Dynamic.
Rotary vibrations number of revolutions	6,200	10,000	67,500	....	....	Dynamic.

For dynamic and static test on a number of "alloy" steels, consult Kent, p. 477.

The American Society for Testing Materials has adopted a

[1] *Transactions* A. I. M. E., xxviii, p. 702.

dynamic test for stay-bolt iron, called the "vibration test," in which the bar must stand a minimum of 6,000 revolutions under the following conditions: "A threaded specimen, fixed at one end, has the other end moved in a circular path, while stressed with a tensile load of 4,000 lb. The circle described shall have a radius of $\frac{3}{32}$ in. at a point 8 in. from the fixed end of the specimen."

A drop test is at times substituted for the physical tests enumerated, this being general for steel rails. "The drop-testing

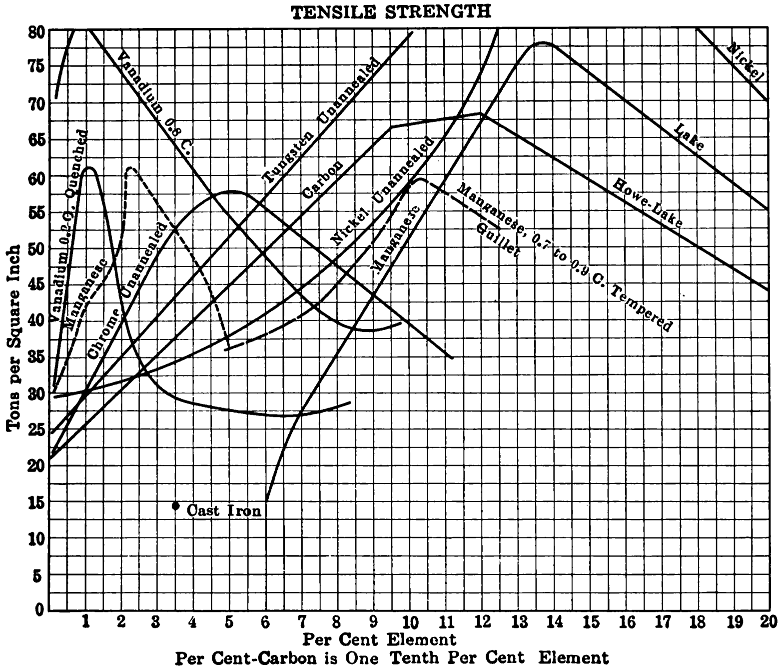

FIG. 69.—The tensile strength of various alloy steels.

machine has a tup or free falling weight of 2,000 lb., the striking face of which has a radius of not more than 5 in." For a rail weighing from 91 to 100 lb. the height of drop is 18 ft., for 81 to 90 lb. 17 ft., and the test piece 4 to 6 ft. long when placed with the face of the rail up, resting on solid supports 3 ft. apart should not break under this impact.

The elastic limit of carbon steel is about half the tensile strength, and is so considered in most specifications. In the alloy steels the elastic limit varies, increasing or diminishing generally with the tensile strength, but not always in the same proportion; in fact, the great value of these alloys is in the fact that the elastic limit at times is as great as the tensile strength. This may be seen in some of the vanadium alloys, which means that this steel breaks practically without any permanent deformation; its ductility therefore cannot be very high. For the purpose of considering the static stresses alone we will consider the tensile strength or resistance to being pulled asunder, Fig. 69, the elastic limit or the stress required to produce a permanent deformation,

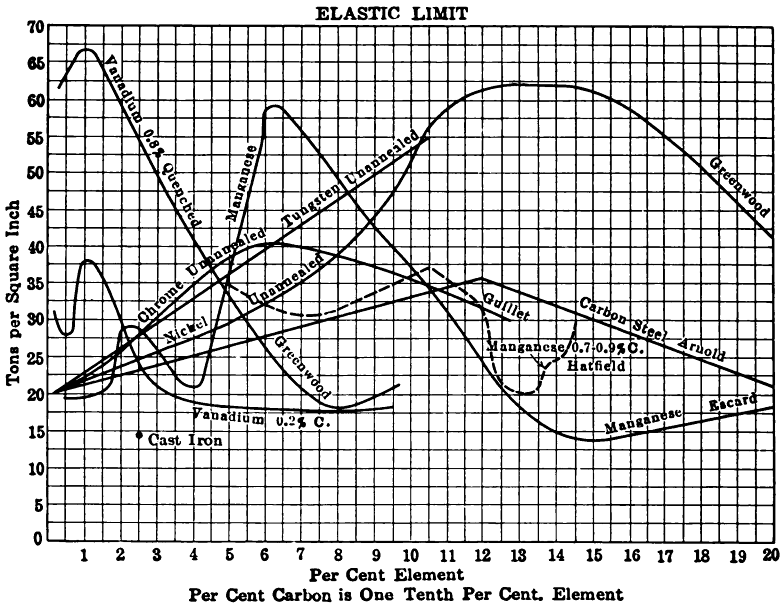

FIG. 70.—Graph of the elastic limit of various alloy steels.

Fig. 70, and the ductility represented by the percentage of elongation produced in a bar of known area and length, Fig. 71.

If we now look at the graphs some peculiar features are at once apparent; first the remarkable tensile strength of nickel, tungsten and vanadium steels, and with this their low ductility; the increased tensile strength of carbon steel up to 1.20 per cent., with its decreasing ductility, but its high ductility with low carbon content; the high tensile strength and high ductility of 12.50 to 14 per cent. manganese steel, and its comparatively low elastic limit; and the fact that manganese when added to steel has in some respects a contrary effect to that of carbon, for when quenched, instead of increasing, it decreases the hardness, and instead of decreasing the ductility as with carbon steel, it is increased. The effect of the addition of nickel to steel causes somewhat similar variations, as does manganese.

There are alloys made with varying percentages of two or more elements used for special work, and their behavior is sufficiently indicated in the graphs by combining the curves of the several elements. The dynamic qualities developed in steel by the addition of the elements mentioned appears to be due to molecular changes producing internal structures more favorable to withstand certain classes of stresses.

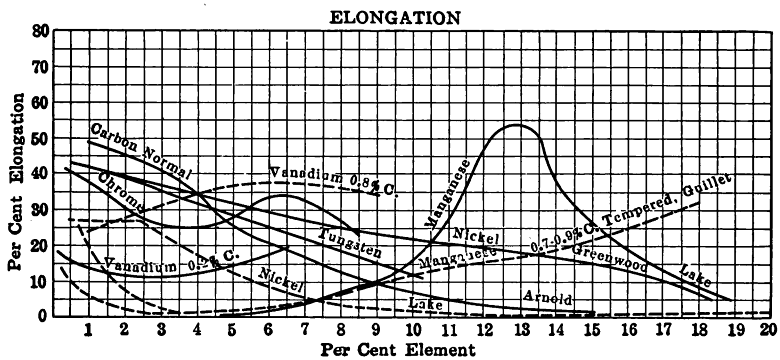

FIG. 71.—Graph of the ductility of various alloy steels.

Some of the alloy steels have been used with benefit in the automobile industry, particularly to withstand great vibratory stresses, and where an increase of strength allows less weight of material. Axles are made of nickel-chrome and vanadium steel, both said to withstand the dynamic stresses to a remarkable degree.

The effects of different heat treatments of carbon steel or alloy steel has a marked influence upon their behavior under physical stresses. The general influence is to cause an increase in the elastic limit, with a lowering of the percentage of elongation and reduction of area, but the subject is too complex to be treated here. The effect of heat treatment on carbon openhearth steel is indicated in the following table.

To compare the comparative endurance of wrought iron and

	Untreated	Heat-treated
Tensile strength, pounds per square inch	50,600	85,300
Elastic limit, pounds per square inch	28,950	27,300
Elongation, per cent	37.00	28.25

mild steel Souther[1] tested bars of each material of similar static qualities in a standard machine, where the specimen is supported in the middle and loaded at both ends, the loads being carried on ball bearings, which push snugly over the end of the specimen against a collar. The test piece was revolved with the load of the suspended weights.

TENSILE TESTS

	Elastic limit, tons per sq. in.	Tensile strength, tons per sq. in.	Reduction in area, per cent.	Elongation, per cent.
Wrought iron	18.7	26.5	35.0	24.0
Basic steel	18.0	28.2	64.5	40.0

ENDURANCE TESTS

	Loads, lb.	Revolutions	
Wrought iron	80	20,882,800	17,556,200
Basic steel	80	835,000	1,891,300

These tests show that wrought iron of nearly the same static qualities as mild steel, under 80 lb. load, has much greater power to resist fatigue.

Basquin[2] shows that the relative dynamic qualities of wrought iron and mild steel depends upon amount of the stressing loads. Under small loads, such as 7½ tons per square inch, the wrought iron shows greater endurance than any carbon steel, but under high stresses up to about 27½ tons per square inch, the best grade of spring steel shows greater resistance to fatigue.[3] This emphasizes the fact already stated that material should be tested under conditions as near as possible to those for which it is in-

[1] HENRY SOUTHER, American Society Testing Materials, 1908.

[2] Endurance Tests, American Society Testing Materials, 1911.

[3] ROSENHAIN, "Fatigue is caused by slip bands between the crystal grains extending from one crystal to another in nearly the same direction."

tended. A camshaft, with a sectional area of 28 to 38 sq. in., is under a load of from 6 to 12 tons, so may be classed with the wrought iron showing greater endurance than mild steel.

The difference between wrought iron and mild steel is not so much the difference in the per cent. of carbon content as in the structure due to the method of manufacture. Owing to the high temperature at which steels are finished, the metal is so fluid as to allow of the separation of the slag, while malleable iron ordinarily contains from 1 to 2 per cent., although this can be considerably reduced by further squeezing and forging. Malleable iron is soft, ductile and tough; mild steel strong and flexible.

"The carbon content of steel usually varies between 0.10 and 2 per cent. Metal having more than 2 per cent. is called cast iron, and is used as such. Until recently wrought iron was about the only useful iron product that contained less than 0.10 per cent. of carbon. At the present time, however, so-called ingot iron (made in the openhearth furnace, and by some considered as mild steel), consisting of about 99.9 per cent. pure iron, with only a trace of carbon (and no slag), is made commercially. With a carbon content of from 0.10 to 0.30 per cent., steel is soft, and cannot be hardened enough to prevent cutting with a file. With a carbon content of from 0.30 to 2 per cent. it can be hardened so as to cut other steels or metals, and is then called tool, half-hard or high-carbon steel, according to the carbon content."[1]

The shrinkage of manganese steel is excessive, amounting to $\frac{5}{16}$ in. per foot as against $\frac{3}{16}$ to $\frac{1}{4}$ in. in ordinary practice. The patterns therefore must be made with proper shrinkage allowance and for this reason the founders usually prefer to make their own patterns from original drawings supplied by their customers. Allowances for shrinkage are not confined simply to the dimensions of the pattern but also include the distribution of the metal in the casting. This is important since the heat treatment and cooling produce strains that must be taken care of. Manganese steel has the combined qualities of toughness and hardness, which makes it especially reliable in service where there is excessive abrasive wear. Its physical characteristics are similar to any high-grade steel, as indicated by the following figures which are an average of eight tests of manganese steel produced by the American Manganese Steel Co. for Komata tube-mill liners, made by Robt. Hunt & Co., Chicago:

[1] E. T. Lake, "Composition and Heat Treatment of Steel."

Elastic limit per square inch, 57,775 lb.; tensile strength per square inch, 92,420 lb.; per cent. elongation in 2 in., 26.91, and per cent. reduction of area, 27.02.

AVERAGE ANALYSIS OF CAST MANGANESE STEEL

	Per cent.
Carbon	1.25
Silicon	0.30
Manganese	12.50
Sulphur less than	0.02
Phosphorus, about	0.08

After the castings are poured they must be annealed, but before this is done it is sometimes necessary to give them a preliminary heat treatment to neutralize the shrinkage strains. Usually they are allowed to cool and are then carefully annealed to make them ductile and tough.

Castings which must be carefully finished are taken to the machine shop for the final operations. Manganese steel, although ductile and in a comparative degree, soft, is also exceedingly tough and dense and for this reason ordinary machining methods are unsuccessful and grinding must be resorted to.

The thickness of metal which can be successfully annealed has been gradually increased until now castings with walls 5½ in. thick are successfully treated. It is reasonable to suppose that this limit will be somewhat increased with additional experience, since 6-in. bars have already been annealed experimentally. This has an important bearing on the general problem since every increase in the depth of annealing widens the field for manganese castings.[1]

Manganese steel with 13 to 14 per cent. manganese and 1 per cent. carbon has a tensile strength of 50 to 75 tons per square inch, elongation of 50 per cent., and elastic limit from 17 tons for untempered low-carbon steel, to 41½ tons, for tempered high-carbon steel. To obtain this high tensile strength and ductility the manganese must be subjected to a heat treatment by being reheated to a high temperature and quenched. This process for large forgings produces a tendency to crack, due to interior stresses, which may account for the comparatively low elastic limit. These interior defects would prohibit its use for camshafts, and perhaps this is the reason it has not been tried for this purpose.

[1] *Bulletin* 52, American Manganese Steel Co.

At the Nipissing stamp mill chilled cast-iron mortar liners lasted 30 days while those of manganese steel of the same thickness lasted 165 days, thereby saving more than the difference in cost. At the Morro Welho mill, Brazil, fluted linings of manganese steel are used in the tube mills at an estimated cost of 94 cts. per day replacing chilled iron that cost $1.82 a day. At other mills manganese linings have not given satisfaction.

In the comparative list of iron and steel is a nickel-chrome alloy recommended by the manufacturers for camshafts. It has a tensile strength as high as 62 tons per square inch, elastic limit up to 51 tons per square inch, an elongation of 20 per cent. and reduction area of 60 per cent. This nickel-chrome steel, as well as the ordinary vanadium steel, possesses static qualities that commend its use in place of wrought iron or mild steel for camshafts, but how it will behave dynamically under the conditions requisite for sustained effort in the stamp mill can be determined only by trial.

Rockbreaker jaws must be made of material combining hardness with strength, so we see carbon, chrome and manganese steels used for the purpose. Both of the latter are superior to carbon steel, which although hard is more or less brittle. The addition of the chromium or manganese communicates to the steel the power to resist abrasive forces. Between 10 and 13 per cent. manganese steel is probably the best for rockbreaker jaws.

Line shafting is seldom subject to any but bending strains, and is always worked with a big factor of safety. Hence this part of the mill seldom fails, the mild steel used answering all our requirements. Pulleys seldom give any trouble, so there is no need of changing from cast iron or carbon steel so long as the rim, spoke and hub are sufficiently strong to withstand the strains due to the pull of the belt and the centrifugal force of the moving pulley.

Referring to the graphs of the alloy steel I infer that the tests were from small lots melted in crucibles or electric furnaces, and would therefore not represent big forgings which may, as in the case of 13½ per cent. manganese, develop such internal stresses as to be worthless for some purposes.

The data for the graphs were taken from Greenwood and Sexton, Lake, Howe, Hatfield, Guillet and Escard. While these graphs may be faulty in as far as the carbon content is not taken

Chemical Composition and Physical Tests on Iron and Steel Compounds

Character	Chemical composition: C, per cent.	Mn, per cent.	P, per cent.	S, per cent.	Si, per cent.	Ni, per cent.	Cr, per cent.	Va, per cent.	Physical tests: Tensile strength, tons per sq. in.	Elastic limit, tons per sq. in.	Elongation, per cent.	Reduction of area, per cent.	
Wrought iron	......	0.14	....	....	0.17	...	...	...	24	13½	25	45	Spec. Am. Soc. Test Mat.
Wrought iron	0.17	0.50	0.05	0.05	0.02	...	...	...	24	12½	22	20-28	General components.
Ingot iron	0.008–0.02	0.025	0.001–0.004	0.014–0.019	0.002–0.006	...	...	...	25.6	14	30	66	Cushman openhearth process.
Ingot iron, basic O. H.	0.025	0.04	0.009	0.024	Cu. 0.08	...	...	...	23½	13½	59	....	Lake.
Ingot iron	0.01	0.025	0.005	0.02	0.005	...	...	...	21	12½	25	60	Ahlbrant.
Mild steel	0.12–0.3	....	0.1	0.1	......	...	...	...	29	14½	28	35	Spec. Am. Soc. Test. Mat.
Mild steel camshaft	0.28	....	0.08	0.09	......	...	...	...	25–30	25½–15	25	....	7-in. shaft for——mill.
Manganese-steel rails	0.95–1.15	10–13	0.04	0.04	0.2–0.4	...	...	...	50	26½	20	....	A drop test is used for rails instead of these physical test here enumerated.
90- to 100-lb. steel rails	0.6–0.75	0.06–09	0.04	....	0.20	...	...	...	40–50	20–24	15	....	
100-lb. steel rails	0.62–0.75	0.7–0.1	0.04	....	0.1–0.2	...	...	...	40–48	20–24	15	....	
Carbon steel axle	0.60	0.4–0.8	0.05	0.05	......	...	...	...	42½	25	22	45	
Carbon steel cold-rolled	0.40	0.4–0.8	0.05	0.05	......	...	...	...	35	30	18	35	
Carbon steel auto frame	......	....	....	....	......	...	...	...	25	14½	37	....	Untreated.
Carbon steel auto frame	......	....	....	....	......	...	...	...	42½	28½	28¼	45	Heat-treated.
Carbon steel cast	0.40	....	0.05	0.05	......	...	...	...	42½	19¼	15	20	Hard variety.
Nickel steel	0.25	0.05	0.03	0.04	......	3.2	...	...	38–43½	24½–28½	31–34	45	Ordinary nickel steel.
Nickel steel	0.25	....	0.04	0.04	......	3.5	...	...	42½	27½	24	45	Heat-treated spec. Am. Soc. Test Mat.
Nickel-chrome steel	0.95	....	....	....	......	15.0	...	...	67	63	12½	45	High percentage nickel.
Nickel-chrome steel	......	....	....	....	......	...	...	...	50–63	44–54	12–20	60–70	Roebling's for shafts.
Nickel-chrome steel	0.25	0.45	....	0.02	0.2	3.0	0.6	...	50–62	44–51	20	60	Recommended for camshafts.
Chrome steel	0.9	....	....	....	....	...	0.5	...	85	.....	5	....	"Adamantine" shoes and dies.
Chrome cast steel	0.9	....	....	....	....	...	0.5	...	85	.....	5	....	"Adamantine" tappets, cams and bosses.
Vanadium steel	......	....	....	....	....	...	...	0.2	42½	25¼	28	67.5	United Steel Co. Va steel.
Vanadium steel	......	....	....	....	....	...	...	0.2	104½	102½	11	44.0	Heat-treated extreme example.
Vanadium-chrome steel	......	....	....	....	....	...	...	0.2	63½	58½	16	61.5	United Steel Co.
Vanadium-chrome steel	0.24	0.44	0.03	0.017	0.2	...	0.75	0.2	46¾	37½	27	61.5	Shaft of a machine. Resisting great torsional and tensile strains.
Vanadium-chrome steel	0.35	0.49	0.011	0.011	0.125	...	0.98	0.26	64	51¼	18	50.5	Shaft of a machine.

into account, in all cases except in the carbon steel, it is good so far as our knowledge goes.

The line of tensile strength for nickel goes to nearly 100 tons per square inch. The alloy between 8 and 15 per cent. is very brittle, and from 20 per cent. up the toughness increases, compensated for by the lowering of the elastic limit and tensile strength. The usual percentage used is from 3 to 3½ per cent.

APPENDIX

USEFUL DATA

1 sec.-ft. of water equals 40 California miner's in.

1 sec.-ft. of water equals 38.4 Colorado miner's in.

1 sec. ft. equals 7.48 U. S. gal. per second; equals 448.8 gal. per minute; equals 646,272 gal. per day.

1. U. S. gal. equals 231 cu. in. or 8.33 lb. water at 4°C.

1 cu. ft. equals 7.48 U. S. gal. or 62.4 lb. water at 4°C.

Atmospheric pressure at sea level equals 14.7 lb. per square inch.

Height of mercury column at sea level 29.9 in.

Height of water column at sea level equals 33.9 ft.

1 B.t.u. is equivalent to the expenditure of 778 ft.-lb. of work.

Pounds per square inch equals 0.434 times head of water in feet.

To convert Centigrade to Fahrenheit:

Temp. C. = $\frac{5}{9}$ (temp. F. − 32).

Temp. F. = $\frac{9}{5}$ (temp. C. + 32).

1 hp.
- 746 watts.
- 0.746 kw.
- 33,000 ft.-lb. per minute.
- 550 ft.-lb. per second.
- 42.44 B.t.u. heat units per minute.
- 23.5 lb. cal. units per minute (pounds, degrees C.).
- 0.707 heat units per second.
- 0.175 lb. carbon oxidized per hour.
- 150 sq. ft. heating surface in a standard tubular boiler.
- 2.64 lb. water evaporated per hour from water at 212°F.

$\frac{\text{volts} \times \text{amperes}}{746}$ for continuous currents.

Specific heat
- Stones generally 0.2 to 0.22.
- Quartz 0.188.
- Marble 0.21.

CONVERSION OF THE METRIC SYSTEM

1 m. = 39.37 in. or 3.281 ft.

1 km. = 0.6214 miles

1 hectare = 2.471 acres.

1 sq. m. = 10.76 sq. ft.

1 c.c = 0.061 cu. in.

1 liter = 61.02 cu. in. or 0.2642 gal.

1 cm. per second = 0.03281 ft. per second.

1 gram = 15.43 grains = 0.03527 oz.
1 kg. = 15,432 grains or 2.205 lb.
1 kg. per square meter = 0.2048 lb. per square foot.
1 kgm. = 7.233 ft.-lb.
1 kw.-hr. = 1.342 hp.-hr.
1 metric hp. = 0.9863 hp.

CYANIDE CHEMISTRY

Determination of Gold and Silver. Eight Methods

1. Boil with aluminum and sulphuric acid. Scorify precipitate with granulated lead.
2. Acidify and precipitate with aluminum sulphide and scorify.
3. Evaporate to dryness in lead boat made of sheet lead about 8 in. long and 3 in. wide. Scorify and cupel.
4. Evaporate to dryness with litharge in porcelain dish, scorify and cupel.
5. Agitate 1,000 c.c. with 2 grams powdered copper sulphate. Add 15 c.c. conc. HCl and agitate. Filter, dry, scorify and cupel.
6. Add mercuric chloride (20.846 grams per liter of water) until precipitation ceases. Filter, wash and dry. Run in crucible and cupel.
7. Take 10 A. T. and add 20 drops bichromate solution. Add nitrate of silver until deep red color of chromate of silver appears. Add zinc dust and mix. Add H_2SO_4 to dissolve zinc. Filter, wash, dry, wrap in lead foil, scorify and cupel.
8. Take 10 A. T. solution and when nearly boiling add 70 to 75 c.c. conc. lead acetate and from 1 to 1½ grams zinc dust. When boiling has started add 20 c.c. conc. hydrochloric acid and boil for 10 min. The lead sponge is now at the top. Pour off the liquid and wash the sponge and roll into a ball. Put in a cupel in the muffle. This is by far the best and quickest method. Care must be taken not to break up the sponge or it will have to be filtered and run in a crucible with lead flux and silica.

Determination of Cyanide

The standard solution of silver nitrate is made by dissolving 6.5232 grams pure $AgNO_3$ in 1,000 c.c. distilled water. If we take 10 c.c. mill solution every cubic centimeter $AgNO_3$ solution will represent 1 lb. free cyanide per ton of solution. To test pure solutions of cyanide of potassium, titrate 10 c.c. cyanide solution with the standard solution of nitrate of silver using a few drops of 5 per cent. solution of potassium iodide for an indicator.

Total Cyanide. Two Methods

1. Boil 10 c.c. solution with mercuric oxide in excess. Add a 10 per cent. solution of sulphide of soda and shake. When precipitate has stopped forming, add freshly precipitate carbonate of lead in excess, filter, cool and estimate cyanide present by standard silver nitrate solution. This gives cyanide present whether excising as single or double cyanides.

2. To 50 c.c. sample add 5 to 10 c.c. of 4 per cent. solution of sodium hydrate containing 1 per cent. potassium iodide; titrate with standard silver nitrate to yellow tint.

Protective Alkali.—The protective alkali in a cyanide solution is the amount of alkali or alkaline carbonates present to offset the effect of acid radicals in the ore or water used in the mill. To 50 c.c. sample, add slightly more (about 0.2 c.c.) than the amount of silver nitrate found to be required for the total cyanide, together with a slight excess of potassium ferrocyanide, then 8 to 10 drops of phenolphthalein are added to the turbid liquid and the mixture titrated (without filtering) with standard acid (N/10 sulphuric acid) until the pink just disappears.

The usual mill procedure is to take two samples of 50 or 10 c.c. according to individual preference; one sample is titrated for free cyanide and alkali and the other for total cyanide using a 2 or 4 per cent. solution of sodium hydrate with 1 per cent. potassium iodide.

A sulphuric acid solution that will indicate 1 lb. of lime to the ton of solution for every c.c. using 10 c.c. of mill solution may be made by adding to a liter of water the number of c.c. of acid found by dividing 8.78 by the specific gravity of the acid.

MELTING ZINC PRECIPITATE

Short zinc may be melted with the following flux:

Short zinc	100 parts
Borax	40
Soda	20
Sand	10
Lime	5

Gold precipitate may be fluxed as follows:

Precipitate	100 parts
Borax	15
Sodium carbonate	8
Sand	4
Iron in excess.	

ASSAY OF ZINC PRECIPITATE

Crucible Method:

$\frac{1}{10}$ A.T. precipitate
70 grams litharge
5 grams sodium carbonate
1 gram flour
5 grams silica
2 grams borax glass

Scorifier Method:

$\frac{1}{10}$ A.T. precipitate mixed with
$\frac{3}{10}$ A.T. litharge covered with
40 grams lead

Indicators.

Methyl orange gives a pale yellow tint with alkalies and pink with acids. Phenolphthalein gives deep pink or rose with alkalies and colorless with acids.

The former is most sensitive to alkalies while the latter is more sensitive to acids.

SLIME DENSITY RELATIONS[1]

The table is based on the percentage of solid in the slime, opposite which is given the ratio of solid to liquid. The numbers heading the double columns following, are the specific gravities of the dry solid (that of water being taken as unity). The columns headed S. G. show the specific gravities of the slime, that of water being taken as 1.000; that is, the figures show directly the weight of a liter of slime in grams. The columns headed Vol. show the number of cubic feet of the slime in 1 ton of 2,000 lb.

The specific gravities of solids chosen will probably cover the range of slimes ordinarily met with and the intervals are sufficiently small to admit of interpolation without appreciable error. The last column (4.50 is a hypothetical concentrate and is the specific gravity of a mixture of 80 per cent. pyrite and 20 per cent. quartz. The average specific gravity of working cyanide solutions is so small as to be negligible.

The table is convenient for ascertaining the amount of solid and of solution in slime pulps from the number of cubic feet, determined by rod or float, in the tank; and specific gravity of the slime, determined by taking the weight of a liter or by a specific gravity indicator in the tank. It is useful in calculations for ascertaining the amount of solution to be abstracted or added in thickening and diluting, for correcting the strength of the solutions, for checking tonnage and for other purposes.

Assume that in a plant in which the specific gravity of the solid is 2.7, a tank is shown, by the depth of pulp in it, to contain 3,530 cu. ft. of pulp, a liter of which weighs 1,223 gm. From the table it is found that the specific gravity 1,223 corresponds to 26.16 cu. ft. per ton and to 29 per cent. solid. The weight of pulp, therefore, is 3,530 ÷ 26.16 = 135 tons and the weight of solids 135 × 0.29 = 39.15 tons. The weight of solution is, by difference, 95.85 tons. If the solution titrates 1.05 lb. cyanide per ton and it is desired to bring the strength up to 2.5 lb. per ton, we have 2.5 − 1.05 = 1.45 lb. cyanide to be added per ton. Therefore, 95.85 × 1.45 = 139 lb. cyanide to be added to the tank.

The table is useful in determining the sizes of tanks necessary for any given capacities. Thus, if it is desired to agitate 50 tons of dry slime (specific gravity of solid 2.6, with 3 parts solution, the table shows this to contain 25 per cent. solids and to have a volume of 27.08 cu. ft. per ton; therefore, 50 ÷ 0.25 = 200 tons of slime × 27.08 = 5,416 cu. ft., the required effective working capacity of the tank, to which an amount must be addeb to secure the desired height of curb above the charge.

For more accurate tables based on specific gravity of water being greater than unity see *Engineering and Mining Journal*, Dec. 19, 1914, pages 1079–1094.

[1] *Metallurgical and Chemical Engineering*, June, 1912.

Slime Density Relations.[1]

Per cent. solids	Ratio of solids to solution	Specific gravity of pulp and volume of 1 ton in cubic feet, for slimes containing solids of different specific gravities									
		2.50		2.60		2.70		2.80		2.90	
		S.G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.
5	1:19.000	1,031	31.03	1,032	31.01	1,032	31.01	1,033	30.97	1,034	30.95
6	1:15.667	1,037	30.85	1,036	30.82	1,039	30.79	1,040	30.76	1,041	30.74
7	1:13.286	1,044	30.66	1,045	30.62	1,046	30.59	1,047	30.56	1,048	30.53
8	1:11.500	1,050	30.46	1,052	30.43	1,053	30.39	1,055	30.36	1,055	30.32
9	1:10.111	1,057	30.27	1,059	30.23	1,060	30.19	1,061	30.15	1,063	30.11
10	1: 9.000	1,064	30.08	1,065	30.03	1,067	29.99	1,068	29.95	1,070	29.90
11	1: 8.091	1,071	29.88	1,073	29.83	1,074	29.79	1,076	29.74	1,078	29.69
12	1: 7.333	1,078	29.70	1,080	29.64	1,082	29.59	1,083	29.53	1,085	29.48
13	1: 6.692	1,085	29.50	1,087	29.44	1,089	29.39	1,091	29.33	1,093	29.27
14	1: 6.144	1,092	29.31	1,094	29.24	1,097	29.19	1,099	29.12	1,101	29.06
15	1: 5.667	1,099	29.18	1,102	29.05	1,104	28.99	1,107	28.91	1,109	28.85
16	1: 5.250	1,106	28.93	1,109	28.85	1,112	28.78	1,115	28.71	1,117	28.65
17	1: 4.882	1,114	28.74	1,117	28.65	1,119	28.58	1,123	28.50	1,125	28.44
18	1: 4.556	1,121	28.54	1,125	28.45	1,128	28.38	1,131	28.30	1,134	28.23
19	1: 4.263	1,129	28.35	1,133	28.26	1,136	28.18	1,139	28.09	1,142	28.02
20	1: 4.000	1,136	28.17	1,140	28.06	1,144	27.98	1,147	27.89	1,151	27.81
21	1: 3.762	1,144	27.97	1,148	27.87	1,152	27.77	1,156	27.68	1,159	27.60
22	1: 3.545	1,152	27.78	1,157	27.67	1,161	27.57	1,165	27.47	1,168	27.39
23	1: 3.348	1,160	27.58	1,165	27.47	1,169	27.37	1,174	27.27	1,177	27.18
24	1: 3.167	1,168	27.39	1,173	27.27	1,178	27.17	1,182	27.06	1,186	26.97
25	1: 3.000	1,176	27.21	1,182	27.08	1,187	26.97	1,191	26.85	1,195	26.76
26	1: 2.846	1,185	27.01	1,190	26.88	1,195	26.77	1,201	26.65	1,205	26.55
27	1: 2.704	1,193	26.82	1,199	26.68	1,205	26.56	1,210	26.44	1,215	26.34
28	1: 2.571	1,202	26.62	1,209	26.49	1,214	26.36	1,220	26.24	1,224	26.13
29	1: 2.448	1,211	26.43	1,217	26.29	1,223	26.16	1,229	26.03	1,234	25.92
30	1: 2.333	1,220	26.24	1,226	26.10	1,233	25.95	1,239	25.83	1,244	25.71
31	1: 2.226	1,229	26.05	1,236	25.90	1,242	25.75	1,249	25.63	1,255	25.50
32	1: 2.125	1,238	25.86	1,245	25.70	1,252	25.55	1,259	25.42	1,265	25.29
33	1: 2.030	1,247	25.66	1,255	25.50	1,262	25.35	1,269	25.21	1,276	25.08
34	1: 1.940	1,256	25.47	1,264	25.31	1,272	25.15	1,279	25.01	1,287	24.87
35	1: 1.857	1,266	25.28	1,274	25.12	1,283	24.95	1,290	24.80	1,298	24.66
36	1: 1.778	1,276	25.09	1,284	24.91	1,293	24.75	1,301	24.60	1,309	24.45
37	1: 1.703	1,285	24.90	1,295	24.71	1,304	24.55	1,312	24.39	1,320	24.24
38	1: 1.632	1,295	24.70	1,305	24.52	1,314	24.35	1,323	24.19	1,332	24.03
39	1: 1.564	1,305	24.51	1,316	24.32	1,326	24.14	1,335	23.98	1,343	23.82
40	1: 1.500	1,316	24.32	1,326	24.13	1,336	23.95	1,346	23.77	1,355	23.61
41	1: 1.439	1,326	24.13	1,337	23.93	1,348	23.74	1,357	23.57	1,367	23.40
42	1: 1.381	1,337	23.94	1,348	23.73	1,359	23.55	1,370	23.36	1,380	23.19
43	1: 1.326	1,348	23.74	1,359	23.53	1,371	23.34	1,382	23.16	1,392	22.99
44	1: 1.273	1,359	23.55	1,372	23.33	1,383	23.15	1,395	22.95	1,405	22.78
45	1: 1.222	1,370	23.36	1,383	23.14	1,395	22.94	1,407	22.74	1,418	22.57
46	1: 1.174	1,381	23.17	1,395	22.94	1,408	22.73	1,420	22.54	1,432	22.36
47	1: 1.128	1,393	22.98	1,407	22.75	1,420	22.54	1,433	22.33	1,445	22.15
48	1: 1.083	1,404	22.78	1,419	22.55	1,433	22.33	1,446	22.12	1,458	21.94
49	1: 1.041	1,416	22.59	1,431	22.35	1,446	22.13	1,460	21.92	1,473	21.73
50	1: 1.000	1,429	22.39	1,444	22.15	1,460	21.92	1,473	21.71	1,487	21.52
51	1: 0.961	1,441	22.21	1,458	21.96	1,473	21.72	1,488	21.51	1,502	21.31
52	1: 0.923	1,453	22.02	1,471	21.76	1,487	21.52	1,502	21.30	1,517	21.10
53	1: 0.887	1,466	21.82	1,484	21.56	1,501	21.32	1,516	21.10	1,532	20.89
54	1: 0.852	1,479	21.63	1,498	21.36	1,515	21.12	1,532	20.89	1,548	20.68
55	1: 0.809	1,493	21.44	1,512	21.17	1,530	20.92	1,547	20.69	1,564	20.47
56	1: 0.786	1,506	21.25	1,526	20.97	1,545	20.72	1,563	20.48	1,580	20.26
57	1: 0.754	1,520	21.06	1,540	20.77	1,560	20.51	1,579	20.27	1,596	20.05
58	1: 0.724	1,534	20.86	1,555	20.58	1,574	20.31	1,595	20.07	1,613	19.84
59	1: 0.695	1,548	20.67	1,572	20.38	1,591	20.11	1,611	19.86	1,629	19.63
60	1: 0.667	1,563	20.48	1,585	20.18	1,607	19.91	1,628	19.66	1,645	19.42
61	1: 0.639	1,577	20.29	1,601	19.98	1,623	19.71	1,645	19.45	1,664	19.21
62	1: 0.613	1,592	20.10	1,617	19.79	1,641	19.51	1,662	19.25	1,683	19.00
63	1: 0.587	1,608	19.90	1,633	19.59	1,657	19.30	1,681	19.04	1,703	18.79
64	1: 0.563	1,623	19.71	1,650	19.40	1,675	19.10	1,698	18.94	1,723	18.58
65	1: 0.538	1,639	19.52	1,667	19.20	1,692	18.90	1,718	18.73	1,742	18.37
66	1: 0.515	1,656	19.32	1,684	19.00	1,711	18.70	1,738	18.53	1,762	18.16
67	1: 0.493	1,672	19.14	1,701	18.80	1,730	18.50	1,757	18.32	1,783	17.95
68	1: 0.471	1,689	18.94	1,719	18.61	1,749	18.30	1,776	18.11	1,803	17.74
69	1: 0.449	1,706	18.75	1,738	18.41	1,768	18.10	1,797	17.81	1,825	17.53
70	1: 0.429	1,724	18.56	1,757	18.21	1,786	17.90	1,818	17.60	1,847	17.32

[1] Formula, $P = \frac{100S(a - d)}{a\,(S - d)}$

Where P = percentage of dry slime n wet pulp.
S = specific gravity of dry slime.
a = specific gravity of wet pulp.
d = specific gravity of solution.

Slime Density Relations

Per cent. solids	Ratio of solids to solution	Specific gravity of pulp and volume of 1 ton in cubic feet, for slimes containing solids of different specific gravities									
		3.00		3.10		3.20		3.30		[1]4.50	
		S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.	S. G.	Vol.
5	1:19.000	1,035	30.93	1,035	30.92	1,036	30.90	1,036	30.89	1,040	30.76
6	1:15.667	1,042	30.72	1,042	30.70	1,043	30.68	1,043	30.66	1,049	30.51
7	1:13.286	1,049	30.51	1,049	30.48	1,050	30.46	1,051	30.43	1,058	30.26
8	1:11.500	1,056	30.30	1,057	30.27	1,058	30.24	1,059	30.21	1,067	30.01
9	1:10.111	1,064	30.09	1,065	30.05	1,066	30.02	1,067	29.99	1,075	29.76
10	1: 9.000	1,071	29.87	1,072	29.83	1,074	29.80	1,075	29.77	1,084	29.51
11	1: 8.091	1,078	29.65	1,080	29.61	1,082	29.58	1,083	29.54	1,093	29.26
12	1: 7.333	1,087	29.44	1,088	29.40	1,090	29.36	1,091	29.32	1,102	29.01
13	1: 6.692	1,095	29.23	1,096	29.18	1,098	29.14	1,099	29.10	1,112	28.76
14	1: 6.144	1,103	29.01	1,105	28.96	1,106	28.92	1,108	28.88	1,122	28.52
15	1: 5.667	1,111	28.80	1,113	28.74	1,115	28.70	1,117	28.66	1,132	28.27
16	1: 5.250	1,119	28.59	1,122	28.53	1,124	28.48	1,125	28.43	1,142	28.02
17	1: 4.882	1,128	28.37	1,130	28.31	1,132	28.26	1,134	28.21	1,152	27.77
18	1: 4.556	1,136	28.16	1,139	28.10	1,141	28.04	1,143	27.99	1,163	27.52
19	1: 4.263	1,145	27.95	1,148	27.88	1,150	27.82	1,153	27.76	1,173	27.27
20	1: 4.000	1,154	27.73	1,157	27.66	1,159	27.60	1,162	27.54	1,184	27.02
21	1: 3.762	1,163	27.52	1,166	27.44	1,169	27.38	1,171	27.32	1,194	26.77
22	1: 3.545	1,172	27.31	1,175	27.23	1,178	27.16	1,181	27.09	1,206	26.52
23	1: 3.348	1,181	27.09	1,184	27.01	1,188	26.94	1,191	26.87	1,218	26.28
24	1: 3.167	1,190	26.88	1,194	26.79	1,198	26.72	1,201	26.65	1,230	26.03
25	1: 3.000	1,200	26.67	1,204	26.58	1,208	26.50	1,211	26.42	1,241	25.78
26	1: 2.846	1,210	26.45	1,214	26.37	1,218	26.28	1,222	26.20	1,253	25.53
27	1: 2.704	1,220	26.24	1,224	26.15	1,228	26.06	1,232	25.98	1,266	25.28
28	1: 2.571	1,230	26.03	1,234	25.93	1,239	25.84	1,242	25.75	1,278	25.03
29	1: 2.448	1,240	25.81	1,244	25.71	1,249	25.62	1,253	25.53	1,291	24.78
30	1: 2.333	1,250	25.60	1,255	25.50	1,260	25.40	1,264	25.31	1,304	24.53
31	1: 2.226	1,261	25.39	1,266	25.28	1,271	25.18	1,275	25.08	1,317	24.28
32	1: 2.125	1,271	25.17	1,277	25.06	1,282	24.96	1,287	24.86	1,331	24.04
33	1: 2.030	1,282	24.96	1,288	24.85	1,293	24.74	1,299	24.64	1,345	32.79
34	1: 1.940	1,293	24.75	1,299	24.63	1,305	24.52	1,311	24.41	1,359	23.54
35	1: 1.857	1,304	24.53	1,310	24.41	1,317	24.30	1,323	24.19	1,374	23.29
36	1: 1.778	1,316	24.32	1,322	24.19	1,329	24.08	1,335	23.97	1,389	23.04
37	1: 1.703	1,328	24.11	1,334	23.98	1,341	23.86	1,347	23.75	1,404	22.79
38	1: 1.632	1,340	23.89	1,346	23.76	1,353	23.64	1,360	23.52	1,420	22.54
39	1: 1.564	1,351	23.68	1,358	23.55	1,366	23.42	1,373	23.30	1,435	22.29
40	1: 1.500	1,363	23.47	1,371	23.33	1,379	23.20	1,387	23.08	1,451	22.04
41	1: 1.439	1,376	23.26	1,384	23.11	1,393	22.98	1,400	22.85	1,468	21.79
42	1: 1.381	1,389	23.04	1,397	22.89	1,406	22.76	1,414	22.63	1,485	21.55
43	1: 1.326	1,402	22.83	1,411	22.68	1,419	22.54	1,428	22.41	1,502	21.30
44	1: 1.273	1,415	22.61	1,425	22.46	1,433	22.32	1,442	22.18	1,519	21.05
45	1: 1.222	1,429	22.40	1,438	22.24	1,447	22.10	1,456	21.96	1,538	20.80
46	1: 1.174	1,443	22.19	1,452	22.02	1,462	21.88	1,471	21.74	1,557	20.55
47	1: 1.128	1,457	21.97	1,467	21.81	1,477	21.66	1,487	21.51	1,576	20.30
48	1: 1.083	1,471	21.76	1,483	21.60	1,493	21.44	1,503	21.29	1,595	20.05
49	1: 1.041	1,485	21.55	1,497	21.38	1,508	21.22	1,519	21.07	1,615	19.81
50	1: 1.000	1,500	21.33	1,512	21.16	1,524	21.00	1,535	20.85	1,637	19.56
51	1: 0.961	1,515	21.12	1,528	20.94	1,540	20.78	1,551	20.62	1,658	19.31
52	1: 0.923	1,531	20.91	1,544	20.73	1,556	20.56	1,568	20.40	1,679	19.06
53	1: 0.887	1,547	20.69	1,560	20.51	1,573	20.34	1,585	20.18	1,700	18.81
54	1: 0.852	1,563	20.48	1,577	20.29	1,590	20.12	1,603	19.96	1,724	18.56
55	1: 0.809	1,579	20.27	1,594	20.08	1,608	19.90	1,621	19.73	1,748	18.31
56	1: 0.786	1,596	20.05	1,611	19.87	1,626	19.68	1,640	19.51	1,772	18.06
57	1: 0.754	1,613	19.84	1,628	19.65	1,645	19.46	1,659	19.29	1,796	17.81
58	1: 0.724	1,631	19.63	1,646	19.43	1,663	19.24	1,678	19.06	1,822	17.56
59	1: 0.695	1,649	19.41	1,665	19.21	1,682	19.02	1,697	18.84	1,848	17.32
60	1: 0.667	1,667	19.20	1,684	19.00	1,702	18.80	1,718	18.62	1,875	17.07
61	1: 0.639	1,686	18.99	1,704	18.78	1,722	18.58	1,739	18.39	1,903	16.82
62	1: 0.613	1,705	18.77	1,724	18.56	1,742	18.36	1,761	18.17	1,932	16.57
63	1: 0.587	1,724	18.56	1,745	18.34	1,764	18.14	1,783	17.95	1,961	16.32
64	1: 0.563	1,745	18.35	1,765	18.12	1,786	17.92	1,805	17.72	1,992	16.07
65	1: 0.538	1,765	18.13	1,786	17.91	1,808	17.70	1,828	17.50	2,023	15.82
66	1: 0.515	1,786	17.92	1,808	17.69	1,830	17.48	1,852	17.28	2,054	15.57
67	1: 0.493	1,808	17.71	1,831	17.47	1,853	17.26	1,876	17.06	2,088	15.32
68	1: 0.471	1,830	17.49	1,854	17.26	1,877	17.04	1,901	16.83	2,123	15.08
69	1: 0.449	1,852	17.28	1,878	17.04	1,902	16.82	1,927	16.61	2,159	14.88
70	1: 0.429	1,875	17.07	1,902	16.83	1,926	16.60	1,953	16.39	2,195	14.53

[1] 80 per cent. pyrite and 20 per cent. quartz.

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