1889.
NEW ZEALAND.

MINING MACHINERY AND TREATMENT OF ORES IN AUSTRALIAN COLONIES.

REPORTS ON MINING MACHINERY EXHIBITED AT THE MELBOURNE EXHIBITION. ON MINING, AND PLANTS FOR THE REDUCTION AND TREATMENT OF ORES, IN THE AUSTRALASIAN COLONIES. ON PROCESSES ADOPTED IN AMERICA FOR TREATMENT OF AURIFEROUS AND ARGENTIFEROUS ORES.

Presented to both Houses of the General Assembly by Command of His Excellency.

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Presented to both Houses of the General Assembly by Command of His Excellency.

Mr. H. A. Gordon, F.G.S., Inspecting Engineer, to the Under-Secretary of Mines.

Sir,— Mines Department, Wellington, 7th June, 1889.

In accordance with the instructions of the Hon. the Minister of Mines to visit the Melbourne Exhibition and the Australian Colonies, and report on the different classes of modern machinery adapted for working the mines in New Zealand and for reducing and successfully treating the various ores, and also to collect all information that was likely to be beneficial to the mining community in this colony, I have the honour to report on the same as follows:—

The mining machinery at the Melbourne Exhibition was not so complete as one would expect to find at an International Exhibition. There were very few new improvements in the machinery exhibited, although some of the quartz-reduction mills were new to the Australasian Colonies, such as the Huntingdon, and the Bryant Roller-mill, both of which are very compact, and do good work. There were several other new reduction-mills, one of which had stamps fixed in a semicircular position, and lifted by a revolving circular table, having steel bars on the upper side at its periphery set on an angle, so that when the stamp had to be lifted the bar went under the disc, and as it revolved the stamp was lifted by the wedge-form of the bar until it got to the highest point of elevation the bar stood at, and then it dropped, to be lifted again by the next bar. The bars on this circular revolving disc or table resembled in shape the teeth of a ratchet-wheel, having a tooth about 9in. deep and 1ft. 6in. long. No tests were made with this mill, and there was nothing in it which could be recommended to supersede the ordinary stamp-battery.

In dealing with the different subjects they are placed under the following heads: Crushing and pulverising machinery, ore-concentrators, Newbery-Vautin process of chlorination, Hedley's patent electric ore-converter, rock-drills, water-augers, steam-boilers, pumping machinery, mineral railway, new explosives, Mount Bischoff tin-mine and machinery, Broken Hill mine and works, zinc desilverisation process used at Dry Creek Smelting-works, Adelaide, and a description of the processes used for extracting metals from the ore at the Boston and Colorado Works, in America. Also an Appendix containing a report from Professor Roberts-Austen, F.R.S., on the processes of treatment of ores at the Saxon Government works, Freiberg.

CRUSHING AND PULVERISING MACHINERY.

The question of an economical machine for the reduction of ores is one deserving the attention of every mine-owner in the Australasian Colonies, and more especially in the Colony of New Zealand, where there are a number of lodes, containing low-grade ore, which cannot at the present time be profitably worked on account of the expense of reduction and subsequent treatment which is required to extract something like a fair percentage of the precious metals they contain. The recognised system hitherto in the Australasian Colonies for pounding up and pulverising the ore has been by stamping-batteries, and it is still a moot question whether a stamping-battery, properly constructed, is not the most economical class of machinery to use under certain conditions. In the gold- and silver-fields of America they are still largely employed, although the roller-mill principle is beginning to supersede them for certain classes of ore; but in America the stamping-batteries are differently constructed from those employed in these colonies, and, judging from the mode of their construction, they are far more effective than the stamps employed here.

Before entering on the subject of roller-mills it may be of interest to mention a few of the important points in the construction of stamping-mills.

All those who exhibited American mining machinery at the Melbourne Exhibition state emphatically that no quartz-mill owner or battery-manager in America would ever think of putting large pieces of quartz into a stamp-mill. All the material which will not pass over a grizzly is put through rock-breakers, and reduced to a maximum size of about 2in. in diameter.

The ore as it leaves the mine is conveyed in trucks and dumped on to a grizzly, and what will not pass through runs down over the grizzly into the stone-breaker. The ore-bins or paddocks are generally sufficiently large to admit of holding all the quartz or ore that can be raised from the mine in one shift. The paddocks also should be at such an elevation above the reduction machinery as will admit of the material being all led down through the different processes by gravitation.

Rock-breakers are the most important adjuncts in connection with crushing ores, and are the cheapest form of crusher which has yet been brought into general use. Indeed, they do a large portion of the work, and bring down the ore to a somewhat uniform size, that admits of far more work being done by stamps. When large pieces of hard-quartz ore are fed into the stamps the first blow breaks certain large portions, sending them at a high velocity against the screens, which tends to burst them, or, at least, to knock holes in them; whereas if the material is well broken before going into the stamping-mortar this does not take place to the same extent. Also, when the material is of a uniform size it can be fed into the stamping-mortar with much more regularity; and on this depends in a great measure the quantity of ore that can be pounded up sufficiently fine by the stamps to pass through the grating or screen.

All the ore coming from the bins or paddock should be run into an ore-feeder, which feeds the stamps automatically, and these ore-feeders can be so nicely adjusted that there is always a uniform quantity of loose material in the stamp-mortar; the great desideratum being to get the stamp to do the maximum amount of duty, provided that a maximum percentage of the value of the ore shall be the result.

The regularity of feeding is deserving of study and attention. It depends to a certain extent on the class of ore that is being crushed: if it is low-grade, containing sulphurets, it may be found more economical not to have too fine a screen, and to keep the stamp as close down to the die as possible. The less the quantity of material—that is, within certain limits—which lies between the die and stamp, the greater the quantity will be put through. But this is a matter which a battery-manager can best determine, so as to get the best results in working the different classes of ore.

In mentioning the descriptions of stamping-mortars, Joshua Hendy, of the City Ironworks, San Francisco, states, "They are made of different forms, to suit the peculiarities of the gold- and silver-bearing ores, as well as their value. They are made for low, high, and overflow discharge, sometimes wide and flaring, sometimes narrow and contracted at the bottom up to the discharge-opening. The conditions of use must determine the form to be adopted. For gold-mills the discharge lip or apron should be of iron, cast as part of the mortar, as this prevents any leakage escaping from around the screen-frame. This lip or apron should be gradually drawn or tapered to the point of discharge, which should be open and not less than 16in. wide—the width of the discharge-sluices—and an easy descending grade be given, to permit of an even and shallow flow of the pulp over the copper or silver plates with which the bottoms of the tables are covered. The frames of the sluices should never be constructed in contact with the mortars or battery-frames, but independent, and resting upon independent bearings, so as to prevent jar and thereby leakage. . . . . The screen-frames should be set with an outward inclination at the tops, and for milling gold-ores of an average value the distance from the bottom of the screens as set in the frames to the top of the dies in the mortars should be at least 7in."

The whole pith of this is that, the screens being so high above the top of the dies, there is far less wear and tear on the screens, and, the discharge being high, fine crushing can be accomplished with a much coarser grating than would be required were it set down to within 2in. of the top of the dies, as is done in many cases in the colonies. There is no gainsaying the fact that we can learn a great deal from the Americans in the matter of reducing and pulverising ores, and also with respect to their subsequent treatment.

The defects in the stamping-batteries in use in the Australasian Colonies are as follows:—

(1.) The cam-shaft is placed too far away from the stamp-shank.

(2.) The cams in many batteries are not properly curved.

(3.) The stamp-mortar is too wide for the diameter of the stamp-shoe used, and the stamps are placed too wide apart in the mortar.

(4.) The screens or grating are not properly set to produce a maximum discharge.

Position of the Cam-shaft.—There is scarcely a stamping-battery constructed in the colonies where the cam-shaft is less than 2in. distant from the stamp-shank, and there are many where the distance is over 3in. The result of this is they require more power to drive them, and they cannot be run at a high speed. This will be seen by any one after a few moments' reflection. Say, for illustration, that the diameter of the cam-shaft is $4\frac{1}{2}$in., and of the stamp-shank 3in., the distance from centre to centre if the two were close together would be $3\frac{3}{4}$in.; but, as it requires a clearing between them, so that in vibration the surfaces of the shaft and stamp-shank will not touch one another, of say, $\frac{1}{4}$in., then the fulcrum would be 4in. If a clearing of 3in. is left between the two, then the fulcrum becomes $6\frac{3}{4}$in., instead of 4in., which is ample to do the work. The difference in the power, therefore, required to lift the stamp on a fulcrum of $6\frac{3}{4}$in. beyond that required on a fulcrum of 4in. is in exact proportion to its length. If a lever 4ft. long had a fulcrum of $6\frac{3}{4}$in., and 800lb. were placed on the fulcrum end, it would require $112\frac{1}{2}$lb. on the end of the lever to balance it; but if the same lever had a fulcrum of only 4in. it would require $66\frac{2}{3}$lb. to balance it. The same thing applies in relation to the lifting of the stamp: the power applied is then as follows: As 112·5 : 66·66 : : the weight lifted : to the difference or saving in the power required to do the work. This clearly demonstrates that there is a great waste of power used in working our stamping-batteries. The saving in power if the batteries were properly constructed would soon pay for a new stamp-mill. Say, for example, that ten-horse power was required to drive ten heads of stamps, weighing 800lb. each, with the cam-shaft 6¾in. distant from the centre of the shaft to the centre of the stamp-shank, then it would only require six-horse power if the centres were only 4in. apart, thus showing a saving of four-horse power.

Curve of Cams.—The cams in general in the stamping-batteries in the colonies have not a proper curve, and the further away that the cam-shaft is from the stamp-shank the longer the cams have to be in the ends. A properly-constructed cam is formed in an involute curve, as by this construction the same point of the tongue is, during the lift of the stamp, raised vertically and uniformly, the lift of the stamp always having a proportion to the segment of circle described by the cam.

The Order in which Stamps should fall.—This is a subject on which there are many opinions, but the Americans advocate one of two orders, as having a great influence on the crushing-power of the battery. The order in which the stamps fall in many of the batteries in use is 1, 4, 2, 5, and 3, and 1, 5, 2, 4, and 3. The latter order is strongly advocated by Joshua Hendy, manufacturer of quartz-mining machinery, San Francisco; and he adduces the following reasons: No. 1 dropping first throws the water through the length of the battery, which No. 5 catches in its descent, throwing back, which No. 2 catches, throwing it back until No. 4 catches it, and the fall of No. 3 equalises the wave. This is theoretically correct, and in practice it shows a regular wave in the mortar, from the ends to the central point of discharge. He also states that the battery feed-water should always flow in at the back of the mortar with the ore: if arranged to do so in front it measurably prevents a perfect wave, and certainly precludes a perfect discharge of the pulp through the screens or grating, as it necessarily produces a swash in front of them.

Stamps in the Mortar.—The stamps are in general placed about ⅞in. to 1¼in. apart in the batteries constructed in the colonies, whereas ⅝in. apart is sufficient; and there is too much space left between the stamp-shoes and the side of the mortar. The principle is to have as little material in the mortar as possible, so that the stamp on falling does proper duty, and the blow is not cushioned by waste material lying in the mortar. The effect that a body of loose material has in the mortar is that the blow of the stamp on falling is deadened, and the loose material splashes about in the mortar, causing more wear and tear on the sides, and has a far greater tendency to make holes in the screens or grating. This is one of the great reasons that rock-breakers and ore feeders should be employed. The rock-breaker first reduces the ore to a small size, which can be uniformly fed by the ore feeder into the mortar, so that not more than 2in. in depth of material is under the falling stamp at any time.

The general usage in New Zealand is to feed the stamps by hand, and boys are employed for this purpose on account of cheap labour; but the feeding by hand can never be performed with so much regularity as by an automatic process, and stones of all sizes are thrown into the mortar, which cushions the blow of the stamp, causes more wear and tear, and the grating or screens are continually getting broken. The idea of cheap labour to take the place of automatic machinery, in a case like this, is purely one of false economy.

Grating or Screens.—The grating or screens attached to all the stamping-mills in the colonies are set vertically, so that it is entirely due to the force of the splash of the water from the falling stamps striking the grating that the pulp is driven through the holes or slots, as the case may be. The American stamping-mills have the screens placed on an angle, the top projecting outwards. By this arrangement the pulp is not only forced out by the splash of the water, but it runs down the face of the screen on the inside, and comes through the holes or slots in the screens continually.

The great desideratum in crushing and pulverising gold- and silver-ores is to have as little wear and tear as possible for the amount of material operated on, as the wear of the iron produces slimes which are refractory to extract the metals from. The duties of a battery-manager are not so simple as they appear at first sight, irrespective of his knowledge with regard to amalgamation, concentration, and keeping the plates in proper order. Joshua Hendy remarks that he has to carefully scrutinise the condition and operation of every part of the driving machinery, see that it is properly worked, and be able to remedy its defects; examine the ore-bins, the quantity and quality that is passing through the rockbreaker, and to see that every part is well oiled and running true, or if any parts need replacing to have this done at the earliest moment; see that the ore-feeders are properly adjusted to regulate the feeding of the ore; go over the battery-parts carefully, see that all the bearings are cool and properly oiled, that no grease can reach the mortars, that the drop of the stamps is up to the standard of speed, that the cams are tight on the shaft, and that the discs are set to a regular drop for each stamp, to prevent an unequal wear of the shoes and dies; never to take off the battery-screens unless necessity requires—such as, a shoe has dropped, or if floating wood obstructs the free discharge of the pulp; if a screen be taken off, to cleanse it thoroughly, and, if laid away for future use, to be certain that it is well dried and kept in a dry place. If the time between a clean-up and the commencement of another run be long, do not put the screens in place on the mortar, but store them in a dry room. This care will avoid rust, which weakens their texture and causes them to break.

There were several machines for crushing and pulverising ores exhibited at the Melbourne Exhibition which, although not of an entirely new pattern, are very little known in the colonies. Amongst these are the Huntingdon Centrifugal-roller Quartz-mill, the Bryan Roller Quartz-mill, the Cyclone Crusher, the Globe Pulveriser, the Dodge Pulveriser and Granulator, and several exhibits of pneumatic stamps.

Messrs. Parke and Lacey, of San Francisco and Sydney, had three complete crushing-plants, with stone-breakers and concentrating machinery. Amongst their machinery is the Huntingdon Centrifugal Roller Quartz-mill. These mills are made of two sizes—namely, 3ft. 6in. and 5ft. in diameter.

The mill having a diameter of 3ft. 6in. has a crushing-capacity of 12 tons of quartz per day of twenty-four hours. It weighs 2 tons 3 cwt., and requires four-horse power to work it at the proper speed, which is eighty-five revolutions per minute.

The larger-sized mill has a crushing-capacity of 20 tons of quartz per twenty-four hours, its total weight being 4½ tons. The power required to work it is equal to six-horse, which drives it at sixty-five revolutions per minute.

Before commencing to describe this quartz-mill it will be necessary to mention that a rock-breaker is required where one of these mills is used. The ore is first put through the rock-breaker and reduced to about 1in. in diameter, thence taken into a Challenge ore-feeder (see Fig. 1*a*), which feeds the quartz-mill automatically. This ore-feeder has a circular plate on the bottom, set on a slight incline towards the mill, on the under side of which there is a toothed wheel which is worked by a pinion having a very small rack-motion, which moves the circular plate about ¼in. at a time. This motion is sufficient to feed the ore as fast as the mill can crush it.

The mill itself consists of a circular cast-iron basin 3ft. 6in. in depth in the inside. There is a circular die-ring, against which the horizontal rollers are pressed with the centrifugal motion of the pan as it revolves. There are three rollers in the smallest mill, and four in the largest size. These are suspended by a cross-arm and shaft to the top side of the pan. The cross-arm on the top of the vertical shaft of each roller has bearings turned on each end, and these are fitted into plummer-blocks, so that the whole is suspended, and all allow the vertical shaft and horizontal roller to fly against the side of the pan when it is set in motion. Indeed, it is somewhat on the same principle as the governor of a steam-engine, only, instead of the rollers being balls, they are about 18in. in diameter, with a 6½in. face, each roller having a steel tire about 3in. in thickness, which can be easily replaced as the tires get worn out. The rollers are suspended about ½in. above the bottom of the pan, so that they never come in contact with the mercury which is placed in the bottom to collect the free gold. By this arrangement the mercury is not liable to get floured.

The die-ring is placed at the bottom, and directly above this the gratings are fixed, one in front and one at each side, so that it may be said there is a grating round the pan for about one-third of its circumference. The gratings are 8in. in height, and those that were used at the mill worked in the Exhibition had about No. 15½ mesh, or 240 holes per square inch.

As soon as the pan is set in motion the rollers fly outwards like the governors of a steam-engine, and revolve with the pressure against the die; and as the ore is fed into the mill it is impelled by centrifugal motion towards the outside of the pan, and is ground up between the rollers and the die-ring, while the water and pulverised material pass through the screens on to an inclined table, which is covered with electro-plated copper plates, and after passing over this table the material goes into a well in which the end of a steam-ejector is placed. This ejector lifts the water and tailings into a hopper which feeds a Frue vanner, and there all the concentrates are saved. Quicksilver is used in this mill to collect all the free gold; but, as before described, the rollers do not come in contact with the silver, but are suspended directly above it. This is done to prevent the mercury from being broken up, which causes flouring.

This mill is highly spoken of in the *Mining and Scientific Press*, of San Francisco, and has been in general use on the Pacific Slope for several years. The cost of these mills landed in Auckland is as follows: Small size, 3ft. 6in. in diameter, £250; large size, 5ft. in diameter, £375. The former size has a crushing-capacity of 72 tons, and the latter of 120 tons per week—that is, working night and day.

The following are the advantages that the manufacturers claim for this mill; and these advantages are bound to be acknowledged to a great extent by any one conversant with quartz-crushing machinery:—

(1.) The cost of a mill of the same capacity is not more than half that of stamps.

(2.) The cost of transport about one-fourth that of stamps.

(3.) The cost of erection about one-tenth that of stamps.

(4.) These mills only require about one-third the power of that of stamps of equal crushing-capacity.

(5.) The wear and tear is much less than that of stamps, and the wearing-parts are easily duplicated.

(6.) This mill leaves the pulp in a better condition for concentrating than the stamp-mill, and it is also a better amalgamator, as it saves about nine-tenths of the gold in the mill; and its simplicity of construction obviates the need of mechanical skill. The rotary method of crushing the ore so granulates the pulp—which is discharged the moment it is crushed—that a complete concentration of ore containing sulphurets is rendered most easy.

The superintendent of the Paradise Valley Mining Company (Mr. J. V. McCurdy) states that his company used three of the small-sized mills, 3ft. 6in. in diameter. During twelve months these mills crushed 7,631 tons of hard silver-ore at a cost of 3s. per ton, including all wear and tear of machinery. Mr. T. G. Morgan, Superintendent of the Pittsburgh Mill and Mining Company, states that his company has been running two of these mills for two years, and find them superior to the stamp-mills, the cost of crushing hard quartz being a trifle more than one-half that by stamps. The wearing-parts are easily and speedily removed when worn out, and replaced with new ones—thereby keeping the mill almost constantly in motion. He also states that from 80 to 85 per cent. of the metals in the ore is saved in the mill; which proves it to be a very good amalgamator. The mills run at a speed of from eighty to eighty-five revolutions per minute, and crush, on an average, 25 tons in twenty-four hours through a No. 8 slot-screen, which is equal to a No. 40 wire-screen; and a twenty-horse-power engine can work four of these mills, besides a No. 1 Dodge rock-breaker.

Annexed is a sketch of the Huntingdon quartz-mill, which will enable the above description to be more clearly understood. (See Fig. 1.)

Cyclone or Wall Rolls.

This machine is one of the greatest crushers or pulverisers that are made. At the same time a crushing-plant with only one set of these rolls would not do for crushing quartz to the desired degree of fineness. Two sets are required, with revolving screens under each set, and what will not pass through the first screens goes into the next set of rolls and is again screened, and the residue from the inside of the last screen falls into a chamber, and is then lifted up by elevators into the ore-feeder, and is again put through the rolls.

As far as their crushing-capacity is concerned, they will crush a far larger quantity of material than any stamp-battery with the same power employed to work them; but they are specially adapted for dry crushing. If the material was wet and mullocky the rolls would be likely to clog, and the crushed material would not go through the screens. Therefore, for crushing wet mullocky quartz they could not be used to the same advantage; but if the material was hard, dry, solid quartz, their crushing-capacity could not be excelled. The manufacturers claim that they can be used for wet crushing as well as dry; but this appears to be misleading, as any one examining their construction will see that they cannot be used for crushing wet mullocky material to the same advantage as a well-constructed stamping-battery.

Their crushing-capacity is merely a question of length of face and the speed they are driven at. They are not limited to speed, like a stamping-battery, which is limited to such a speed that the cams will not strike the falling stamp. It has been found in using these rolls that the greatest economy is gained by increasing the width of the face, and not the diameter of the rolls. The shells or rolls are made of crucible cast-steel, and wear evenly, as is not the case with the common smooth-faced rolls. A sketch of these rolls was given in my annual report last year.

The faces of the rolls consist of a series of parallel corrugations extending across the face of the shells, either parallel or inclined to their axis, the corrugations being rounded or curved with such proportions that when intermeshed and rotated every portion of the surface will press equally upon the counterparts of the opposite roller, and, being held firmly in position by steel gear, slipping of the crushing-faces upon each other or upon the material to be crushed is rendered impossible. They are sexagonal in shape, with the corners rounded and the sides hollowed out, fitting into each other on the same principle as Root's or Baker's blowers. When at work the faces of these rolls must not touch, but be held apart by check-nuts. If a fine product is required the rolls may be placed within $\frac{1}{8}$in. or even $\frac{1}{16}$in. of each other, but if the ore be coarse they require to be from $\frac{1}{4}$in. to $\frac{3}{8}$in. apart. The meeting- or crushing-faces present at all times overlapped curved surfaces, between which the material is firmly held and crushed by almost direct pressure, thus avoiding the grinding or uneven wear of the rolls. It is well known that any rapid grinding-surface used in crushing ores produces refractory slimes. These rolls do away with this to a very great extent. Each set of rolls is supplied with four large and four small gears, the smaller to take the place of the larger as the rolls become worn; and also with each mill a wall feeder is provided. The following prices were quoted by the agents for delivery at Auckland:—

Diameter.	Length of Face.	Approximate Weight.	Net Cash Price.	
In.	In.	Lb.	£	
16	14	6,000	390	Feeder,
16	16	6,800	435	£45
16	20	7,500	470	extra.

The wall feeder is especially designed for feeding these rolls, and will feed any material, wet or dry, with perfect regularity. It is driven by belt attached to small pulley on the roller-driving shaft.

The following prices were handed me by the agents of Parke and Lacey for a complete crushing-plant in connection with Wall or Cyclone rolls:—

1 8in. by 10in. Giant rock-breaker	£115
1 automatic ore-feeder	50
2 sets of Cyclone rolls, 16in. by 10in.	520
1 revolving sizing-screen, with shaft, gearing, &c.	80
1 automatic link-belt elevator	60
1 3in. shaft, 20ft. long, with all the necessary pulleys and belting ...	140
Total cost	£965

They also supply a complete set of plans for the erection of the plant.

With this plant the agents will give a guarantee that it will crush 20 tons of dry ore, or 25 tons of wet ore, per day of twenty-four hours, with screens of 200 mesh. The total weight of the whole of the machinery is 14,000lb. If the plant is for wet crushing there is nothing included in the price given for supplying water-pipes. The price is also for the whole of the machinery landed in Auckland, and the terms are one-quarter cash on order, one-half on delivery, and the balance to stand as against the guarantee that the machine will do the work specified; but the agents require 7 per cent. interest on the balance of money until paid. The address of the manufacturers is, "Parke and Lacey, 189, Clarence Street, Sydney."

Bryan Roller Quartz-mill.

This is a quartz-crushing mill, manufactured by the Risdon Ironworks Company, San Francisco. (See annexed sketch, Fig. 2.) It consists of three steel-tired rollers, 30in. in diameter, with a face 6in. in width, each weighing 1,200lb. These rollers run over steel dies in a circular mortar somewhat on the principle of a Chilian-mill basin, having a round post or shaft in the centre keyed on to basin, and this shaft or upright post is for keeping all the gearing in position when the mill is at work. Each roller has a steel tire 3in. in thickness, and one set of tires is said to be capable of crushing 1,600 tons before requiring renewal.

The axles are fixed solid in the rollers, and are journalled on an annular plate which rotates loosely around the centre post or shaft. All bearings are lubricated by suitable oil-channels, so as to prevent any oil or grease entering the mortar. An iron tank, or drum, is placed on the top of the rollers, which serves as a pulley to drive the mill. This tank is of the same diameter as the outside of the circular basin, which varies according to the size and capacity of the mill, and it has a depth of about 3ft. 6in. This tank, or driver, also acts as a weighted load on the top of the rollers, and increases their crushing-capacity in proportion to the weight of material placed in the tank.

The rollers are kept clean from the pulp of the crushed material by an adjustable spring and scraper, which follows each roller in its rotation. The spring is set to the die, and its tension is readily adjusted. The direct purpose of the scrapers is to keep the face of the die, and also the rollers, clean from pulp, which would otherwise collect and adhere to the rollers, and reduce their crushing-force. The scrapers serve also to discharge the pulp, and to distribute and equalise on the die-ring of the mortar the ore received from the ore-feeder.

The mortar is provided with three openings for the screens, 4ft. long by 6in. wide, which allows ample surface for the discharge of the pulp from the mortar. There is a launder extending round the mortar, to carry off the pulp on to an inclined table which is covered with electro-plated copper plates.

When the mill is in operation the pulp runs round the mortar next to the screens at a velocity of about 300ft. per minute, but towards the centre—that is, on the inside of the rollers—the current is much slower, and as the gold is liberated from the matrix it falls to the eddy-side of the current, and is in practice found amalgamated in mass round the bottom of the centre cone of the mortar.

The movement of the mill is at the rate of fifty to fifty-five revolutions per minute. A mill 4ft. in diameter is said to have the same crushing-capacity as ten heads of stamps, and only requires about one-half of the power to work it. The mortar, with launder and spout, is usually cast in one piece, but for transportation in hilly country it is cast in sections weighing about 300lb. The whole weight of the mill is about 5½ tons. The manufacturers claim the following advantages for this mill over any other—namely :—

(1.) It is cheaper than any other mill.
(2.) It is more durable.
(3.) It crushes a larger amount of ore with less power.
(4.) It amalgamates a larger percentage of the gold.
(5.) Its clean-up is quicker.
(6.) It is portable, requiring no framework, and is ready for the foundation as soon as it leaves the shop.

The price of a 4ft. mill complete is £360 (this includes electro-plated copper plates), and its crushing-capacity is from 12 to 16 tons per day of twenty-four hours. The cost of a 5ft. mill is £500, and its crushing-capacity is from 25 to 30 tons of hard ore per day. The whole of this description of mills requires stone-breakers to be attached to them.

Globe Crushing-mill.

The Globe Crushing-mill was exhibited at the Exhibition as a mill suitable for pulverising quartz to the greatest degree of fineness. It was 5ft. in diameter, with a cast-steel ball of about 12in. diameter. This mill is used in England for grinding clinker for cement; but, judging from its construction and the power required to work it, the mill would not be an economical one for reducing quartz, and is not likely to come into general use in the colonies for that purpose. Two of these mills have been erected at Waihi by the Waihi Gold- and Silver-mining Company, but they are only about 2ft. 6in. in diameter. They had not been tested at the time of my visit to that company's works, and therefore nothing can be said definitely respecting their capabilities; but, judging from the working of the large mill erected at the Exhibition at Melbourne, it is very questionable if this company will continue its use for any lengthened period. Annexed is sketch of machine, Fig. 3.

This mill consists of two discs keyed on to a horizontal shaft, and so placed that in revolving they cause a circular steel ball to travel at the same rate as their periphery. These discs and ball are enclosed in a cast-iron casing lined with steel, having a half-round path in the inside for the ball to rotate in. On each side of the discs inside the cast-iron casing there is a fine wire-grating placed, through which all the fine dust and sand of a certain fineness passes into a chamber where there is an exhaust-air fan, which draws out the dust into a settling receiver, ready for further treatment by amalgamation or chlorination process.

The manufacturers state that the only wearing-parts of this mill are the grinding-path in the casing and the ball, which are both made of hard steel. According to the tests made in England in grinding cement, the ball loses in weight about ½oz. to every ton of cement ground, and the ball can lose 10lb. in weight before a new one is required. Mr. Henry Faija, M. Inst. C.E., in reporting on one of these mills, states that one ball may grind over 300 tons of cement before requiring to be renewed, and the grinding-path, so far as he could determine, does not wear to any appreciable extent. The mill he had been experimenting with had ground some 100 tons of various materials, such as quartz, basic slag, &c. The sand-marks in the casting were still plainly visible.

The size of the mill Mr. Faija was experimenting with was 30in. in diameter, with a ball 8¼in. in diameter, weighing 85lb., and also with a ball only 54lb. weight. He found the lighter ball the most effective. He gives the result of two experiments, both of which extended over one hour each :—

1. Speed of mill, 300 revolutions; speed of fan, 2,000 revolutions; weight of ball used, 85lb.; quantity of cement ground, 2,666lb.; indicated horse-power absorbed by mill and fan, 28·5; indicated horse-power reduced to tons per hour, 23·8; fineness of grinding, 3 per cent. residue on 100 × 100 sieve.

2. With same number of revolutions of mill and fan—weight of ball used, 54lb.; indicated horse-power of mill and fan, 29.5; indicated horse-power reduced to tons per hour, 22·4; fineness of grinding, 4 per cent. residue on 100 × 100 sieve.

Although these mills are said to give very satisfactory results in crushing cement, they cannot be commended as economical mills for reducing quartz, and they are only suitable for dry crushing.

Rock-breakers required.—In using either the Huntingdon Mill, Cyclone Rolls, Bryant, or Globe Mills, the ore must first be put through a rock-breaker, and be reduced to something like a maximum size of about 1¼in. diameter. The same principle also holds good in crushing with stamps, in order to reduce in cost of crushing. As regards those mills, the ore must be broken small before they will work properly; so that in estimating the cost of a plant a rock-breaker has to be included as well as a proper ore-feeder. Rock-breakers and ore-feeders have not as yet, with exception of the new mills in the North Island district, been used in connection with crushing-batteries; but the day is not far distant when they will be attached to all the mills in the colonies. The old conservative notions must give way in order to utilise the low-grade ores, which require an inexpensive method of treatment to make them pay for working. The rock-breaker is decidedly the class of machine that will reduce the ore to a medium size at a less cost than any other that has yet been used. It has very few wearing-parts, and those parts can be replaced at a small cost and also very quickly. It is totally against all natural laws to attempt to crush large blocks of stone quickly in a mortar. One has only to try this on a small scale by putting a large stone in a common hand-mortar. The pestle will break the stone, but if the quantity of material in the mortar is large it will take double the time to pulverise it to a sufficient fineness that it would take if the quantity were divided in two or three lots and pulverised separately. Therefore as long as large blocks of stone are put into a stamp-mortar we can never expect to crush cheaply. After the ore has been broken by a rock-breaker an ore-feeder is indispensable, as the greatest regularity has to be observed in keeping a uniform quantity of material; and this can never be done so well by hand as by an automatic feeding-machine.

Marsden's Fine Crusher or Pulveriser.

This is a machine which commends itself as an economical dry-pulveriser. It will be seen on referring to the sectional elevation of the machine appended hereto (Figs. 4 and 5) that the power required to work it and the quantity it will pulverise per hour will necessarily vary according to the nature of the material under treatment; but in any case its crushing-capacity for the power employed to drive it is as great as any machine now in use. It will be seen that it is merely a rock-breaker with the jaws set close together at the bottom, and having a revolving screen set under it. The material which does not pass through the screen falls into a chamber and is lifted with elevators to be treated again, on the same principle as that used in crushing with rolls.

The wear and tear in a machine of this description must be very small, as there are only four wearing-parts—namely, the two jaw-faces and the two side-plates—and these can easily be removed and changed in about half an hour by an ordinary labourer. These machines are made of six different sizes at the mouth—viz., 5in. by ½in., 6in. by 1½in., 10in. by 2½in., 12in. by 3in., 20in. by 3in., and 20in. by 5in. As previously stated, their capabilities are according to the nature of the material and the degree of fineness to which it has to be crushed. The following table will give approximately the products and power required:—

Size of Machine at Mouth.	Approximate Product per Hour to pass Ordinary Gauze of 2,500 Holes to the Square Inch.	Nominal Horse-power required to drive.	Price of Machine, Woodwork not included.	Price of Screening Apparatus	Weight of Machine.	Prices of Best Copper-sewn Belts. (Extra.) Double, Driving.	Prices of Best Copper-sewn Belts. (Extra.) Single, Screening.	Speed of Pulley.	Extra for Machine arranged on a Bed-plate, with Screening Apparatus overhead, supported on Iron Pillars, in which case the Wood Framing shown in Sketch is done away with.
In. In.	Cwt.		£	£	Tons cwt.	£ s. d.	£ s. d.		In. In. £
*5 by ½	½	1 man	21	5	0 1*	0 19 6	0 11 0	300 revolutions per minute.	.. .. ..
6 by 1½	2	2 men	85	15	0 10	2 12 0	0 16 6		.. .. ..
10 by 2½	7½	3 n.h.p.	130	25	2 5	7 4 0	2 2 0		10 by 2½, 25
12 by 3	10	4 n.h.p.	150	30	3 0	7 4 0	2 2 0		12 by 3, 25
20 by 3	16	6 n.h.p.	250	40	4 7	8 16 0	3 0 0		20 by 3, 35
20 by 5	20	12 n.h.p.	300	45	6 10	10 14 0	3 0 0		20 by 5, 40

* Sampling machine.

Blake-Marsden Stone-breaker or Ore-crusher.

This is one of the best stone-breakers and ore-crushers that are made. Everywhere they are erected they give satisfaction. The Blake stone-breaker is an old machine, and well known. This machine has been so improved in its details that it is far superior to those of the old type. One of the great improvements is its toggled jaws. The position of the teeth can be altered upon the front toggle up or down upon the cushion of the lever. The motion or length of the stroke of the jaw can be increased or decreased, and the material operated on can be crushed to any size required.

There is a false back to the machine, accurately planed and fitted to the frame, against which three jaw-faces are bedded, thus avoiding any degree of concussion, and also providing a means whereby these faces can be renewed or reversed in position in a few minutes at any time. These faces are fitted with surface-strips on the backs, which also gives a dead bearing. The swinging-jaw is also planed, and the wearing-faces of this fitted in the same manner as the others. The whole of the shafts and axles are made of the best hammered steel, and the toggle-cushions are made of solid crucible cast-steel. The motion obtained by the use of the toothed toggle and lever-cushion gives an interrupted movement to the jaw to suit any kind of material, and this prevents any clogging taking place. (See annexed sketch, Fig. 6.) The following table shows the sizes, weight, prices, and products:—

Size of Machine at Mouth.*	Approximate Product per Hour to Road-metal Size.	Power required.	Weight of Frame.	Total Weight of Machine on Feet.	Weight of Wheels, Axles, and Horse-shafts.	Weight of Automatic Screening Apparatus.	Price of Machine on Feet.	Price of Wheels, Axles, and Horse-shafts complete. (Extra.)	Price of Automatic Screening Apparatus. (Extra.)	Best Copper-sewn Belts.		Speed of Pulley.
										Double, Driving.	Single, Screen'g.	
In. In.	Cwt.	N.H.P.	Tons cwt.	Tons cwt.	Tons cwt.	Cwt.	£	£	£	£ s.	£ s.	
10 by 8	83	3	2 5	4 5	0 17	4	105	12	10	8 0	0 18	250 to 300 revolutions per minute.
12 by 8	108	3	2 10	4 10	0 17	4	113	12	10	8 0	0 18	
12 by 9	112	3	2 12	4 12	0 17	4	120	12	10	8 10	0 18	
15 by 8	125	5	3 0	5 15	1 0	4	135	15	13	9 15	0 18	
15 by 10	150	6	4 0	7 0	1 8	5	150	15	13	10 15	0 18	
18 by 9	175	7	4 10	7 17	1 8	5	165	15	13	10 15	0 18	
20 by 10	200	8	5 15	10 0	1 8	5	180	17	15	10 15	1 1	
24 by 13	300	10	7 7	13 10	1 10	6	263	25	16	15 0	1 10	
24 by 17	325	14	7 12	14 5	1 12	6	282	25	17	15 0	1 10	
24 by 19	350	16	10 0	18 0	1 12	6	300	25	17	15 0	1 10	
30 by 13	350	16	8 10	14 5	1 12	6	300	25	18	15 0	1 10	
30 by 15	378	16	8 15	14 15	5 5	7	315	25	18	17 10	1 15	
30 by 18	406	18	9 0	15 0	5 10	7	375	27	20	20 0	2 0	
24 by 24	406	18	11 0	17 10	6 10	7	375	30	20	22 10	2 5	
30 by 24	462	20	12 0	19 0	8 0	7	413	40	25	22 10	2 5	

* These dimensions denote the size of stone, &c., each machine will take in.

The Dodge Rock-breaker.

There are three sizes of these machines made. They are specially adapted for ore-crushing, and can be adjusted to crush to any fineness required. All the wearing-parts—namely, the jaws—are studded with hardened-steel pins, which are set thickly into a plate of untempered steel. The plate wearing away from the pins leaves them projecting as high points, which split the rock in pieces with the minimum amount of power.

The advantages claimed by the manufacturers of this rock-breaker are as follows:—

(1.) The trouble of adjusting or setting up jaw-plates is avoided in this rock-breaker, because the steel pins wear so much longer than the plates in other machines.

(2.) This rock-breaker can be set up to do much finer work than most machines now made, and the breaker has been thickened up so that it is twice as strong as it was when it first came out.

(3.) Rubber springs are used over the caps of the journal-boxes that the eccentric revolves in, so that when the eccentric is worn out of round the journal-box will not break—an advantage other machines do not possess.

The means of adjustment are very simple. The jaw-shaft rests in movable boxes. To change the size of the material crushed it is only necessary to loosen or tighten and place packing-blocks on either side of the movable shaft-boxes. If for fine crushing put the packing-blocks on the side of the movable boxes towards the eccentric, and if for coarser crushing put the packing-blocks in front of the screws, always bearing in mind that the screws should be set up tight with lock-nuts before commencing crushing.

Annexed is a sketch of this rock-breaker, which represents the middle size (see Fig. 7), which has only one fly-wheel, but the other two sizes have two fly-wheels. All sizes are supplied with fast and loose pulleys.

The following table shows the dimensions, capacity, price, and power required:—

Number.	Size of Jaw-openings.	Actual Horse-power required.	Capacity in Tons per Hour.	Revolutions per Minute.	Weight complete.	Price in Auckland.
					Tons. cwt. qr.	£
1	12in. by 8in.	8 to 12	2 to 5	220	2 3 3	105
2	8in. by 7in.	4 to 8	1 to 3	235	1 18 2	65
3	6in. by 6in.	2 to 4	½ to 1	275	0 10 3	30

Giant Rock-breaker.

This is a more powerful rock-breaker than the Dodge, and can be arranged to crush either coarse or fine. The annexed sketch (Fig. 8) will show the difference between it and the Dodge. It will be seen that the jaws are similar to those of the Dodge, being studded with hardened-steel pins. There are three sizes made, as will be seen from the following table:—

THE

HUNTINGTON

Centrifugal Roller Quartz Mill

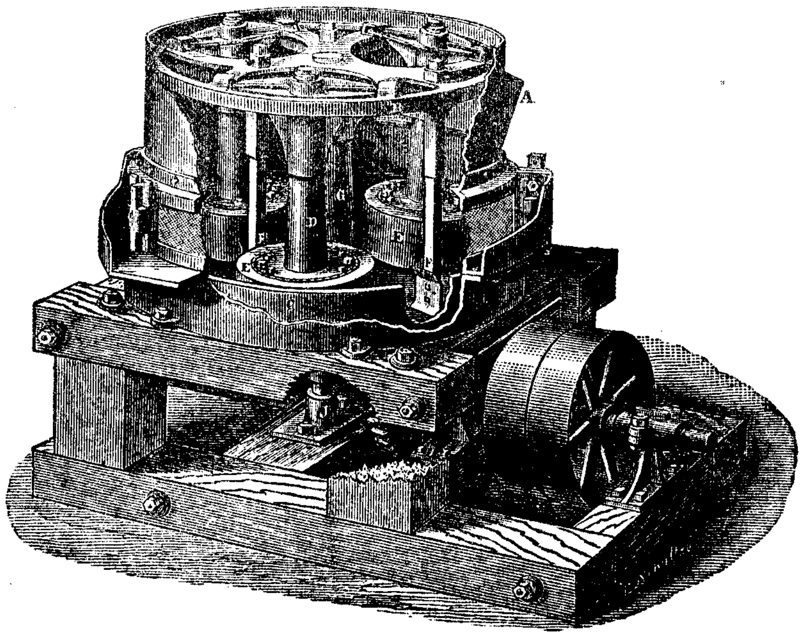
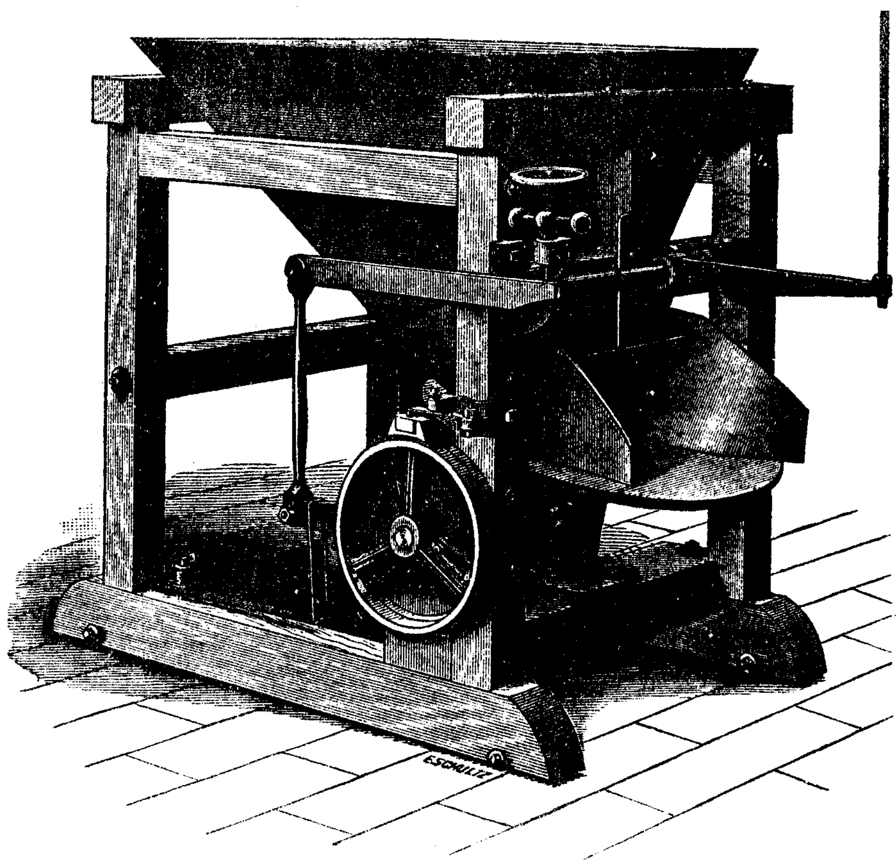
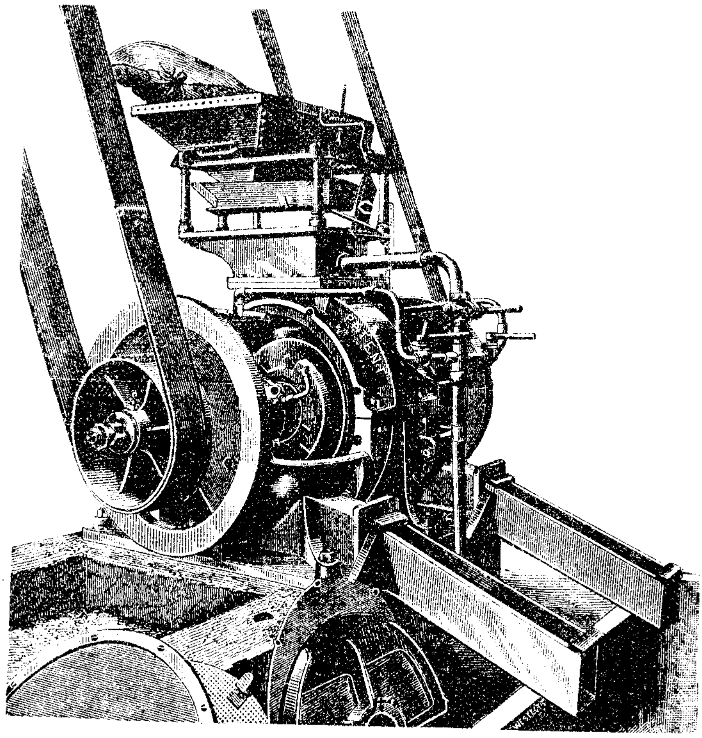


Fig 1

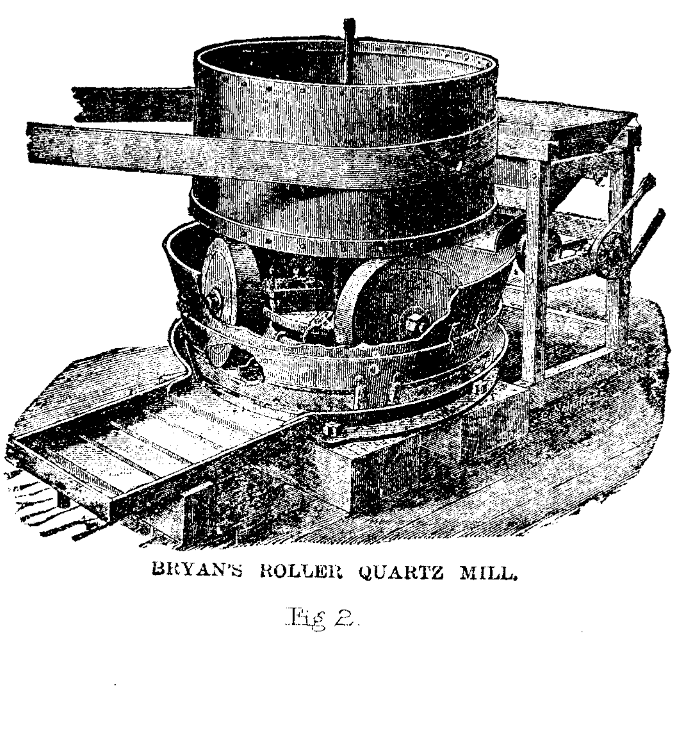
CHALLENGE" ORE FEEDERS





THE GLOBE MILL

Fig 3.



BRYAN'S ROLLER QUARTZ MILL.

Fig 2.

MARSDEN'S New Patent

Fine Crusher or Pulverizer

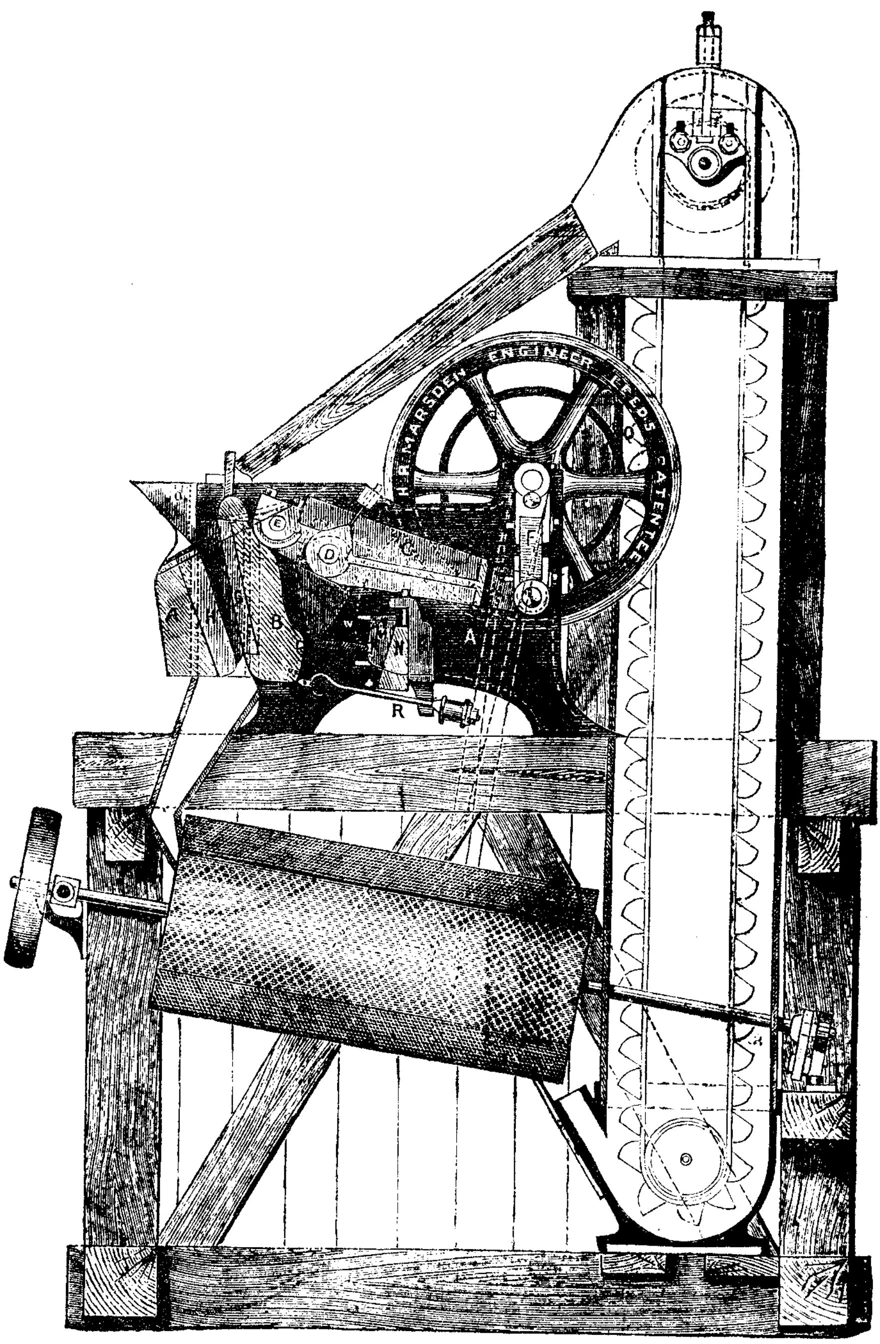
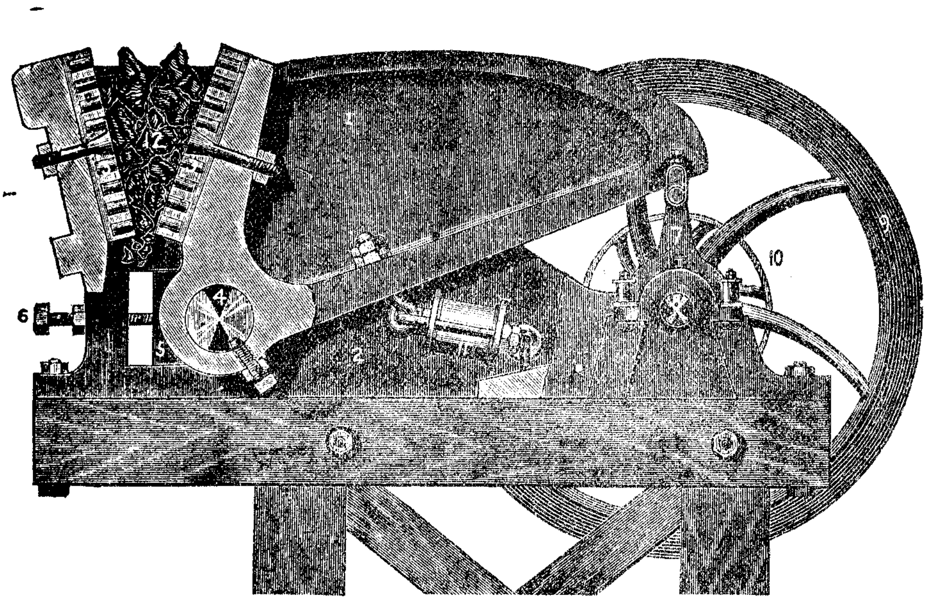


Fig. 4.

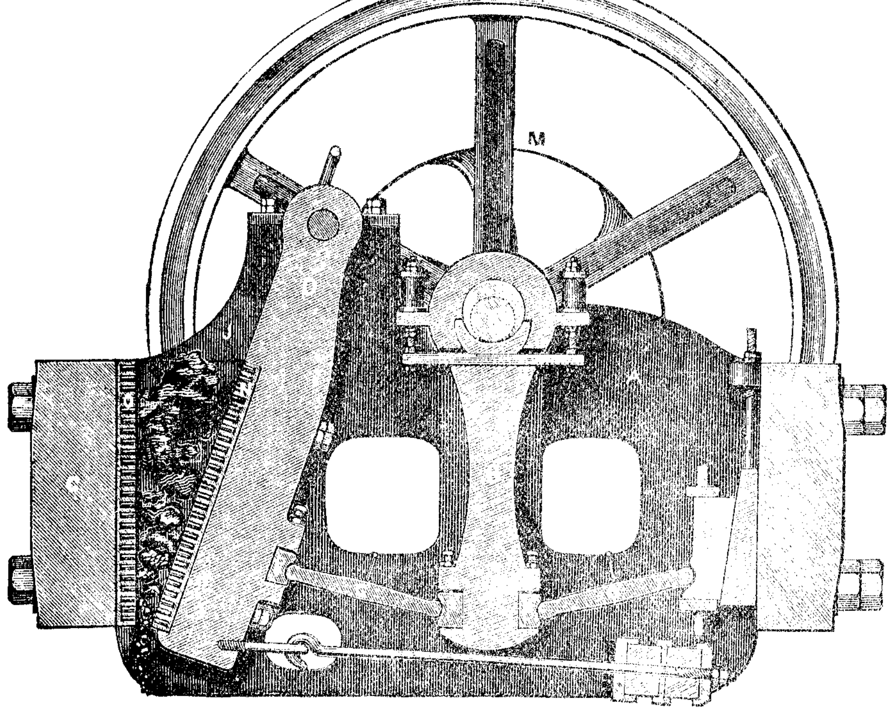
Dodge's New Improved Rock Breaker.

Fig 7.



GIANT ROCK-BREAKER.

Fig 8.



DODGE'S PULVERIZER & GRANULATOR

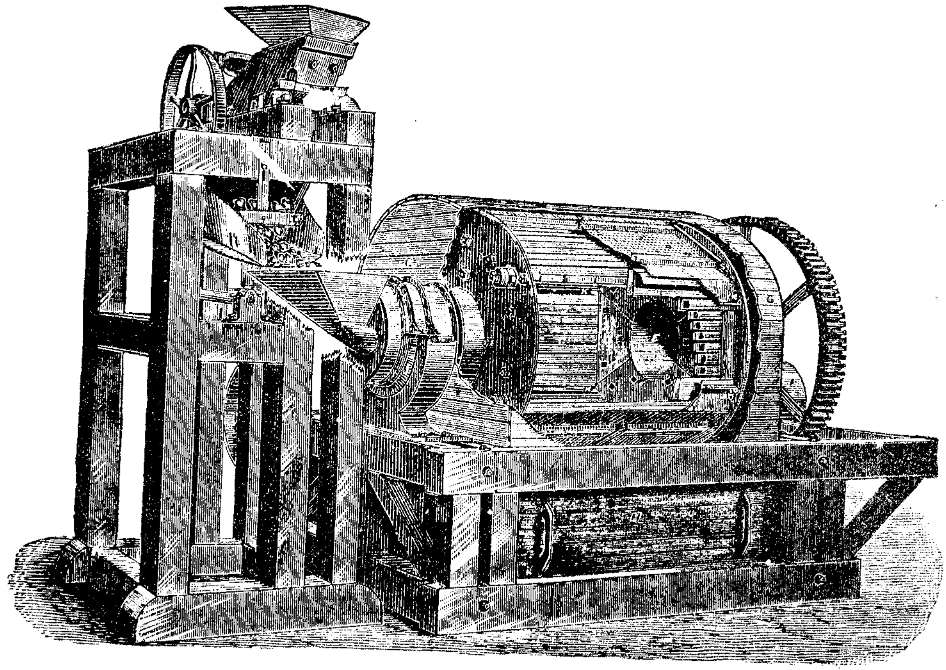


Fig 9.

Size of Jaw-openings.	Horse-power required.	Capacity in Tons per Hour.	Revolutions per Minute.	Weight.	Price, delivered in Auckland.
				Tons cwt. qr. lb.	£
10 by 8	8	6	250	2 1 0 8	115
15 by 9	10	8	225	4 18 1 0	170
20 by 15	14	11	150	15 12 2 0	600

This rock-breaker was in use at the Exhibition, and did its work extremely well. The quartz that was put through it was very hard—equal to any quartz found in New Zealand.

In calculating for a crushing-plant on the centrifugal-roller system a stone-breaker has to be included. Therefore, in making comparison with the cost of a stamp-mill having an equal crushing-capacity, the cost of the mill and breaker has to be taken as one. The cost of a Huntingdon Centrifugal-roller Mill 3½ft. in diameter, and a No. 2 Dodge Crusher, would be £315: but a Dodge Crusher would be sufficient to keep four Huntingdon mills at work; therefore the larger the plant the more cheaply the material could be pulverised.

Dodge Pulveriser and Granulator.

This pulveriser has been used in America for several years, and is highly spoken of by Professor Egleston in his book recently published on the metallurgy of silver and gold. It is eminently suitable for dry crushing, being far superior to stamps; and, as many of the ores in the North Island of New Zealand will require to be crushed dry and chlorinated before anything like a fair percentage of the metals can be obtained from the ore, these machines deserve some attention as a crusher to replace stamps. They are not only used in America as dry-crushers, but also used for wet crushing, and are considered far preferable to the stamping-battery, as no dust or slime is formed as in a stamp-mill, which causes considerable loss in concentration and leaching.

For pulverising and granulating the ore, and for concentrating and leaching, this machine requires no elevators or screens on reels to produce certain-sized ore, the same as rolls do, as the screens are on the machine itself, and any size may be used to get the required fineness for the character of the ore to be dealt with. The manufacturers state the best result is obtained from this machine when rolls of 1,200lb. in weight are used in the largest machine, which is 4ft. long and 4ft. in diameter, and of 600lb. in the small machine, which is 3ft. long and 3ft. in diameter. These rolls are made of steel 3in. in diameter, and cut into lengths at the roll-mills of 3in. each. This class of rolls is said to give far greater results than a round ball, inasmuch as the latter runs in many cases ahead in the machine, and does not pulverise so well. Even hard pieces of rock or iron can be used in this machine to pulverise with.

The machine is a polygon of six squares in shape, and there is not a great deal of wear on it. The rock does not slide in a mass as in a cylindrical pulveriser, but falls over at each angle on itself, thus throwing the wear on the ore to a great extent, more than on the machine. The greatest wear is on the grate-bars and cheek-plates, which are made of solid steel, and are said by the manufacturers to last for a long time.

There are two sizes made of this machine, namely: No. 1 machine has a crushing-capacity of 25 tons per twenty-four hours, and the price of the machine in Auckland is £430; No. 2 machine has a crushing-capacity of 10 tons in twenty-four hours, and the cost, delivered in Auckland, is £225.

When set up a suitable platform is erected to stack the ore, and for a person to stand to feed the rock-breaker, which acts as a feeder to the pulveriser as well as a crusher. The work is evenly divided between the two machines, so that they work evenly and together. The steel grinders, or rocks, or whatever is used to pulverise with, are placed in the machine at E (see sketch annexed, Fig. 9), being the opposite end of the machine from the rock-breaker, and more rolls or pieces of rocks are added as these wear away by grinding up. The grate-bars are held in place by cheek-plates D, and iron wedges K. As the ore gets pulverised it passes through the grate-bars on to the screens F, the fine ore sifting through into the box H or into conveyors, and the coarse particles coming back into the pulveriser through the grates of the wedges K. The machine has to be run in a certain way to allow these coarse particles to come back into the machine to be reground fine enough to pass through the screens. The wooden housing G effectually confines the dust in the machine. There are tight and loose pulleys at I, turning a pinion geared into a large spur-wheel, which revolves twenty-five times a minute in the No. 1 machine, and thirty-two times in the No. 2 machine. J J are the bearings, which are made of lignum vitæ.

There were several other crushing-plants exhibited, on the stamp-mill principle. One of these was a direct-acting steam-stamp similar to that manufactured by Fraser and Chalmers, of Chicago, only much smaller than a machine of this form which was being erected at the Broken Hill Proprietary Mine at the time of my visit there. The direct-acting steam-stamp at the Exhibition was not working, and therefore no idea could be formed of its capabilities. The stamping-mills exhibited were of the ordinary pattern with the exception of the semicircular battery before referred to, and in which there was nothing to commend it for use in the colony. In the German Court there was a disintegrator for pulverising; but this machine would be of little advantage to operate on any of the ores in New Zealand. As stated previously, there were very few new reduction-machines exhibited that would be of service to mill-proprietors in the colony, and therefore a description of each machine would be of no benefit.

All the serviceable exhibits for reducing ores have been described, and plans annexed hereto will enable any one to follow the description and understand their design. The greatest want now is, for those who have crushing-plants already erected, to get good machinery for concentrating the ore. This will be fully explained when dealing with the different classes of concentrators.

ORE-CONCENTRATORS.

The concentration of ore is a subject which deserves attention in New Zealand. There is no one single method of treating auriferous and argentiferous ores that will answer for all classes. Some ores are suitable for smelting, some are more profitable to crush dry, then roast, and either chlorinate, leach, or amalgamate; while in others, which are termed free-milling ores, the precious metals can be partly saved on quicksilver-tables, blanket-tables, and then run over a concentrator, and the concentrates either smelted or roasted, and the gold extracted by chlorination, the silver by leaching, or both these roasted concentrates can be treated by pan amalgamation. The concentrating machinery in use in Victoria is principally Haley's tables and common concave buddles, and a very few of the Frue vanners. The principal objection to using more of the latter is their price, which is about £150 each. This becomes a great item where there are large reduction-plants, as a Frue vanner is required to concentrate the ore coming from every five heads of stamps, and in many instances two vanners are required to produce proper concentration; but, notwithstanding their price, there is no concentrator at the present day equal to them and the Triumph, and they are very suitable for a great deal of our ores, especially those found in the Middle Island.

About four years ago a description of those concentrators was given in my report on mining, and mining machinery in use, in the Australian Colonies. After having again visited many of the Australian mines and reduction-plants, there does not now appear to be any concentrator equal to them. In treating pyrites-ores they are invaluable, and will soon pay for the first cost. It is simply ruinous to mine-owners to go on year after year as they are doing in New Zealand, treating pyritiferous ores by the ordinary battery-process, without taking any steps to concentrate the tailings. Many of the mine-managers, and even those employed at the crushing-batteries, are so wedded to the old process of extracting the gold that it is only sheer necessity that will get them to try any new method. This is to be accounted for, to a great extent, by the fact that they are not able to make assays themselves of the residue which is run to waste; and credit must be allowed them to this extent, that they conscientiously believe they are getting a fair percentage of the precious metal that the ore contains. But since the establishment of the schools of mines in the principal quartz-reefing districts they are beginning to learn that the loss of gold by the process hitherto adopted is far beyond anything they conceived, and no doubt the day is not far distant when their thoughts will be directed into other channels, and more attention will be paid to the reduction and treatment of auriferous quartz. Indeed, many of the lodes will not pay for working unless a different process of treatment is adopted. This does not refer to the refractory ores only which are found in the North Island, but also to those found in the Middle Island, which may be termed principally free-milling ores. It is in treating these that the use of the Frue vanner and Triumph concentrator would make a great difference in the returns from many mines.

Description of the Improved Form of the Frue Concentrator.

By referring to the annexed plans of this concentrator (Figs. 11, 12, and 13) the description will be more clearly understood. A A are the main rollers that carry the belt, and form the ends of the table. Each roller is 4ft. 2in. long and 13in. in diameter, made of galvanised sheet-iron. The bolts which fasten the boxes of A A to the ends of F also fasten to F the upper supports, which rest on uprights N. The rollers B and C are of the same diameter, and are made in the same way as A A. The roller-post of C is shorter than that of A A and B, and also has rounded edges, the upper portion of the belt, with its edges, passing over it. The belt E passes through water underneath B, depositing its concentrations in the box No. 4; and then, passing out of the water, the belt E passes over C, which is the tightening roller. B and C are hung to the shaking-frame F by hangers P P, which swing on the bolts fastening them to F. By means of hand-screws B and C can be adjusted on either side, thus tightening and also controlling the belt.

The boxes holding A A in place have slots and adjusting-screws, so that by moving them out or in A A can be made to exert a very strong influence on the belt E, and, as the belt sometimes travels too much towards one side, this tendency can be stopped most quickly by lengthening or shortening on one end or the other of A A, always remembering that a belt travels to the high side of a pulley. The swinging of the rollers B and C also controls the belt. C C are bolts and washers to take up the end-play of the rollers A A. These bolts pass through holes in the gudgeons of the rollers A A.

D D are small galvanised-iron rollers, and their support causes the belt to form the surface of the evenly-inclined-plane table. This moving and shaking table has a frame of timber, F, bolted together, having the two rollers A and A at its extremities, and the whole of the frame is securely braced with five cross-pieces. The bolts holding the frame pass through the sides close to the cross-pieces, and these cross-pieces are parallel with the rollers A A and D D, &c., their position being understood by the three flat spring connections R O, &c., which are bolted to three of them, one to each underneath the frame, as shown in section A B.

The belt E is an endless one, made of indiarubber, 4ft. wide and 27½ft. in length, having raised flanges projecting above the surface of the belt for about 1in.

G G is the stationary frame, which is bound together by three cross-timbers extending on one side to support the crank-shaft H. This stationary frame supports the whole of the machine, and the grade or inclination of the table is given by elevating or depressing the lower end of G G. This is accomplished by means of wedges; for this frame rests on uprights, Nos. 3 3, fastened to two sills, which form the foundation of the machines in the mill.

The frame F is supported on G G by four uprights, N, on each side. These uprights are of flat wrought-iron, with cast-iron bearings above and below. Each middle bearing on the frame F has one bolt-hole, and there are two of them on each side. The end ones have two bolt-holes, and there are four of them—namely, two on each side. These bolts pass through the frame F, and also hold to the frame the bearings of the rollers A A, which work in a slot. The bearings of the upper or head roller A are higher than those of the foot-roller A; for instance, the roller A is a trifle higher than the regular plane of the table, and the first small roller D should be raised a little.

The shape of the lower or bottom bearings of the uprights N can be understood by examining *b*, as shown in the end-elevation and partly in the elevation. The lower bearing *b* extends across G underneath, and is supported by a bolt passing through G. A lug on the upper side and on the outside end of *b* rests on G, and *b* hangs on the head of a bolt, and is kept stationary by the weight of N and its load. By striking with a hammer the face of *b*, shown in the elevation, *b* is moved, changing the position of the lower bearing, and thus making N more or less vertical. By thus moving the lower supports of N the sand-corners on the belt, hereafter explained, are regulated.

The cross-timbers binding together G G and resting on them are extended on one side as previously mentioned to support the bearings of the main or crank shaft H. The cranks of this shaft are ½in. out of centre, thus making the throw 1in.

I is the driving-pulley, that forms with its belt the entire connection with the power. J is a cone-pulley on the crank-shaft H. By shifting the small leather belt connecting J and W the uphill travel of the main belt E is increased or diminished at will. The small belt connects to J the flanged pulley W, which is on the small shaft K, and by means of the hand-wheel can be shifted on the shaft K and held in place. Y is a cast-iron shell protecting the worm Z and the worm-gear L, and supporting the bearings of the shaft K. This cast-iron shell turns on a bearing bolted to the outside of the stationary frame, and thus becomes a fulcrum for W and K. The object gained by this is that the weight of the shaft and pulley K and W hangs on the small leather belt, preventing slipping or wear.

There is a cam, *a*, used to relieve the small belt from the weight of the shaft and pulley K and W, taking all the strain off the small belt, and thus stopping the uphill travel when desired.

M is a hand-screw, by means of which the worm-shaft K can be moved, adjusting the small belt on the cone J, thus regulating the uphill travel.

K is the worm-shaft, and terminates in the worm Z, which connects with the worm-gear L. The short shaft which L revolves terminates in an arm, S, which drives a flat steel spring, M, which is a section of a circle, connected with the gudgeon of the roller A. N, &c., are the upright supports of the shaking-table F, carrying the belt E.

R, &c., are three flat steel spring connections bolted underneath the cross-pieces of the shaking-table, and attached to the cranks of the shaft H by brass boxes O, &c., on which are cups for lubricating-compound. These springs give the quick shaking motions—about two hundred a minute.

No. 1 is the clear-water distributor, and is a wooden trough, which is supplied with water by a perforated pipe, and the water discharges on the belt in drops by grooves 3in. apart.

No. 2 is the ore-spreader, which moves with the frame F, and delivers the ore and water evenly on the belt.

W is a copper well that fits in and shakes with the ore-spreader at the place shown in the drawing. This is used in concentrating gold-ores for saving amalgam and quicksilver escaping from the silvered plates above, and can be taken out and emptied at any time. Into this well falls all the pulp from the battery. Its ends are lower than the wooden blocks of the spreader, so that the pulp passes over the ends of the well, and is evenly distributed.

For some gold-ores it is desirable to use on the ore-spreader a silvered copper plate the size of the spreader; and when this is used the wooden blocks of the spreader are fastened to a movable frame on the top, so that they can be renewed when the plate is cleaned up—once or twice a month.

Nos. 5 5 are cocks to regulate the water from the pipes Nos. 6 6.

No. 8 is a section of the launder to carry off the tailings; and No. 9 is a box into which the concentrations fall when scraped out of No. 4, the latter being the concentration-box, in which the water is kept at the right height to wash the surface of the belt as it passes through. The overflow from the concentration contains finely-divided sulphurets in suspension. This is passed through boxes shown, Nos. 7, 7, 7, in order to allow the sulphurets an opportunity of settling.

Method of Working.

The ore is fed with water on the belt E by means of the spreader No. 1. Thus the feed is spread uniformly across the belt. A small amount of clear water is distributed by No. 2, which is a wooden trough in which is a perforated pipe, No. 6.

Both No. 6 and No. 2 can be supported from the upper cross-timber of G G.

A depth of ⅜in. to ½in. of sand and water is constantly kept on the table.

The main shaft H should be given the proper speed for each kind of ore. We seldom find the ore in one district just the same as in other places; but when the machine is adjusted to the ore, and the best speed is established, this motion should be kept uniform. The best motion will probably be found between 180 and 200 revolutions of the crank-shaft per minute, with 1in. throw.

The uphill travel or progressive motion varies from 3ft. to 12ft. a minute, according to the ore, and the grade or inclination of the table is from 3in. to 6in. in 12ft., varying with the ore. The inclination can be changed at will by wedges at the foot of the machine, these wedges being under the lower end of G G, and resting on shoulders of uprights from the main timber of the mill.

The motion, the water used, the grade, and the uphill travel should be regulated for every ore individually, but once established no further trouble will be experienced in the manipulation.

In treating ore directly from the stamp too much water may possibly be used by the stamps for proper treatment of the sand by the machine. In such a case there should be a box between the stamps and the concentrator, from which the sand with the proper amount of water can be drawn from the bottom, and the superfluous water will pass away from the top of the box; but as mineral will also pass away with this water there should be settling-tanks for this water, and the settlings can be worked from time to time as they accumulate.

The main body of the belt suffers hardly any wear at all, since it merely moves its own weight slowly around the freely revolving rollers, and the life of the belt is lengthened by this precaution—viz., to keep it clean from sand at every point except the working-surface; thus sand cannot come between the belt and the various rollers. All the bearings should also be kept clean: a machine cleanly worked gives better results and less wear and tear, and requires less power to drive than a machine allowed to be covered with dirt. With a clean machine the wear of bearings is very slight.

The concentration-box, No. 4, which is kept full of water, and through which E passes, may be of any size or depth desired. Though not indispensable, it is best to have a few jets of water playing above and underneath on the belt as it emerges from the water in No. 4, so as to wash back any fine material adhering to the belt; and, as such a method will cause an overflow in No. 4, the waste water, being full of finely-divided mineral, should be settled carefully in the boxes, Nos. 7, 7, 7. Every few hours the concentrations may be scraped out with a hoe into the box No. 9, and if this box be on wheels it can be readily run on a track to the place where the concentrations are stored. Such a method seems clumsy, but there is comparatively a small quantity to handle.

Rules for Proper Consistency of Pulp.

The proper quantity of water used with the pulp from the stamps is very important, and this should be carefully regulated. There should be formed on each side of the belt a slight corner of sand—*i.e.*, there should be on each side sand with less water in it than there is in the balance of the pulp on the belt. If there is not a slight sand-corner the corner will be sloppy, and there will be a loss. Sloppy corners are caused by using too much water with the pulp from the stamps passing on No. 1. Frequently, on the other hand, there may not be enough water with the pulp from the stamps, and the result will be too-heavy sand-corners. The remedy for this is to use more water in the pulp coming on No. 1.

As regards the proper amount of water to be used in the water-spreader No. 2, use just enough—no more—to keep covered the field between No. 1 and No. 2, so that no points or fingers of sand shall show on the surface. The whole width of the belt between the water-spreader and the ore-spreader should be kept quite wet. If dry streaks or points occur, and water, as a consequence, runs in streaks at the junction of the wet and dry channels, mineral will be picked up and "floated" away on the surface of the water. This "floating" of mineral is caused by its dryness, not by its lightness: it has been coated with a film of air.

The proper amount of water with the pulp on No. 1, and the proper amount of water in No. 2, being fixed, the carrying-over of the clean concentrations past the jets of No. 2 should be accomplished and regulated by the uphill travel.

Frequently the sand and water on the belt will be distributed unevenly, the sand working to one side of the belt and making a heavy, broad corner, while the other is sloppy. To control and remedy this, see first that there is no jar about the machine—that there are no loosely-working parts, that everything is working noiselessly, and that all the parts are in line. If, then, there is not an exact balance of the pulp on the belt the heavy sand-corner forms on one side or the other. To adjust the load and keep the sand evenly distributed on the belt, the lower bearings, *b*, of all the uprights, N, on one side of the machine are moved forwards or backwards by slight blows of the hammer. The change of position from the vertical of N, &c., thus occasioned, affects the pulp on the belt; and by changing the position of *b*, &c., on one side or the other, the right balance or equilibrium will be obtained, and the sand and water, or pulp, will be uniformly distributed across the belt: *e.g.*, if the heavy sand-corner is on the shaft side move the bottom bearings, *b*, &c., on the opposite side, out. Again, the sand-corner can be partly controlled by bending the end of the driving spring that is fastened in the collar towards the side having thickest sands. The same effect, and even more positive, is produced by moving the crank-shaft, and with it the table, the same way as the end of the driving-spring is bent. The underneath rolls have also some effect on the corners—by swinging one end of each either towards one another or in the opposite direction.

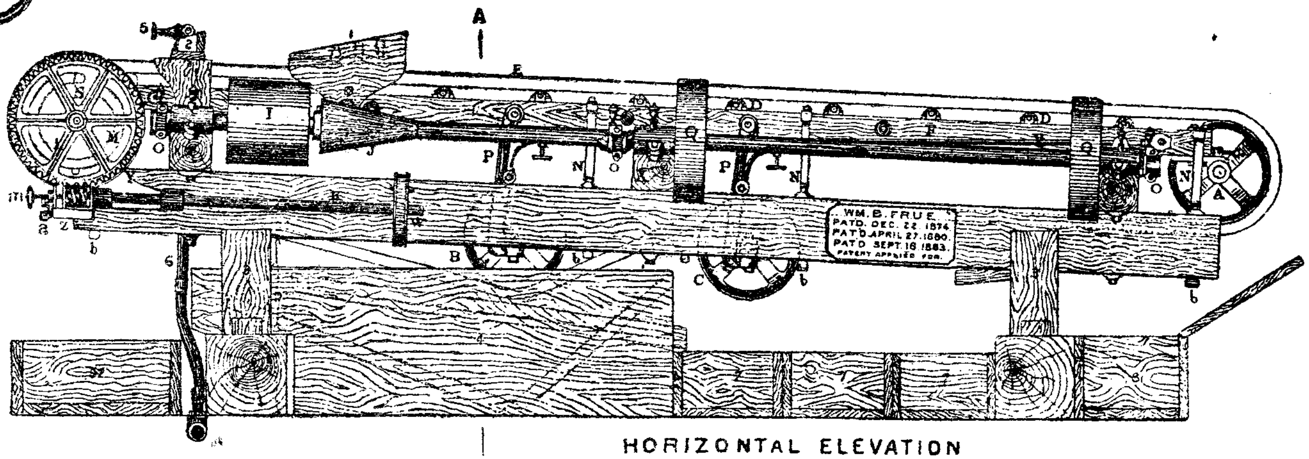
The water in the concentration-box is constantly agitated by the motion of the belt, and consequently the water escaping from this box carries in suspension quite an amount of very finely-divided sulphurets of high assay-value. To save these there should be used settling-boxes Nos. 7, 7, 7, which can be cleaned out once a month, and a product obtained which will add materially to the value obtained from the ore. Two men understanding the machine can put it together in a few hours.

Regarding the bearings of A A, those of the head roller are higher than those of the foot roller. The head roller is a little higher than the regular plane of the table, and it is also advisable to raise the small roller and its bearing next to it by a piece of wood. This additional elevation enables us to use less water at No. 2 than would be otherwise necessary. The lower edge of No. 1 should be within an inch of the surface of the belt E.

Price, Weight, Water, and Capacity.

The price of each machine is £150, landed in Auckland. The machine proper, boxed ready for shipment, weighs 1,800lb., and no part weighs over 160lb.

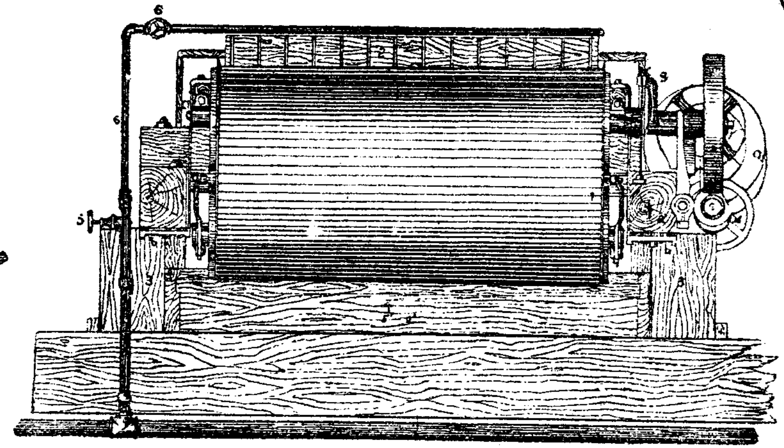
For one machine water is required to the extent of six and one-eighth gallons a minute, including the water used in crushing. This is as large an amount as is ever needed on any material, while on some ores three gallons will answer; and by returning the water from the settled tailings, one half-gallon will keep up the loss.



HORIZONTAL ELEVATION

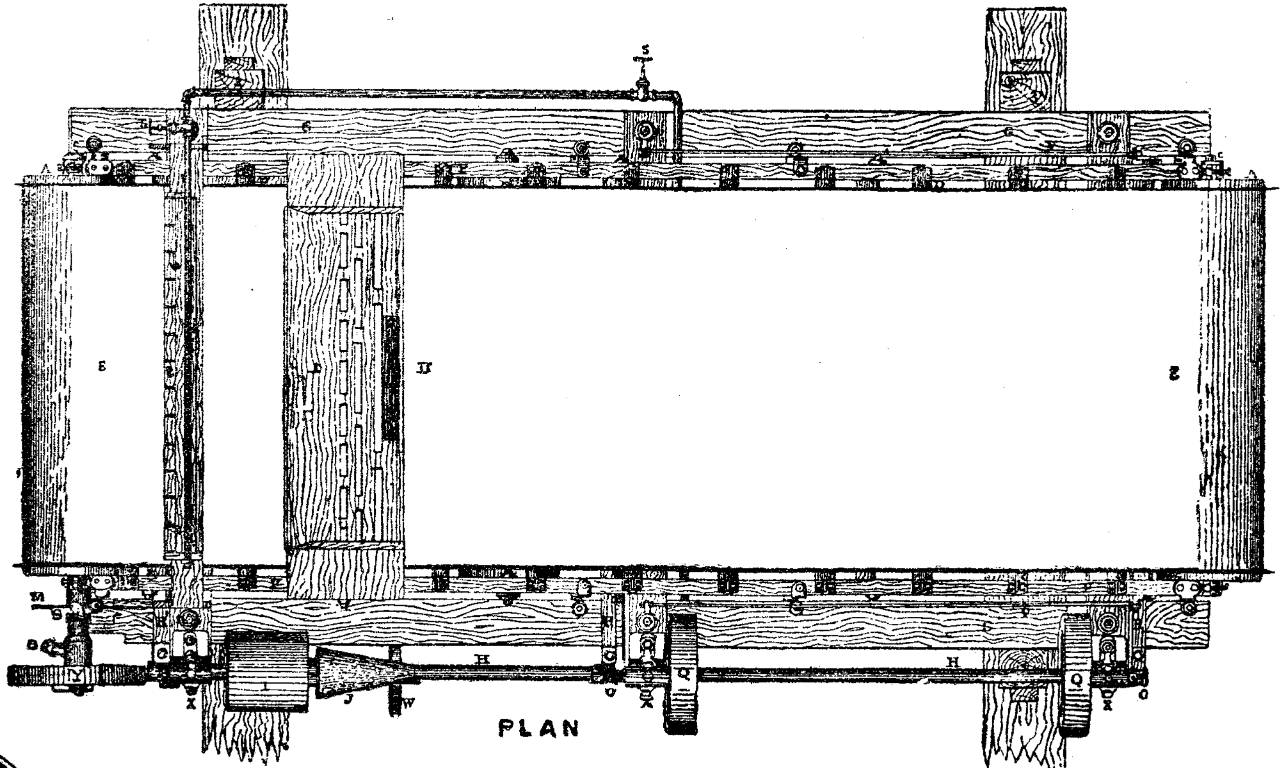
Fig 11.

SCALE

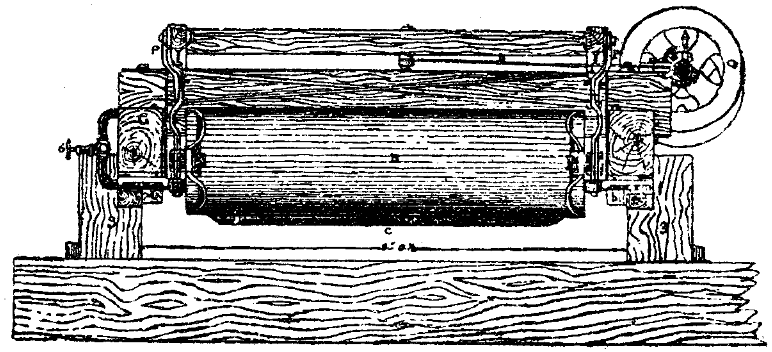


END ELEVATION

Fig 12.



PLAN



SECTION A.B

Fig 13.

FRUE
Ore Concentrator.

THE FRUE ORE CONCENTRATOR.

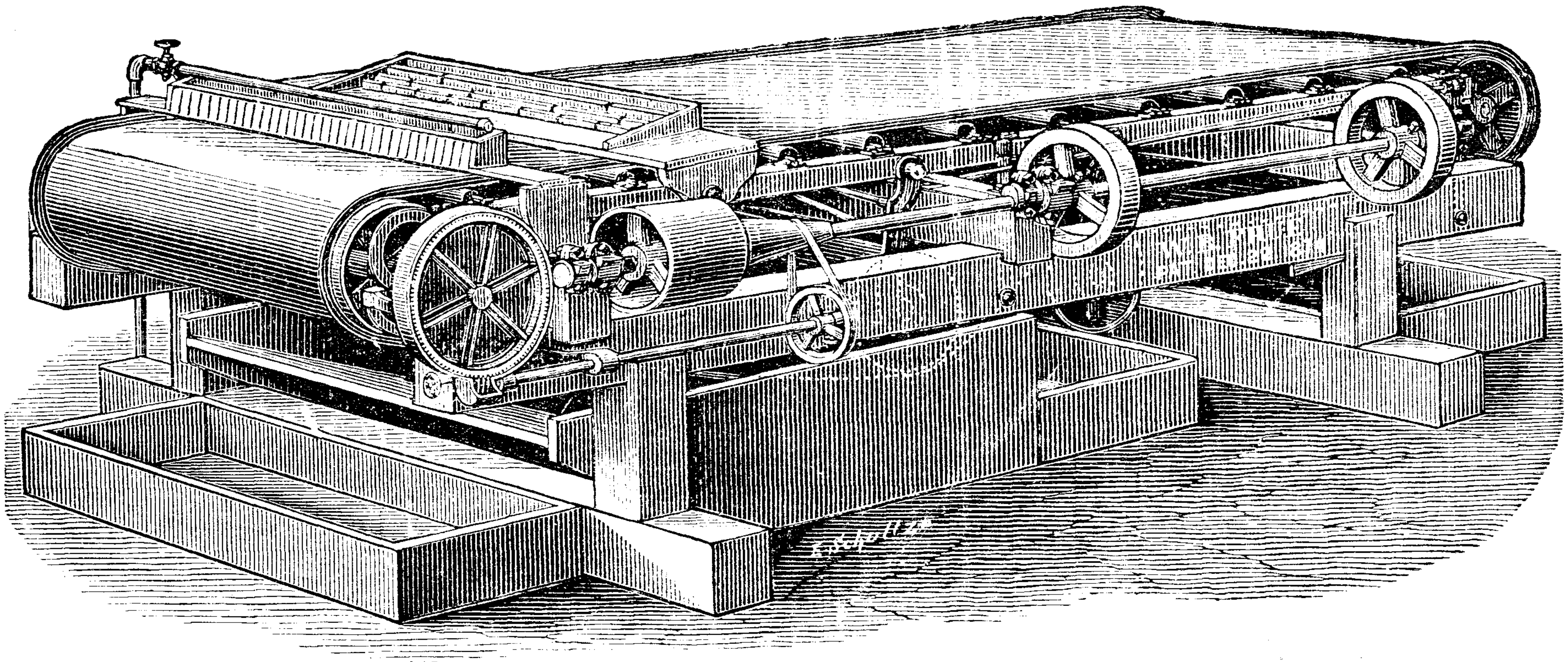


Fig 11.

As regards the capacity of the Frue ore-concentrator, late practice demonstrates that about six tons per twenty-four hours, passing about a forty-mesh screen, is as much as it is advisable to treat. If a battery of five stamps does its duty, the quantity crushed is largely in excess of 6 tons; for this reason the best practice is to put two Frue vanners to five stamps, if the stamps are heavy and the sulphurets are high-grade and difficult to save. It is sometimes urged that this is too large a number of machines for mills to adopt; but where close saving is an object the machines so managed will pay for themselves in a few weeks' run, over the saving of any other. Where pulp from five stamps is fed to two machines the pulp is divided, one-half passing on each. The machines are generally placed in a double row on the same level, head to head, so that the attendant overlooks both rows in walking between them. The concentrator-floor should be so far below the level of the battery as to allow the feed-launder to be above the head of the attendant.

In many cases three Frue vanners to ten stamps will yield entirely satisfactory work, and where the gangue is light, or the stamps not heavy, one Frue concentrator treats all the ore crushed with five stamps, and does perfect work—*e.g.*, in the Empire Mill, of eighty stamps, the property of the Plymouth Consolidated Gold-mining Company, of Amador County, California, sixteen Frues are concentrating all the ore crushed by the eighty stamps. They are doing perfect work—the tailings assay merely a trace. In the forty-stamp mill of the Melones Consolidated Gold-mining Company, in Calaveras County, California, eight Frue vanners are treating all the ore. The tailings assay nothing. The entire cost of mining and milling is less than 4s. 2d. per ton. No sizing of the material is needed; the pulp passes directly from the stamps on the copper plates (if used), and thence on the vanners.

In conclusion, the most important general direction is to keep the machines clean. No splashing of sand over the sides of the belt must be permitted; and all timber-work, as well as metal-work, should always be bright and clean. There is no reason why a Frue concentrator should not be kept as clean and in as good condition of wear as a well-run engine; and in good mills this is done. It can be accepted as an axiom that a dirty machine is not doing its duty. There have been sold in the Eastern States and territories of the United States and in Canada over five hundred machines; on the Pacific Coast nearly seven hundred machines have been sold: and in all cases they have given satisfaction. Printed instructions for the erection of the concentrators are furnished by the manufacturers.

Additional Remarks and Explanations.

The belt forms the bed or plane on which the dressing of the ore is effected, being an inclined plane 12ft. long, and bounded down the two sides by projecting rubber flanges, which prevent the water and sand from dropping over the sides. The arrangement of rollers permits of the belt being slowly revolved in the direction of its length and up the incline; thus, though the dimensions of the working-plane remain always the same, its surface is constantly travelling. The crushed rock in a small stream of water falls near the upper end of the belt by means of the sand-distributor No. 1, and flows down the belt towards its lower end. Now, as the inclination at which the belt is set is only from 3in. to 6in. on the 12ft., and as the stream of water is not large and spreads also over the whole width of 4ft., it is obvious that much of the crushed rock contained in the water would settle on the belt, while the water and the finer and lighter particles of sand would alone reach the foot of the table and drop over into a waste-launder. In addition, we have the travelling of the belt upwards and onwards continuously. The effect of this would naturally be to deliver all the rock which was settled on the belt over the upper-end roller A, and deposit the same eventually in the water-tank No. 4 below, through which the belt passes in plying around the roller B. The action of the belt, then, simply amounts to this: that it forms an inclined plane or working-surface, which by its progressive motion will deposit in the tank No. 4 all solid material which settles on it—*i.e.*, whatever is not carried off in suspension by the water flowing from No. 1: the belt is merely a self-discharging bed. To separate the heavier metallic minerals from the accompanying gangue or rock, it is evident that the above-described action of the belt is not sufficient, for not only would the mineral be delivered in the tank below, but also a large proportion of the rock, which would certainly settle on the belt as well. A separation of the two classes has yet to be accomplished. For this purpose a second stream of water is employed. About 1ft. above the sand-distributor—that is, just below the first small roller D—the water-distributor is arranged, which delivers small jets of water 3in. apart over the entire width of the belt. The revolving belt, carrying its load of settled rock and mineral, travels past the jets of water, taking with it such particles of minerals as have weight or specific gravity sufficient to resist the force of the descending water, while the lighter particles of rock are driven back by the water and do not reach the tank No. 4. A belt working in this manner has been in use for many years as a dressing-machine for slimes; in England it is known as Brunton's belt. As then used, however, the machine is yet imperfect; a considerable inclination has been given to the upper surface, very little slime fed on at a time, and a pretty large stream of water; and, after all, unless carefully watched and occasionally broomed over, the slime-sand will pack on to the belt and the water cut channels for itself, down which it will flow rapidly, carrying off the sand and mineral together.

In the Frue concentrator a new principle is introduced: it is that of a gentle shaking motion, either lateral, end-way, rotary, or in any other manner. By the introduction of this secondary motion the sand is kept in gentle agitation, uniformly distributed over the whole width of the belt, and the heavier particles of mineral settling through the sand cling to the belt and are carried up by it past the small jets of water, and deposited in a cleaned state within the tank for collection. Very little water is now needed to effect the separation of rock from mineral. The belt is flatter and the flow of water is proportionately slower, giving every opportunity for the settlement of the minerals before the sand is discharged at the lower end of the waste-launder. The capacity of the machine is very greatly increased, and a class of material can be treated the treatment of which was before impossible or impracticable.

This machine is not adapted to the treatment of very coarse material, nor is it required for such, as the ordinary forms of "jigging" machines in common use are perfect enough for that. In reducing ore, however, to a size convenient for dressing, whether by stamping, crushing, or grinding, a varying proportion of rock and mineral is broken very fine, and, when carried off by water, is technically known as "slimes." Now, "slimes" are always more difficult to treat than the coarser particles, because much of the mineral is in so finely a divided state that it flows off readily in a stream of water and refuses to settle away from the accompanying fine rock. Many forms of washing-apparatus have been devised for the special treatment of slimes. Among the more successful may be mentioned plain and revolving buddles, inclined frames, shaking-tables of various descriptions, plain revolving-belts, blankets, and hide sluices.

In order to understand the conditions under which a machine must operate in order to extract the metalliferous constituents of a "slime" from the accompanying particles of rock or gangue, it will be worth while to examine the process of "vanning" on a shovel or pan, which is the most perfect method of separation we know of, because in it we employ the judgment—a thing which does not enter into the movements of machinery, except in adjusting the same. In vanning comparatively coarse particles of mixed rock and mineral a particular throw is communicated to the shovel or pan, which causes the contents to move forward by a succession of jumps, as it were; the metallic ores, being of a greater specific gravity than the sand, move ahead of it and form a distinct "head" of mineral easily distinguishable from the rock. With "slimes," however, the process of vanning is different: the muddy water is kept in gentle motion for some time by a circular motion of the shovel until the almost impalpable mineral slowly settles to the bottom; the motion is now interrupted for a short time to allow a further settling of the material, when a gentle wave of water is caused to flow repeatedly over it, washing the fine rock across the shovel ahead of the mineral, which withstands better the flow of water. In this second case the mineral cannot be thrown forward, as was first described; it has not weight or "body" enough for that, but is separated from the sand by taking advantage of the greater resistance it offers to a wash of water when once settled on the shovel. Here is a very important point, this clinging property of finely-divided mineral, which can be well illustrated by putting on to a common plate a small quantity of very finely-pulverised galena or other metallic ore. On wetting the ore and shaking it over the plate the clinging power of the fine mineral is shown when we try to wash the plate by a stream of water: as long as it is well covered with water the motion produces no effect to dislodge it. It is to this property of the slime-mineral that much of the great success of the Frue belt must be attributed: the shaking motion settles the mineral from the sand as it flows slowly down the belt, all the material being kept in gentle motion as in the preliminary settling on a shovel. When once the mineral has touched the surface of the belt it clings, and is carried up past the small streams of water at the head of the machine, and is dropped as the belt passes through the water-tank in a reversed position, even as it leaves the plate when reversed.

The shaking-motion communicated to the belt is of the utmost advantage in more ways than the settling of the mineral from the sand; for, by keeping all the material in motion, the belt can be set at a slighter angle, a smaller quantity of water used, and a much greater quantity of material operated on than if a simple belt without shaking-movement were employed. The sand does not "pack," causing the water to cut channels and run off in small streams, but is always uniformly distributed over the whole width of the belt. It is not necessary that the material on which the belt works should consist of the finest slimes; but, of course, as with all dressing machinery, the more uniform the size of the ore is, the better the results obtained.

As regards the ores on which the machine will work, the only point of importance is, that there be a fair difference between the specific gravity of the mineral to be saved and that of the waste matter with it. The following minerals have been worked upon with excellent results: Iron- and copper-pyrites, arsenical iron-pyrites, zinc-blende, galena, tin-stone, cinnabar, native silver, carbonates of lead and copper, and native copper; and, in the case of "tailings" from amalgamating-mills, "floured" quicksilver. "Slimes" flowing from settling-tanks have been experimented on, and made to yield the impalpable mineral which they contained.

The capacity of one machine is from 5 to 8 tons of rock per twenty-four hours. (They have been worked up to 12 tons.) The quantity of rock treated will depend on several circumstances. If the ore be of the very finest slimes, of course not so much of it can be treated as if some of the material be coarser. If a good separation is required the machine should not be crowded. Where the ore is stamped and screened through a screen of fifty holes to lineal inch, from 5 to 6 tons can be well separated; if the ore is a trifle coarser, 6 to 8 tons can be calculated on. For running a single machine, it is estimated that all the power required is only one-third of a horse-power; and one man attends to sixteen machines without difficulty, as the only work necessary is to oil them and keep them clean about the working-parts, regulate the water, and scrape out the concentrated mineral occasionally from the water-tank. As already stated, very little water is used—about half an inch, miners' measure, to each machine—less, probably for the quantity of rock treated than on any other form of washing-appliance. When six machines are used, the estimated cost of treating the sand when it is ready to flow on to the machine is less than 10d. per ton; but this, of course, is dependent upon the conditions under which they work. Of course, in these figures as to cost and ease with which the machines can be managed, it is taken for granted that the work is steady and the conditions are kept uniform, and under these circumstances what has been said in this connection is no mere matter of estimation, but is the result of actual working on a large scale for several years at a time. If the speed at which the machines are run is continually varying, or the quantity of ore delivered at the belt is not regular, it will take a man at each machine continually busy regulating the same to the constantly-changing conditions. This is a matter on which, however, it ought not to be necessary to insist, as any person of practical experience will appreciate at once the difference between fair and unfair conditions of working, and the difference between the cost of an experimental short trial on a single machine, and the results of a steady working of one or more arranged to work automatically.

In running the machine, the point of greatest importance is regularity—regularity in speed, regularity in the delivery of materials on to the belt, and regularity in the supply of clear water. The necessity for this is obvious to any one who thinks of the work to be done by an automatic machine. With hand-labour the judgment of man regulates the means employed in conformity with varying conditions; but in a machine, the object of which is to supersede hand-labour, it becomes obvious that, having once adjusted the movements to effect a certain object under certain conditions, the desired result can only be attained by the maintenance of the necessary conditions. In this concentrator, supposing the inclination of the belt to be fixed for a certain class of material, the regulation of the work to be accomplished is affected by three things—viz., the speed with which the belt revolves, the rapidity of the shake, and the supply of clear water at the head. Having adjusted these three conditions to a given feed delivered on the belt, that feed must remain pretty constant: the result, both in richness of the mineral collected and the poverty of the "tailings" or waste, will be then continuously maintained. It remains now to examine separately the regulating effect of the three conditions mentioned above.

The revolution of the belt is the agency by which the delivery of the clean material is effected. The necessity for a proper travel is perceived if the result of the two extremes be considered. Supposing the belt to remain stationary, no delivery of mineral could possibly take place; while, if a great travel were communicated, everything which falls on the belt from the sand-distributor, No. 1, would be rushed past the clear water at No. 2 and collected in the tank. Between these extremes there is the desired mean, a speed which shall be sufficiently great to deliver continuously all the mineral collected by the belt, but not so fast as to require a flood of water at No. 2 to keep back the sand. If the ore treated be poor in mineral, the upward motion of the belt should not exceed 20in. per minute; if richer, the speed is increased, and the inclination of the belt is also increased.

To examine the influence of the shaking-motion, two extreme cases can also be cited. In the absence of it, with the ordinary supply of material coming on to the belt, no separation can be effected by a reasonable stream of water at No. 2; the greater part of the rock passes over into a tank with the mineral; it "packs" upon the belt. To drive the crank-shaft H at a furious rate, and thus violently agitate the belt and its load, has the effect of working everything off the foot of the table. In this matter, as with the revolution of the belt, there is clearly a desirable mean, a speed at which the material on the belt is kept in gentle motion, lightly suspended in the water, and thus easily carried by it down the belt—a speed which allows and facilitates the settlement of the mineral from the rock, and disturbs it not when once settled on the belt. The customary rate of driving this shaking-motion varies from one hundred and eighty to two hundred revolutions of the shaft H per minute; the former speed being for fine, light "slimes," the latter for rough and heavy sand.

As regards the regulation of the water delivered at No. 1, keep the field between No. 1 and No. 2 nicely covered with water, and bring the mineral through by regulating the uphill travel. To make the final separation of mineral from sand some little judgment is necessary. As already stated, the delivery-holes in the water-launder are 3in. apart across the whole width of the belt. The clear mineral creeps up between these small jets of water, so that, as delivered over the head of the belt at A, the form is that of longitudinal streaks, further or nearer apart, and of greater or less width, according as the richness of the material treated is different. The primary object to be attained in the adjustment of the uphill travel is that the clean mineral shall be allowed to pass over into the tank at the same rate as it is fed on to the belt in the mixture to be separated. For example, suppose that in every hour 800lb. of mixed rock and mineral passes on to the belt, and that the mixture contains 5 per cent. of heavy mineral—say galena. Now, disregarding the small loss of mineral in the proper waste or tailings, the uphill travel must be so regulated that there is a steady delivery of mineral at the rate of 40lb. per hour. No more than this can possibly be delivered unless rocky impurities are allowed to pass and be weighed in; and if less than this pass there must be a continual accumulation of mineral on the belt, which will eventually produce loss in the waste. This may perhaps seem a rather delicate point to hit, and appear difficult to execute, but in reality it is a very simple matter, on which the eye furnishes a sure guide. The gauge by which this adjustment is rendered easy is the extent of "head" of mineral showing at the point No. 2, where the water strikes the belt. Again, the weight of mineral as it gets strong and heavy forces it more past the water. Should the discharge of mineral exceed the quantity falling on the belt, sand or rock will be found close up to the jets of water, and by-and-by passing them in place of mineral. If the uphill travel be too slow the mineral collects below No. 2, forming a great "head," extending towards No. 1, and even below, in which latter case an increased loss of mineral will assuredly take place in the waste. When working properly a small head is always kept below the jets of clear water, and the mineral comes over clean and regularly. A few hours' experience will instruct any one sufficiently on this fact; and, having once adjusted the uphill travel, the machine will work continuously and uniformly as long as the conditions are kept constant; nothing more than this can be expected of any machine.

In starting the machine the driving-belt is slipped from the loose to the tight pulley on the counter-shaft. The whole frame F, with the end rollers A A and B and C, and the belt E, immediately commence a gentle oscillation. It is hardly necessary to say that previous to starting all the working-parts must be oiled. At first the machine may naturally work somewhat stiffly, but after a few hours' running every part will be found to move remarkably smoothly and easily. The machine should work almost noiselessly: if there be any jar or knock the cause must be found and a remedy applied. These jars can be easily remedied, and are not faults in the machine, but in the setting-up and adjustment. If the shaking-motion be found to work smoothly and without a jar the uphill travel or progressive motion can be given. The machine is now at work, and some clear water run on to it from No. 2 will show whether the belt is level across: if not it is easily levelled by the wedges at the foot. Now the ore-feed is started.

Supposing all instructions to be followed the machine will be working now regularly and smoothly. The water and sand flow down the belt uniformly over the whole width of the belt; the lighter sand, kept gently moving, floats along towards the lower end of the belt; the heavy mineral settles on the surface of the belt, and, having once touched the latter, clings with a force not easily overcome. The belt, moving always onwards, brings all mineral up to the clear water No. 2, and here the difference between rock and mineral becomes apparent. The clean mineral passes between the jets of water, and is deposited in the tank below: the sand works gradually down, to be replaced by other particles.

In treating slimes—as, indeed, with all other qualities of material—as little water as practicable should be fed on with it. A large volume of water on a plane inclined surface implies speed and force—two undesirable elements in the separation of fine mineral. For this reason slimes cannot be treated as fast as a rougher quality of sand, since with a given volume of water a greater weight of material can be carried on to the belt when rough than when in the form of slime. From an extended experience with the machine it has been found that with a slightly-increased speed of the shaking-motion any rough particles of rock are much more easily moved than fine mineral; that it is easy to work the coarse sand off the belt, and at the same time produce extremely slight loss even of the very finest mineral. This observation led to the working of mixtures of sizes which should properly, on the usually-accepted theory, have been classified and treated separately. When a mixture of rough and fine—as, for instance, the discharge from a stamp-battery—is fed on the machines, much more can be treated in a given time on a given number of machines. For instance, in treating slimes from 3 to 4 tons in twenty-four hours is about as much as can be fed, while with rough and fine 6 tons can be treated in the same time on one machine, and both coarse and the very finest material saved together. What is meant by comparatively coarse rock is, say, all that will pass a screen of twenty holes to lineal inch. It is preferred to use a screen of forty holes to lineal inch, and with this extremely good results are always obtained. Even with forty holes to lineal inch there is, of course, great difference between the largest particles of rock and the finest "slime" also present; but the side-motion works off the rock and never moves the very finest mineral when it has once touched the belt.

It is not good to make the pulp flowing on the belt too thick, or the particles of mineral cannot settle through it. For this reason a pretty fair current of water must be allowed to go on with the slimes, and the belt placed with very slight inclination, so that the current of water will not be too rapid.

Occasionally, and at intervals varying with the quantity of mineral in the ore treated, the collections are scraped from the box No. 4, in order to prevent the accumulation in such quantities as, by forming a mound, may come in contact with the belt, and, by the rubbing, wear it. This cleaning of the box is accomplished without stopping the machine, as before mentioned.

In most of the mills already erected the material flows directly from the stamps over the machines, and then flows out of the mill as waste, too poor to rehandle; so that from the time of entering the stamps the rock is never handled in any way.

The tailings from the Frue concentrators of some silver-ores contain a varying amount of silver present as chloride; but the base minerals, and the sulphurets containing the gold and silver, have been thoroughly separated and saved in the concentrations, and whatever chloride is present can be saved at a small expense by leaching or by amalgamation without roasting.

To recapitulate, in conclusion, the chief points to be observed in working: The speed of the belt, being once adjusted in conformity with the inclination and the material worked upon, must be kept absolutely regular. The supply of ore must be steadily continuous, and the water flowing with it as small in quantity as possible. Clear water at No. 2 must equally be kept constant, and arranged both in quantity and form of distribution to allow of the necessary discharge of mineral. No jar or blow must be permitted in working, but the whole machine should work smoothly and almost noiselessly. With the proper attention to these details no trouble at all will be experienced in obtaining uniformly good results from working; the concentrated ore will always be clean, and the tailings poor. One man can attend to sixteen of these machines without difficulty, the only work being to occasionally scrape the collected ore from the tanks and to watch the machines generally, that nothing interrupt the feed or water-supply, and that the bearings contain oil. A superintendent is of course necessary to insure attention.

The Triumph Ore-concentrator.

This concentrator is similar to the Frue vanner (see Fig. 14), the difference being that the shaking-motion is lateral instead of crosswise, as in the case of the Frue vanner. It is highly spoken of as a concentrator, and its price is less than the Frue, being only about £100 delivered in Auckland. They are manufactured by Joshua Hendy, Nos. 39 and 51, Fremont Street, San Francisco. This concentrator is greatly in use in America, and a plant consisting of these is now in course of erection at the Globe Company's mill at Reefton, which will be the first in New Zealand. The following is the manufacturer's description of them: "Triumph ore-concentrators possess many advantages over any other style of vanners, vanning-machines, or concentrators yet introduced to the notice of mining-men. These advantages consist in the superior features which enter into their construction and facilitate their operation. They are constructed in the best manner: their frames being of iron insures their solidity, durability, and perfect steadiness of motion when operated. They are built as compactly as their requisite strength will permit, weigh less, require less freight-space in boxes, by which their cost of transportation is reduced, and occupy less mill-room when set up. The endless belt is carried upon a supplementary frame, which is mounted upon springs. The reciprocating movement which is imparted to this supplementary frame and belt tends to settle

"TRIUMPH" ORE CONCENTRATORS.

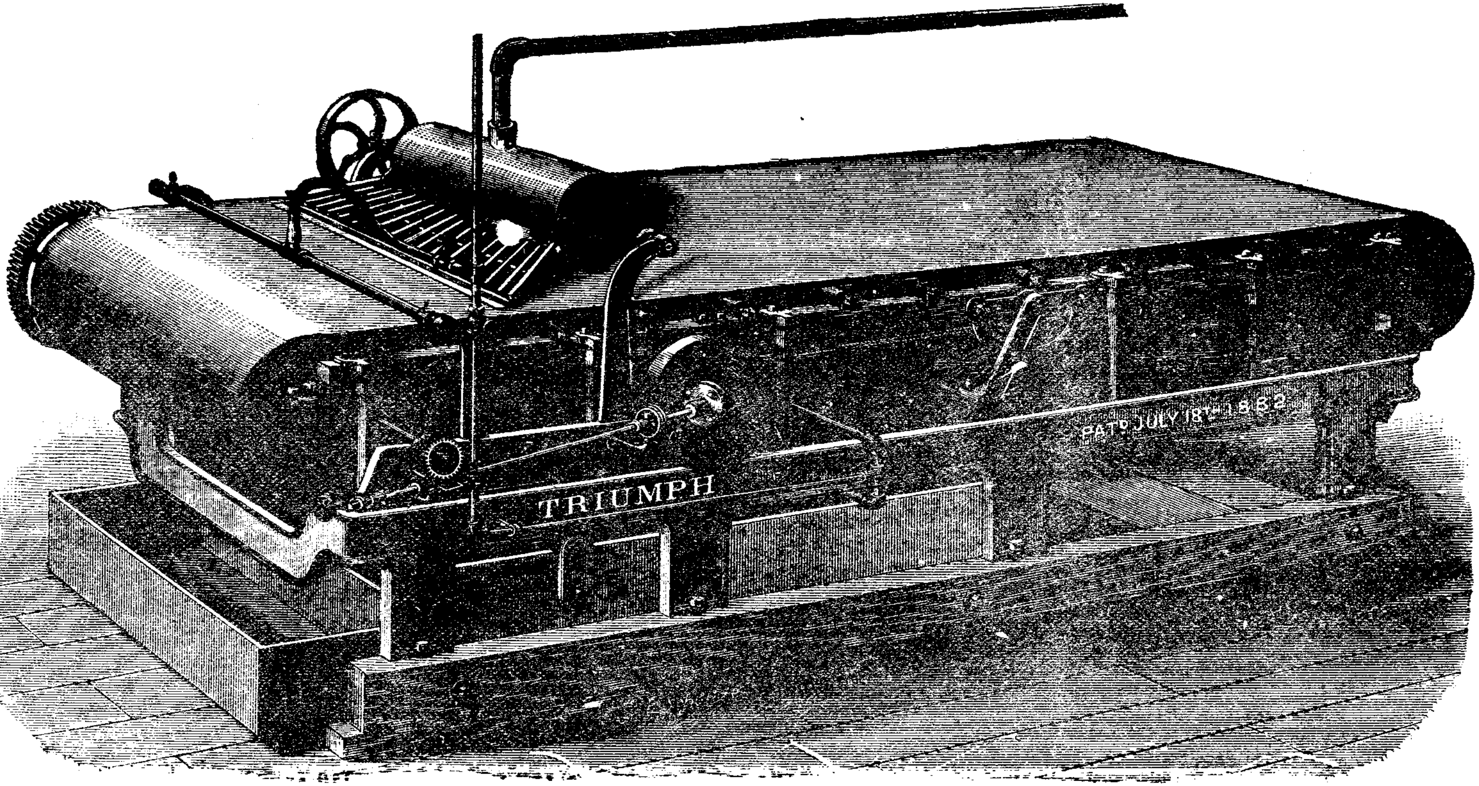


Fig 14.

and retain the sulphurets and heavy and valuable metallic particles upon the belt, until they are discharged at the proper moment. This peculiar movement is of the utmost importance, and enables these machines to perform more work than any other vanners or concentrators yet devised. The rolls supporting the belt are of galvanised metal, which will neither warp, crack, nor rust. The feeding mechanism is perfect, and permits the travel of the belt to be varied at will to any desired speed. The belts are of an improved form, and manufactured especially for this purpose of rubber, and very durable. The arrangements for saving waste amalgam escaping from the batteries are as complete as possible; and all of the parts of these machines are simple and effective in their operation. An important improvement has recently been introduced into their construction, which consists of a riffle-table, placed in front of the feeding-hopper which takes the discharge from the feed and amalgam bowl. This improvement is in the reciprocal motion which is imparted to the table by the longitudinal motion of the shaking-frame to which the table is attached. A further improvement has also been introduced in the feed and amalgam bowl by which the pulp discharged from the batteries, through the conveying-pipe into the bowl, is evenly distributed to its centre and ends, and thereby a more perfect separation is had by the stirring action of the fingers, and a more even discharge of the pulp on the riffle-table is accomplished."

Three of these concentrators are sufficient for the delivery of pulp from, and to thoroughly concentrate the sulphurets from, two batteries of five stamps each—say ten stamps—when the percentage of sulphurets and metallic particles does not exceed 3 per cent. of gangue-matter; six for a twenty-head and twelve for a forty-head stamp gold-quartz mill. A larger percentage of sulphurets and metal particles will of course require a larger number of concentrators to insure perfect concentration. Weight of machine, boxed, 2,270lb. (including the weight of belt, which is 220lb.); weight of heaviest part of machine, 80lb.

For uniform and close concentration, the speed of the driving-pulley of each machine should be adjusted and maintained at 230 revolutions per minute, or as near as possible.

The size of the driving-pulley on concentrators is 9¾in. in diameter and 3in. face—tight and loose.

The power for driving each machine has been carefully determined by an indicator to be less than one-half of one-horse power.

Directions for setting up and operating the Triumph Ore-concentrators.

The concentrating-room should be sufficiently large to allow a space of 20ft. long by 10ft. wide for each concentrator. They can be set in pairs, parallel with each other; but it is deemed preferable to set them head to head in a direct line in front of each battery; the head of the first being set away from the battery, and a passage-way between its head and the head of the second. In the event of the use of three concentrators, then two should be placed with their heads away from and in front of the batteries, and the third be set in the opposite direction, with its head towards the heads of the other two, and a passage-way between be provided; and the same rule should be observed in larger mills, where a larger number of concentrators are to be used. The feeding-end of the machine is denominated the head. This arrangement will bring the heads of the concentrators in close proximity, be convenient for testing the pulp, for overlooking their operation, and afford the attendant an unobstructed view of the battery. The floor of this room should be laid not less than 9ft. below the point of pulp-discharge from the pulp-distributing box, which will permit head-room under the pulp-conveying pipes. It should be given a descending grade from the centre of the passage-way at the head of the machines towards the battery, and also in the opposite direction away from the battery; troughs or sluices being laid at the lower ends of these floors for carrying off any valuable particles to a convenient receptacle when washing the floor. This arrangement will serve to keep the passage-way dry and clean. The grade given each way should be about 2¼in. to each 12ft.

Timber and Bolts for Foundation-frame.—Two pieces, 8in. by 8in., by 11ft. 3in. long, 120ft. B.M.; three pieces, 8in. by 8in., by 5ft. 2in. long, 83ft. B.M.; three bolts, ⅝in. round by 6ft. 1¼in. long, between heads and nuts.

The foundation-frame should be set on a descending grade from its head of, say, 3¾in. to each 12ft.

The concentration-tank should then be firmly joined together at its ends and sides, using white-lead or paint in all of the joints. Paint the tank inside and outside, and put in its place between the sides of the foundation-frame. The concentrator-frame (iron) can then be bolted to the foundation-frame, and its ends and intermediate lower cross-piece bolted to the side frames. The upright wooden springs are then to be fitted in their sockets on this iron frame, six on each side, being careful that they are level from corner to corner and across their tops, and in a line lengthwise with the frame. The shaking-frame should then be bolted together, and its cross-pieces fitted. Then screw on to this frame the boxes for the small zinc rollers, fifteen on each side, for fifteen rollers. The first or head set of these roller-boxes are ½in., and the second ¼in., higher than the succeeding sets, which are all of an equal height. Before placing the small rollers in their boxes see that each one is straight from end to end. Then place the two 13in. roller-drums in their boxes, the one which is raised towards its centre at the head of the frame, and the one which is straight from end to end at the foot.

Lay the endless rubber concentrator-belt on the floor, alongside of the shaking-frame; then work the belt over the frame and rollers, and, with lifting-pieces, raise the belt, frame, rollers and all, and set carefully on the upright wooden springs already fitted in their sockets as previously instructed.

The tightener-drums, which are 8in. in diameter, are then to be placed in their allotted position, the straight one near the head, and the one with the tapered or rounded ends near the middle. Place the 4in. roller at the head and across the upper side of the inside of the belt, in which position it serves to steady the belt in the reciprocating movement of the shaking-frame. Bolt on the crank-shaft box cross-piece, which has dripping-pans on its ends.

An improvement has recently been introduced into the construction of these machines: this consists of two iron steady-rods, one being bolted on to the wooden shaking-frame at or near the head of the machine; tho other at or near the foot; the lower end of each leading across to and being bolted on to the opposite side pieces of the foundation-frame. The lateral resistance of these rods necessarily effects a steadiness of motion in the shaking-frame, and prevents any side or lateral movement, and is believed to be of great advantage.

Put the main crank-shaft in its boxes, and fit the connecting-rods and eccentric-straps to it. Then fit the cross feed-shaft in its boxes, and see that this shaft and the head and lower drum-shafts are exactly parallel with the main crank-shaft. Then put the side longitudinal feed-shaft and its adjusting feed-gear in place. Then raise the middle tightener, to take up the slack of the belt; some of this slack can be taken up by screwing the ends of the shaking-frame out.

When all the cross-shafts are parallel with each other, level the small zinc rollers, so that they shall have a perfect bearing on both sides of the shaking-frame, and shall all be in line and level from the foot to the second one from the head of the frame, this being ¼in. higher than the others, and the one at the head being ¼in. higher than this, as previously stated.

The water-pipes can then be placed, one under and one over the belt, as shown in cut; their purpose being to wash off any metallic particles which may adhere to the belt in passing through the concentration-tank. The distributing water-pipe in front of amalgam-bowl can also be placed as shown, and all connected with supply-pipe.

The feed and amalgam bowl can then be set in its upright supporting-standards. Pack the small recess at each end of the bowl, to prevent either sand or water from working into the boxes. There are nine stirring- or finger-pieces to be first put on the shaft which runs through this bowl. These are held in position by set-screws.

The driving-pulleys and gears can then be attached.

The riffle-table is then to be placed on its supporting-standards, which are to be bolted to the shaking-frame. This table is an improvement, as will readily be observed from the fact that its motion is reciprocal with the longitudinal motion of the shaking-frame.

The pulp-pipe leading from the distributing pulp-box should be, say, 2in. in diameter, and have a grade towards the feed and amalgam bowl of not less than ⅝in. to each foot of length.

Directions for operating the Triumph Ore-concentrators.

In starting up the feed-gearing, when the concentrator is set up, run part of the shafting at a time until the bearings are worn to a smooth surface, first running the short side shaft, then those which run across the machine; then connect all.

The speed of the driving-pulley should be adjusted and maintained at 230 revolutions per minute.

Quicksilver to the amount of 15lb. or 20lb. is to be placed in the feed and amalgam bowl for catching either free or flour gold or waste amalgam that may escape from the batteries.

The pulp should be kept evenly distributed over the endless rubber belt to a depth of $\frac{5}{16}$in. This is called "the load." It should be of the consistency of paint: if thinner, this indicates the flow of too much water into the battery; if thicker, not sufficient. The consistency of the pulp should be carefully attended to and continuously maintained, and if sand predominates, causing too heavy a load, then either thin down with more water or raise the grade of the machine, or *vice versâ*. The character of the pulp and the percentage of sulphurets or valuable metallic particles governs the grade to be given to the foundation-frame of the machine, which can be regulated by raising or lowering the head or foot of the frame by blocking up or letting down. When the load is as it should be the endless rubber belt should travel over its rollers at the rate of from 3ft. to 4ft. per minute.

The depth of water in the concentration-tank should be kept at such a point that the rubber belt will not be immersed more than ½in. Any overflow from this tank should be run into a separate tight box for the purpose of settling any fine metallic particles that might be carried off.

The quantity of water ordinarily required for the proper concentration of the valuable metallic particles from the pulp of gold-bearing ores is sixty gallons per hour, exclusive of the battery-water delivering the pulp.

Cleanliness is an imperative necessity to accomplish the satisfactory operations of these concentrators. The main and eccentric shaft-boxes must be kept properly tightened, oiled, and constantly clean, in order to prevent any loss of motion, and cleanliness should be maintained about every part of each machine and throughout the concentrating-room.

McNeil's Gold-amalgamator and Ore-concentrator.

This machine consists of a round shallow pan discharging at the centre. (See sketches annexed, Figs. 15 to 21.) Its outside diameter is 6ft., and the flange or rim is 5½in. in depth. There is another rim inside the outer one of the same depth, which forms a circular trough round the outer side of the pan 2¾in. in width. This circular trough is used for collecting the pyrites. The inner rim stands about ½in. above the bottom of the pan, with the exception of its connections with the bottom, which are about 2½in. wide. The inner rim forms portion of the main casting, but there is a slot at the bottom edge leaving solid metal every 15in. for about 2½in. in width.

The bottom of the pan rises on an incline of ¼in. to the foot—that is to say, there is always 1¼in. of depth of material at the outer edge of the pan before it can discharge at the centre. The bottom of the pan is covered with segments of electro-plated copper plate, all fitted closely together, and there are studs ⅜in. in diameter and ½in. in height fixed to the copper plate, about 2in. apart, all over the bottom. These studs are also electro-plated. The pan is held in suspension by four eye-bolts ⅞in. diameter, which go through lugs riveted on to the outside of the pan, and these bolts are brought close together at the upper end at the centre and held by eye-bolts to the framing, which stands about 6ft. above the level of the bottom of the pan.

On the top of the pan there is a convex hood, held in position by four suspension-bolts from the top framing. This hood is 4in. less in diameter than the inside of the pan, thus leaving a 2in. opening between the edge of the hood and the inner rim. On the top of the hood at the centre there is a circular ring 4in. in depth and 16in. in diameter, having slot-openings round the lower side. The material to be treated is fed into this ring, and a small jet of water is turned on, which has the effect of washing the crushed ore through the slot-openings of the ring and over the hood into the pan. This hood is stationary, and not in any way connected with the pan, but merely suspended above it about ½in. above the level of the top side of the rim.

The pan being suspended by the eye-bolts already referred to, there is a vertical sliding bearing fixed on one side of the pan, into which the crank works. The crank having a throw of 2in. gives the pan a horizontal movement or shaking motion of 4in., and makes about 160 shakes per minute. The rapid circular shaking motion thus given has the effect of bringing whatever fine gold there is in the ore in contact with the copper plate and studs, while the iron-pyrites and particles of greater density than the gangue are forced by this motion to the outer side of the pan, while the lighter material is discharged at the centre, and runs to waste.

This machine is highly spoken of as an amalgamator and ore-concentrator, and has been used by the Walhalla Gold-mining Company for several years at Walhalla, in Gippsland, Victoria. The principle of the machine is that of the tin-dish motion, which no doubt will give good results with fine gold, and even when gold is associated with other minerals which will not amalgamate with quicksilver they have a better chance of being collected among the pyrites, and can be taken out and subjected to a different mode of treatment. The cost of one of these machines in Victoria is £100. One is required for every five heads of stamps.

The following are the inventor's remarks and description of this concentrator and amalgamator:—

"This invention relates principally to improvements in those descriptions of machines known as 'Brown and Stansfield's Concentrators,' and the first of which improvements consists in lining the bottom of such pans with electro-plated or silver-coated copper plates, the surfaces of which are either ribbed, barbed, or with similarly-coated upstanding copper studs upon the said copper plates. The second improvement relates to the means of imparting the oscillating motion to the pan by arranging the ball crank-pin at the end of the driving-shaft to work in a sliding-block bearing, which works between guide-bars or brackets, firmly secured or formed upon the pan itself; and the third improvement relates to an alternative means of supporting the pan from an overhead frame by means of a series of hanging-rods, which radiate from one common centre on the frame to the several positions at the side of the pan. To drive a pair of pans I connect them together with a slot eccentric rod, and drive it from a plain crank-pin on the driving-shaft.

"The studs of the first improvement prevent the electro-plated or silver-coated copper plates from being scoured by the agitation of pan and friction of sand or tailings, and likewise form stops to which the amalgam adheres. By treating finely-crushed quartz or fine tailings in these pans, a very large percentage of the fine gold and floured silver is caught, and thus prevented from being led to the waste-heap. Further, under some conditions I find it desirable to introduce a small quantity of sodium amalgam to the pan.

"By working the pan as described for the second improvement, I find it capable of sustaining the excessive shock occasioned by its oscillation; and also the bearings are more readily repaired and adjusted.

"The first improvement may also be applied to other gold-saving machines, such as shaking- or percussion-tables, as, when their working-surfaces are lined with the electro-plated or silver-coated copper plates having the upstanding studs or ribs formed on them, their gold-saving capabilities are greatly enhanced, as the fine gold, floured silver, and fine pyrites are caught by such studs or ribs.

"In order that my invention may be better understood, I will describe it with reference to the accompanying sheet of drawings, which illustrate—Fig. 15, a half-plan of a concentrator-pan mounted upon the frame which supports it and its driving-gear, and showing one-half of the copper-plated surface of the pan with upstanding studs on it, while the other half is formed with radial ribs: Fig. 16, a longitudinal vertical section through the centre of the pan and its driving-gear, the pan being shown with the studs projecting from its copper-plated surface: Fig. 17, a side elevation of the concentrator-pan, showing the means for further supporting it from an overhead frame: Fig. 18, several sections of the copper-plate lining, which may have studs upon it as at *a*, or ribbed as at *b* and *c*: Fig. 19, a side elevation of two connected concentrator-pans, showing the method of imparting motion to them from the one crank-shaft. Fig. 20 is a longitudinal section of an ordinary percussion- or shaking-table, showing the application of the electro-plated or silver-coated copper plates, furnished with upstanding studs to their working-surface. Fig. 21 is a longitudinal section of another form of table, it being constructed of a section similar to that of the pan, with its surface lined with the electro-plated or silver-coated copper plates having upstanding studs on them.

"In Figs. 15 to 18, A is the concentrator-pan, the working-surface of which is lined with the electro-plated or silver-coated copper plates A^1, which are shown with the studs A^2 upon their working-surface. A^3 is its vertical spindle, and A^4 its supporting frame. B are the brackets formed upon or secured to the outside of the pan, having bolted to them the flat bars B^1, between which the sliding-block C works. Such sliding-block is made in halves, bolted together at its end snugs, and bored to fit the ball crank-pin D, which is secured or formed upon the end of the shaft D^1, which is supported in bearings D^2 in the framing, and upon which shaft is fixed the driving-pulleys D^3. In Fig. 19 the concentrator-pans A have snugs formed on their bottoms, in which are secured the ball-pins E, to which the ends of the connecting-rod, E^1, are attached, while motion is imparted to the connecting-rod at its centre, where a slot-eccentric, E^2, is formed for the sliding-block E^3, such slot-eccentric being formed by the two T ends, E^4, of the connecting-rod being bolted together with a distance-ferrule, E^5, between them, the sliding-block E^3 in this case working upon a plain crank-pin on the end of the crank-shaft D^1. E^6 are guide-bearings for the connecting-rods. Fig. 17 shows the stays, F, which are attached to the sides of the pan at F^1, and at their upper ends meet at the one common centre F^2, which is supported in the overhead framing F^3. In Figs. 20 and 21 longitudinal sections of shaking- or percussion-tables G are shown, the working-surfaces of which are lined with the electro-plated or silver-coated copper plates A^1, having the studs A^2 upon their surfaces, similar to that described for the concentrator.

"Having thus fully described and explained the nature of my said invention, and the manner of performing same, I would have it understood that I do not confine myself to the use of such electro-plated or silver-coated copper plates on such concentrating-machines and shaking-tables as I have described only, as such improvement may be used on other gold-saving machines to which an oscillating or percussion movement is given; nor do I wish to confine myself to the nature of the rib or projection upon such plates; but what I do claim as my improvement in concentrator-pans or other gold-saving machines is: (1.) Lining the working-surface of concentrator-pans, shaking-tables, and other similar machines with electro-plated or silver-coated copper plates which have similarly-coated upstanding studs, projections, or ribs formed on them substantially as herein described and explained, and as illustrated in my drawings. (2.) In amalgamators or concentrators forming a bearing on the pan to receive the sliding-block in which the ball crank-pin works substantially as described, and as illustrated in Figs. 15, 16, and 17 of my drawings. (3.) The method of imparting the motion to the connecting-rod of a pair of such pans substantially as described, and as illustrated in Fig. 19 of my drawings. (4.) The method of supporting such pans by swinging-rods from a central pivot supported on an overhead frame, substantially as described, and as illustrated in Fig. 17."

The following is an extract from the quarterly report of the Long Tunnel Gold-mining Company, Walhalla, Victoria, where these machines were first used: "The Treatment of Blanket-sand.—The treatment of tailing blanket-sand has now become an important feature in the company's crushing operations. This material is intercepted and saved by means of blanket-strakes placed outside the battery-house, and contains the finest class of arsenical pyrites, also portion of the fine flour gold and silver, which passes away with the water and tailings from the battery, and passes all the best appliances. This is collected and treated in what is known as Brown and Stansfield's concentrating-pans, and these have been so improved in construction and mode of working at the company's works that I consider them to be the best ore-dressing machines now in use, as they not only save the finest class of pyrites, but all flour-gold contained in the sand treated. The pyrites-sand referred to so saved is of a high assay-value. The result of the working of those pans may prove of some interest. The amount of rough sand treated has been 76 tons 17cwt. 1qr. 22lb.; fine pyrites saved, 22 tons 1cwt. 2qr. 15lb.; amalgam from bottom of pans, 310oz. 7dwt. 12gr., which gave retorted gold 66oz. 5dwt. 8gr., which otherwise would have been lost."

The manager of this company informed me that they were the best combined amalgamators and concentrators that ever he had used. The principle on which they are constructed is correct in theory. The shaking-motion sends all particles of greatest density towards the periphery, while the light sands are discharged at the centre; which is exactly the reverse of Mr. McQueen's amalgamators which he has constructed and is using on the New Era Company's dredge erected on the sea-beach between the Waimangaroa and Ngakawau. The latter has the same shaking-motion, and discharges at the periphery.

Watson and Denny's Grinding and Amalgamating Pans.

These pans are highly spoken of by some of those that are using them as being a very good amalgamating and grinding machine. They are similar in shape to the Wheeler pan, but the grinding-surface is different. The pans are 5ft. 6in. in diameter, and 2ft. 6in. in depth, made of cast-iron. The main bottom of the pan is flat and level near the periphery, but rises a little towards the centre. A false bottom is placed in the pan, consisting of corrugated cast-iron segments placed horizontally on top of the main bottom, having the outer ends resting on supports. These segments have projections at both ends, which fit into places provided for their reception to keep them steady in their places, and prevent them shifting about. There is an elongated wedge-shaped space left between the false and the main bottom, which is partially filled with quicksilver when the pan is at work, and it is here that the process of amalgamation takes place. The upper grinding-segments are fitted on to the carrying-arm on the same principle as that used for carrying the mullers in the Mackay pans, and these segments or shoes have annular corrugations which fit into those in the false bottom or dies, so that when set up there are radial spaces between each piece.

The action of the machine is to grind the pulverised ore coming from the stamping-battery to a fine pulp, and amalgamate the gold and silver it contains at the same time. The only place where there was an opportunity of seeing these pans at work during the time of my visit to the Australian Colonies was at Waukaringa, in South Australia, at the New Alma Company's works. This company has a large lode in their mine of auriferous pyritous ore, varying from 2ft. to 16ft. in width, lying at an inclination of about 1 in 3. The whole of the lode in this mine is so highly charged with pyrites that no free gold is discernible; indeed, a great deal of the ore is almost pure iron-pyrites, containing a large percentage of sulphur and arsenic.

The ore when taken from the mine is first put through the stone-breaker, thence through the stamping-battery, and thence it passes over a riffle-table and goes into the Watson-Denny pans, which are used here for concentrating as well as for grinding and amalgamating, but when they are

J.R. McNEILL'S: PATENT

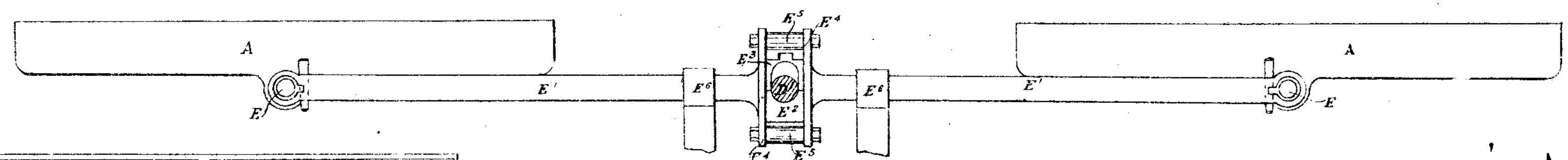


Fig: 19

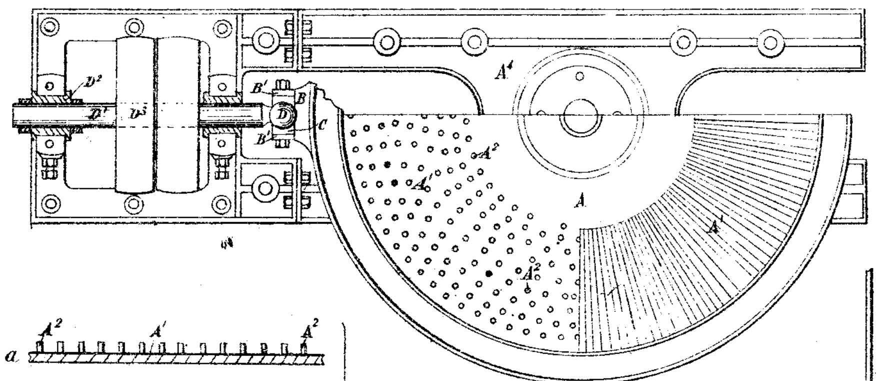


Fig: 15

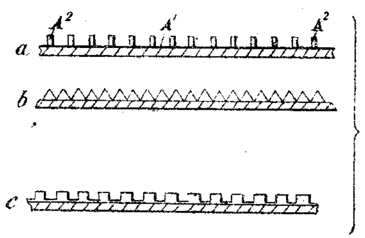


Fig: 18

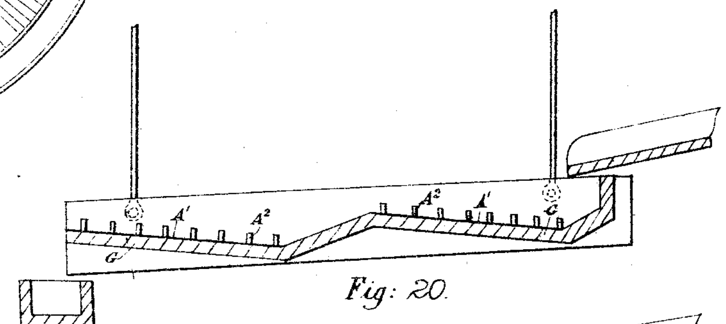


Fig: 20.

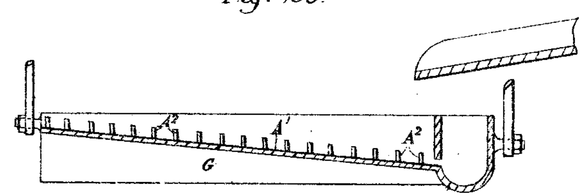


Fig: 21.

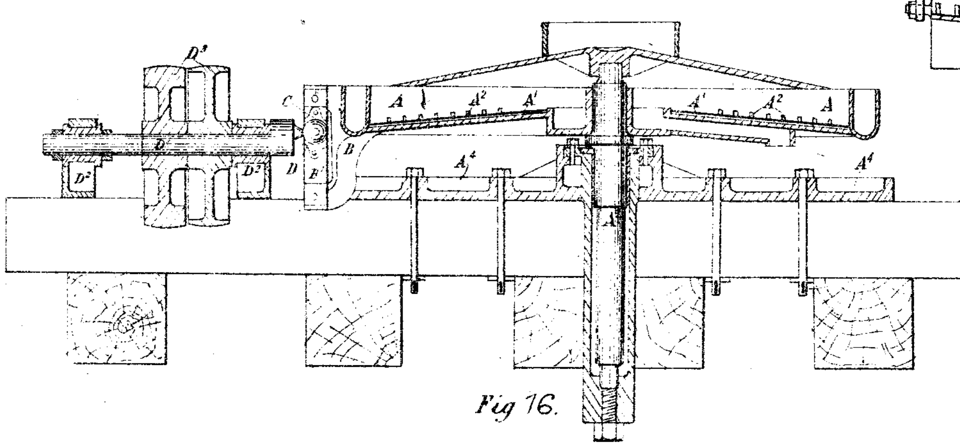


Fig 16.

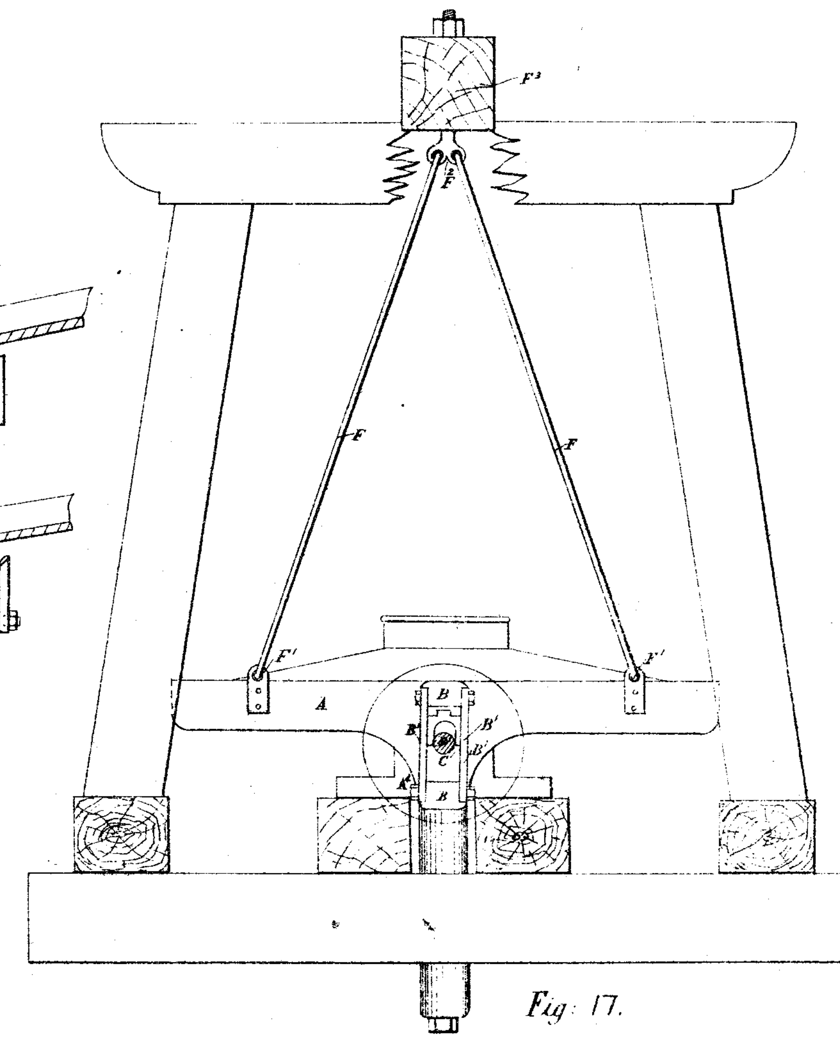


Fig: 17.

used for concentration the shoes are lifted up to a certain height above the bottom. When there is about 6in. of concentrates in the bottom of the pan it is stopped and cleaned out, and the concentrates are then dried and roasted in a reverberatory furnace, after which the roasted material is ground up to a fine pulp in one of these pans, and the gold saved by amalgamation.

The manager of the New Alma Company spoke highly of these pans as grinding and amalgamating machines; but they have not yet been sufficiently tested under different conditions to make an accurate comparison with other and older machines now in use. The price of this pan is £175, and the speed that it has to be driven at is about fifty-five revolutions per minute.

The patentees, however, claim for it the following advantages: That it is able to pulverise the ore into a fine pulp and amalgamate the precious metals at a very low cost, and also capable of treating large quantities. They estimate the cost of working, including wear and tear and all expenses in connection with working, to be about 3s. per ton, and that one pan is sufficient for every five heads of stamps. They also draw attention to the fact that the ore does not require to be so carefully crushed to a fine state by the stamping-battery when it goes through one of these grinding-pans, and that coarser gratings can be with economy used; but the same argument applies to any similar grinding-machine. From my own observations on seeing this pan working, it is an inferior grinding-machine to the Mackay pan, but a better amalgamator.

A classifier is fixed on the outside of the machine, its three sides tapering towards the bottom, where it communicates with the interior of the machine by a hole. In the classifier a deflecting-plate is so arranged that the size of the grains of tailings can be regulated to suit different circumstances. All grains over a certain size are automatically separated and collected in the classifier by this means, and settled by gravitation in comparatively still water, and are sucked through a hole at the bottom into the machine and reground, the waste passing away with the overflow. The total weight of the pan is about three tons. Annexed are sketches of pan in position (see Figs. 22 and 23).

Fixing the Classifier.—This separator is 3ft. 6in. long by 18in. wide. A V table must be fitted to the full width of end of copper table, tapering so as its other end will fit the classifier, making a tight joint. The use of this is to make two classes of stuff; the one class being the worthless soluble mullock, which will flow with the surplus water over the end of the classifier; the other, the valuable material, such as gold, mundic, and heavy sand, being too heavy to be carried away with the water over a surface 3ft. 6in. in length, sinks, and by the shape of the classifier, assisted by the downward current, is carried into the discharge-pipe, which conducts it into the pans. The pipe is supplied with four cast-iron nozzles, having discharges $\frac{1}{2}$in., $\frac{5}{8}$in., $\frac{3}{4}$in., $\frac{7}{8}$in. The operator must determine which to use by the material under treatment. If very mullocky the $\frac{1}{2}$in. would do, the object being to get rid of all the useless mullock, which, if allowed to go into the pan, considerably interferes with amalgamation. The small nozzle is the best to employ so long as the waste flowing over the classifier is found to be of no value. The pan should be fixed so that the top of it is about level with the discharge-pipe of the classifier, thus insuring an easy flow into the pan. The speed of the grinding-muller is fifty-five revolutions per minute, and that of the horizontal spindle on which the pulleys are fitted 110. The speed of cam-shaft being under forty revolutions per minute makes it impossible that the pan can be driven direct off it. The best plan is to drive off an intermediate shaft, which should take motion from the engine-shaft when possible. It is better to get up the speed at once on the intermediate shaft to 110 revolutions; then a 3ft. pulley, 12in. wide, having a 6in. belt, will give correct speed to pan.

The following are the inventor's remarks and description of this grinding and amalgamating pan:—

"This machine is the invention of Mr. Thomas William Watson, of St. Arnaud, and Mr. Thomas Denny, of South Yarra, Victoria, who have had many years' experience in connection with the extraction of gold, silver, and other metals from their ores, and is now the property of the Watson and Denny Gold and Silver Extracting Company (Limited).

"The great advantage of this invention is the extremely low cost of treating large quantities of all classes of auriferous-argentiferous ores. It is a well-known fact that all batteries lose a considerable percentage of gold and silver in the tailings and pyrites, which can nearly all be extracted by this process at a cost of less than 3s. per ton of the ore or quartz crushed. This covers all charges for wear-and-tear and working-expenses, one machine treating and reducing to slime all the material produced by a five-head battery. These machines treat the ores in a raw state, without roasting or other manipulation, no matter what proportion of mundic, pyrites, or other base metals the ores contain.

"The process is more especially intended to extract the gold and silver now being lost in the tailings and pyrites from the various quartz-batteries and other reduction-works. It has been practically proved on a large scale by this process that from 70 to 95 per cent. can be saved of all the gold and silver known by fire-assay to have been lost in tailings and pyrites. These machines, being continuous and automatic, receive all the sand and pyrites either as they flow direct from the battery-boxes or at the end of the copper and blanket tables, and need very little attention, no additional men being required. It not being necessary to crush the ores so fine as is now the custom, much coarser gratings may be used, consequently a larger quantity can be operated on by the battery in a given time.

"In this process, treating free-milling ores, the action of gravity is not depended on to sink the particles of gold and silver to the bottom to be amalgamated, the ore with its metals (no matter how fine and light) being forced by the motion of the produced currents into the quicksilver, which lies underneath, but not in contact with, the grinding-surfaces; therefore the quicksilver cannot become floured or sickened. The particles of gold and silver, being caught in this way, are not further reground. The amalgam is coarse, and yields from 30 to 50 per cent. when retorted, as proved at Granya, Howqua, Malmsbury, Waukaringa, South Australia; Mount Shamrock, Queensland; Swift's Creek, Omeo, and other places.

"Each machine weighs about three tons, is very portable, and is inexpensive to erect. Notwithstanding the great efficiency of this machine, a small proportion of gold and silver is lost, varying with the different classes of ore. In order to save this it is strongly recommended that the Watson and Denny concentrator be afterwards used, thereby obtaining nearly all the valuable metals at an insignificant cost, the concentrates after roasting being re-treated by another grinder and amalgamator similar to those previously mentioned.

"Therefore the complete Watson and Denny process consists of—(1) Grinding the crushed ores and tailings in bulk to a certain degree of fineness, and obtaining therefrom the greater portion of the gold and silver; (2) concentrating the ground ores in the concentrator, then roasting concentrates in portable furnace; (3) roasting and regrinding the concentrates to the finest possible slime, from which the gold and silver is obtained, with the exception of a very small percentage.

"The whole of this process being one continuous operation, and the machines automatic, no handling of the stuff is required from the time it is fed into the battery-boxes until it issues from the last machine as valueless slime, except that necessary for handling the small percentage of concentrates. It will, however, be understood that in most instances the first operation by the grinder and amalgamator suffices to extract nearly all the valuable metals, and therefore the concentrator, furnace, and additional grinder and amalgamator would not be absolutely necessary in all classes of ores.

"The machines are provided with locks and keys, to thoroughly secure the safety of the amalgam until the cleaning-up, which may be every week, fortnight, or month, this operation taking about an hour for each machine. A full description of the method of working these machines will be supplied to purchasers, so that any ordinary battery-man can attend to them."

General Description.—The main body of this machine is a wide circular iron casting about 5½ft. in diameter and 2½ft. deep, the bottom of which is flat near the periphery, but rises slightly towards the centre. A false bottom, consisting of corrugated segments, is placed in a horizontal position above the true bottom, the inner end resting on the upper part of the true bottom, while the outer end rests on supports. These segments have projections at both ends which fit into places provided for their reception to prevent them from shifting. There is an elongated wedge-shaped space left between the lower segments and the true bottom; this is partly filled with quicksilver when working, and it is here that the process of amalgamation takes place. The upper grinding-segments are also provided with annular corrugations that fit into those of the lower segments, so that when set up there are radial spaces between each piece. The upper segments are attached to the carrying-plate, which communicates a rotary motion to them from the driver. Between these grinding-surfaces the ore is ground to an almost impalpable state; and the currents formed by the rotary motion, the inflow and outflow of the material, together with the effect of the guide-blades, causes the reduced material to pass numerous times against and into the quicksilver before escaping into the classifier, prior to being conducted away. As the quicksilver is kept in motion by the produced currents it is constantly presenting fresh surfaces to the material. The gearing for driving purposes is placed under the machine, being thereby protected from dust, and there is no danger of oil from the bearings getting into the machine to sicken the quicksilver.

Molloy's Hydrogen-amalgam Pan.

Some of these pans have been erected in the North Island of New Zealand, but they have not been so successful in the treatment of the ore as contemplated at first outset. However, several improvements have been made, and possibly they may now come into more general use. Experiments with these pans were made in London, and from the description of them and their action on the mercury one might expect that they would be invaluable amalgamating-machines. Annexed is a sketch showing their construction (see Fig. 24). The description appeared in the London *Times*.

"The difficulty, or, rather, the impossibility, of obtaining by mercurial amalgamation anything like a full yield of gold from what are known as refractory ores has long been recognised, and has led to the appliance of various remedies from time to time. The difficulty arises from the circumstance that in some ores the gold is variously associated with sulphur, iron-oxide, arsenic, antimony, or zinc; and the presence of any of these ingredients destroys the 'quickness' of the mercury, and so renders it sluggish, and incapable of seizing and retaining the atoms of gold. The most recent invention in connection with the present subject is the hydrogen-amalgam process, which has been invented by Mr. B. C. Molloy, M.P., the working of which on a practical scale we recently witnessed at the laboratory of Messrs. Johnson and Sons, of Cross Street, Finsbury, assayers to the Bank of England. The principle involved in this process is the well-known one that when gold is brought into absolute contact with clean or 'quick' mercury the gold is absorbed by and retained in the mercury, from which it is afterwards retorted. In cases where refractory ores have to be dealt with, they cause the mercury to 'sicken'—that is, to become coated with an oxide which lies like a sheet of paper on the surface of the body of the mercury preventing contact between the particles of gold and the clean portion of the mercury. This sick mercury also powders away, or, as it is termed, 'flours,' so that the floured liquid metal is carried away and lost, leaving fresh surfaces to be attacked by the injurious ingredients in the ore. The result is, therefore, that not only is the gold not captured, but mercury is lost and carried away in the tailings. With some of the less-refractory ores the loss of mercury is from 2lb. to 6lb. per ton of ore treated; while in some other cases the loss is much greater. Owing to this difficulty in the treatment of auriferous ores, it has been estimated that the average of 40 per cent. of the gold contained in the ores treated is annually lost. The object of the hydrogen-amalgam process is to save this enormous loss of gold and mercury, and, according to authoritative reports, this object is completely and successfully attained.

"The method pursued is—first, to maintain the 'quickness' of the mercury, no matter how deleterious the character of the ore; and, secondly, to insure a continual contact between each separate particle of the pulverised ore and the quick mercury. The apparatus for accomplishing this consists of a shallow pan about 1in. in depth and 41½in. in diameter, which contains mercury about ½in. in depth. In the centre of this pan is a porous jar, so placed and fixed that the mercury cannot enter or move it. Within this jar is a cylinder of lead and a solution of sulphate of soda. This lead cylinder, which constitutes the anode, is connected with the positive pole of a small dynamo machine, while the mercury is connected with the negative pole of the same dynamo. When the current passes oxygen is evolved from the surface of the lead anode, while hydrogen is evolved from the surface of the mercury. This action, which is apparent to the eye, is of course due to the decomposition of the electrolyte formed by the solution of the sulphate of soda. The mercury combines with a portion of the hydrogen, and so forms a hydrogen-amalgam, while the excess of hydrogen so formed passes away. Now, while the mercury is thus charged with hydrogen it cannot oxidize, because of the presence of an excess of hydrogen. Thus, no matter what the character of the ore, the mercury under these conditions is always quick, and greedily attacks and absorbs the gold into itself.

"So far the ingenious but simple method of maintaining the mercury in a quick condition. We now have to describe the equally ingenious and simple means whereby the pulverised gold-ore is brought into absolute and maintained contact with the quick mercury. Floating upon the surface of the mercury is a disc 40in. in diameter, which dimension leaves a narrow outside channel all round the edge of the pan where the mercury is uncovered. The centre of the disc has a circular hole in it so as to clear the porous jar by about 2in. This central opening in the disc has a rim about 2in. high, which forms a hopper. The disc, as it floats on the mercury, is slowly revolved by simple mechanism. The pulverised ore, as it leaves the stamps or other crusher, flows into the hopper accompanied by a stream of water, and is then, by centrifugal action, carried under the revolving disc and rolled round in the mercury in ever-increasing circles until it reaches the periphery of the disc, and, consequently, the outward channel between the edge of the disc and that of the pan. Here, freed from the pressure of the disc, the pulverised ore floats up and over the edge of the pan and passes away, leaving behind it in the mercury every atom of gold it previously contained. This perfect extraction is due to the rolling action, which separates each particle of the ore, and rolls it for some ten seconds in the bright, quick mercury, which wrests every atom of gold from it. The whole machine weighs only about 5cwt., and its working-capacity is 10 tons per day.

"It will be seen that the conditions which here obtain are the most perfect for the purpose of amalgamation. Owing to the perfection of contact no float-gold can escape, and in the presence of hydrogen no sickening of the mercury can take place. Hence every particle of gold is secured, unless mechanically encased in an atom of ore. The process has long since passed the experimental stage, but the Hydrogen-amalgam Company, who are working the patents, were careful not to publish any particulars of the invention until it had assumed a practical form, and could be deemed a commercial as well as a scientific success. Machines are at work in the United States of America (where they have been tested and favourably reported on by Professor P. de Pierre Ricketts), the Transvaal, and Mexico, while some are now on their way to India, Australia, and New Zealand. It is stated that the increased quantity of gold extracted by this process has never been less than 10 per cent., and that in most cases a much larger percentage is reached. In short, this method of applying electricity with the intervention of a porous wall or cell has overcome all the difficulties previously encountered, while the whole cost of treatment by this process is said to amount to only about 3d. per ton for both electrical and mechanical force, and for labour."

A plant of Watson and Denny's pans and Molloy's hydrogen-pans are erected at the New Alma and Victorian Gold-mining Company's Mine, Australia, which gives satisfactory results.

The following tables show the cost of labour and the loss of gold under the old plant at this mine compared with that under the new plant:—

A.—Under the old plant, working three shifts, there were required for working 800 tons per month of solid mundic—

4 boys at blankets.
3 men at strakes and concentrators.
3 men at roasting-furnaces.
1 man carting concentrates and wood to furnaces.
1 man carting roasted ore to mills.
3 men attending amalgamating-pans, &c.

	£	s.	d.
Say 15 men at £8 per month	120	0	0
Loss of gold in tailings, 800 tons at £1 10s.	1,200	0	0
	1,320	0	0

B.—Under present plant, consisting of the Watson and Denny pans and the hydrogen-amalgam pans, as shown on general plan here given—

	£	s.	d.
Six men in front of battery, at £10	60	0	0
No account is taken of extra fuel, as this is covered by fuel formerly used for roasting.			
Tailings shown by fire-assay to be practically worthless.			
Balance	1,250	0	0
	1,320	0	0

This balance of £1,260 monthly shows the profit gained by the new plant.

The Watson and Denny Concentrator.

This is a machine that the patentees claim to be capable of treating 200 tons of ore per week, which appears to me to be more than it is able to treat satisfactorily; but, as the machine has not been sufficiently tried, its concentrating-capacity is not well known, and more will have to be known respecting it before it could be recommended.

The patentees have designed it as a concentrator for treating crushed ores and metalliferous slimes of every description, having the following advantages over other concentrators: (1) The great quantity of material that it is capable of treating in a given time, at a cost not exceeding 3d. per ton; (2) that very little labour and attention is required; (3) the motive-power required to work it is very little; (4) that there is very little wear-and-tear.

General Description.

This is a circular concentrator, consisting of one and two floors: whether one or two floors depends on the material to be treated. If used for concentrating gold and auriferous pyrites one floor only is employed, as in this case it is considered advisable only to separate the headings or concentrates from the tailings. If, however, it is intended to treat other minerals, such as galena, copper-pyrites, zinc-blende, silver- and tin-ores, two floors are used, and the products are "headings," "middles," which are reworked, and tailings, which are allowed to escape.

The inner diameter of the concentrator is 30ft. The floors are stationary, and are made of any suitable material that is durable and gives a sound, smooth surface, such as concrete, brick or stone coated with cement or asphalte. After the slimes have been conducted from the chute into the basin and down the feed-launder, they are spread over the periphery of the floor by a distributor which passes round the table once a minute. During the time to make one revolution of the feed-launder the tailings are separated from the concentrates, and both are washed into their respective gutters, thus leaving a clear floor for fresh material every minute. Wash-water is continually flowing over the floor from an angular ring, which can be regulated by a stopcock in each of the distributing-pipes. The lighter slimes are washed off the table with clean water into a gutter arranged for their reception below, while the concentrates have a special radial pipe provided with copper nozzles, from which strong jets proceed and wash off the whole of the concentrates before fresh slimes are added.

The single-floor concentrator is similar to the outer floor of the double one. The inner floor of the double concentrator is constructed in the same manner as the outer one, only it is smaller. The second floor in the double one is used to rework the "middles." The floors are fixed, and inclined downwards from the periphery to the centre, and in the centre there is a hollow vertical driving-shaft, to which are the radial water-supply pipes, having their other ends connected to an angular perforated pipe. The necessary stays for holding these pipes in position, as also the feed-launder, are also connected with the driving-shafts, and these constitute the only movable parts of the machine. Water flows down the hollow central shaft into a cistern fixed on to it near the bottom, from which the distributing-pipes radiate that convey the water to the perforated angular pipe upon the periphery of the floor. The water from this pipe passes on to the floor, where it sorts the mineral and metallic particles according to their specific gravity. The quantity of water conveyed by each radial pipe to its special part of the angular ring can be regulated by a stopcock. The iron stays supporting these pipes in position are attached at the upper ends to a reservoir, which is placed round the driving-shaft. This reservoir holds the pulp to be treated before it flows down the feed-launder to the floor. The lower end of the feed-launder, which terminates just above the periphery of the floor, is provided with a distributing-board fitted with diamond-shaped buttons to direct the flow of the water and material to be concentrated.

A curved radial pipe, fitted with copper nozzles bent towards the centre of the machine, is arranged just in front of the feed-launder, the water from which washes the headings off the floor down a chute to an inner central gutter, while the tailings are constantly flowing off the inclined plane into a central gutter placed below the floor to receive them. If two floors are used, a portion of this gutter just before the headings is bridged over by a revolving iron bridge, so that the middles can flow over to the inner and lower floor to be further concentrated.

Not having seen these concentrators at work, an exact description of their method of dealing with the concentrates cannot be given; but the principle is similar to a large buddle which separates the lighter from the heavier particles or grains of crushed mineral substance by gravitation; but, as already stated, these machines require to be more in use and sufficiently tried under different conditions before they can be recommended as machines capable of satisfactorily dealing with slimes and concentrates from every class of ore.

On my former visit to the Australian Colonies, about four years ago, the principal concentrators used then were Haley's tables, and these are in use still, and are found to be good concentrators for auriferous pyritous ores; but the objection to these is, they have to be stopped at short intervals to be cleaned out. This is a serious objection when treating poor lode-stuff. The machines that are wanted at the present day are those that can be kept constantly going, and are able to satisfactorily treat the ore at a small cost. The first cost of the machine is but a small item in the expenditure where manual labour has to be employed, combined with a stoppage of the works at short intervals. Any concentrating-machine should be self-acting, and when once in motion it ought to be kept constantly going, in order to reduce the working-expenses to a minimum.

NEWBERY AND VAUTIN CHLORINATION PROCESS.

The plant erected at the United Pyrites Company's works at Sandhurst, and the whole process is the same as that described in a paper read by Richard P. Rothwell before the American Institute of Mining-engineers, New York, and published in the *Mining and Scientific Press*, of San Francisco, of the 20th October, 1883, with the exception of the air-pump that is supplied with the

PLANT AT NEW ALMA AND VICTORIAN G.M.Co.

Watson and Denny Pan

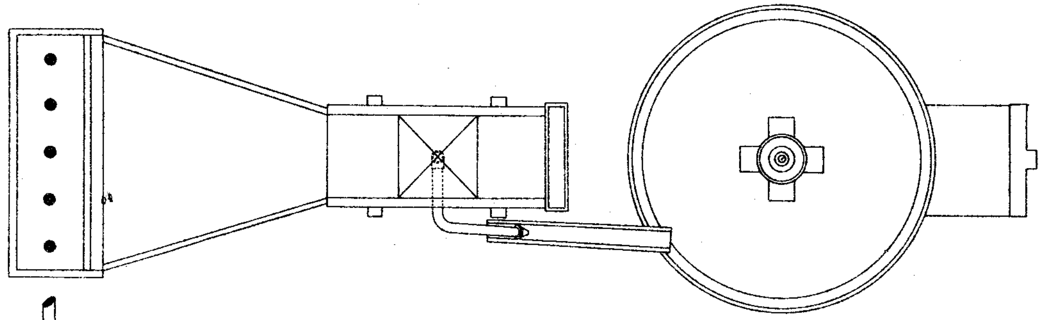


Fig 22.

Section

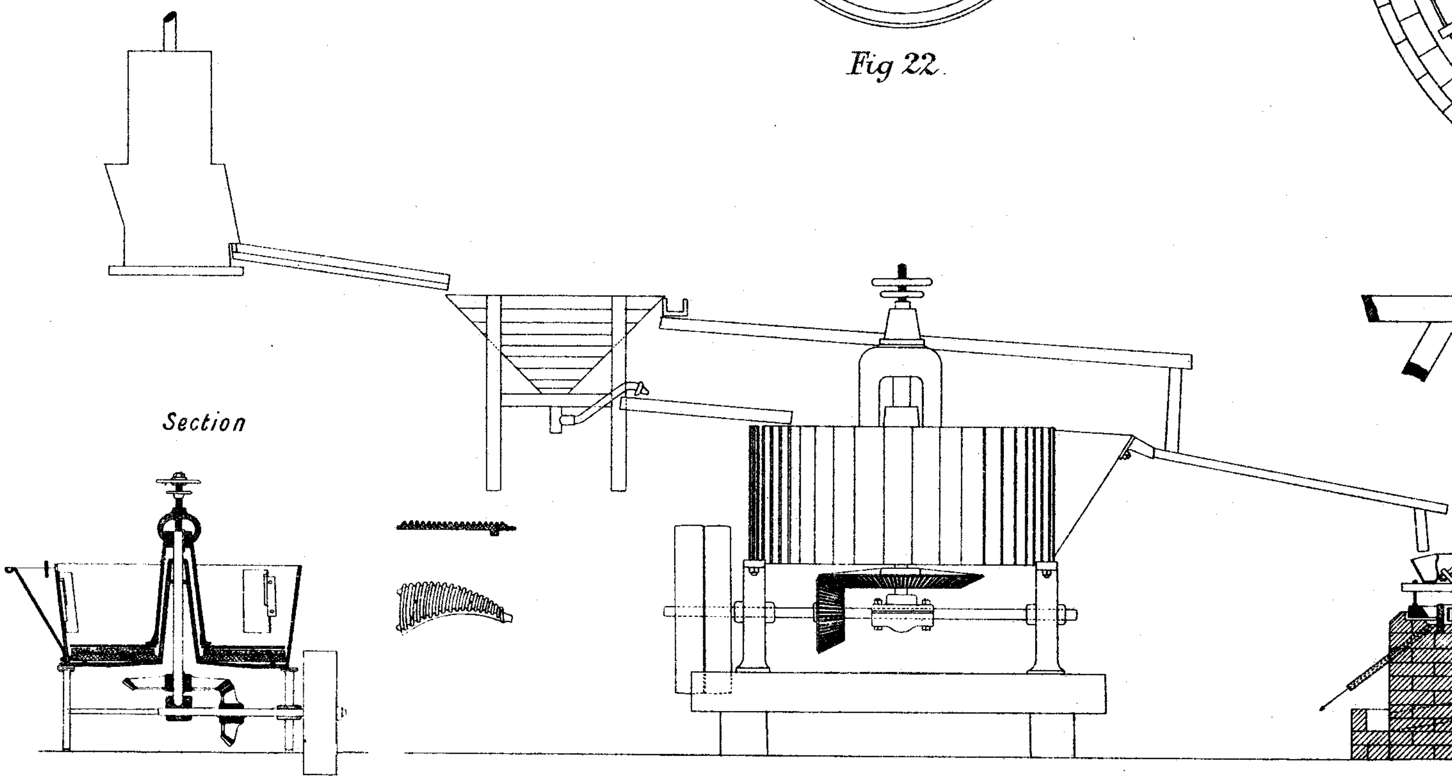


Fig 23.

Hydrogen Amalgam Pan

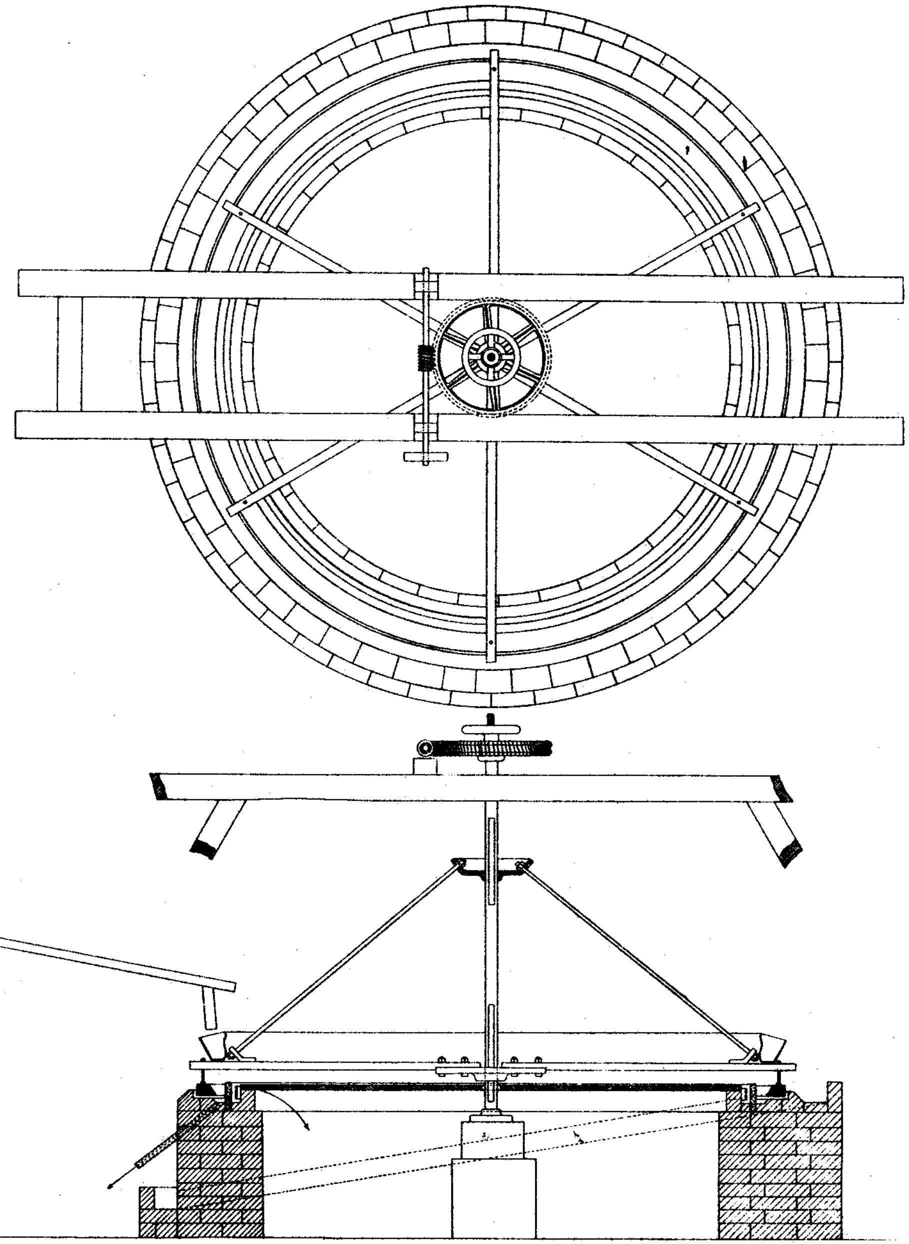


Fig 24.

MOLLOY'S HYDROGEN AMALGAM PAN

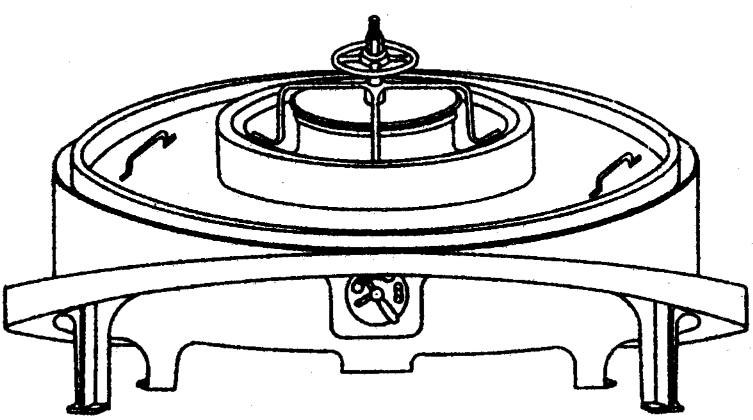


Fig 24.

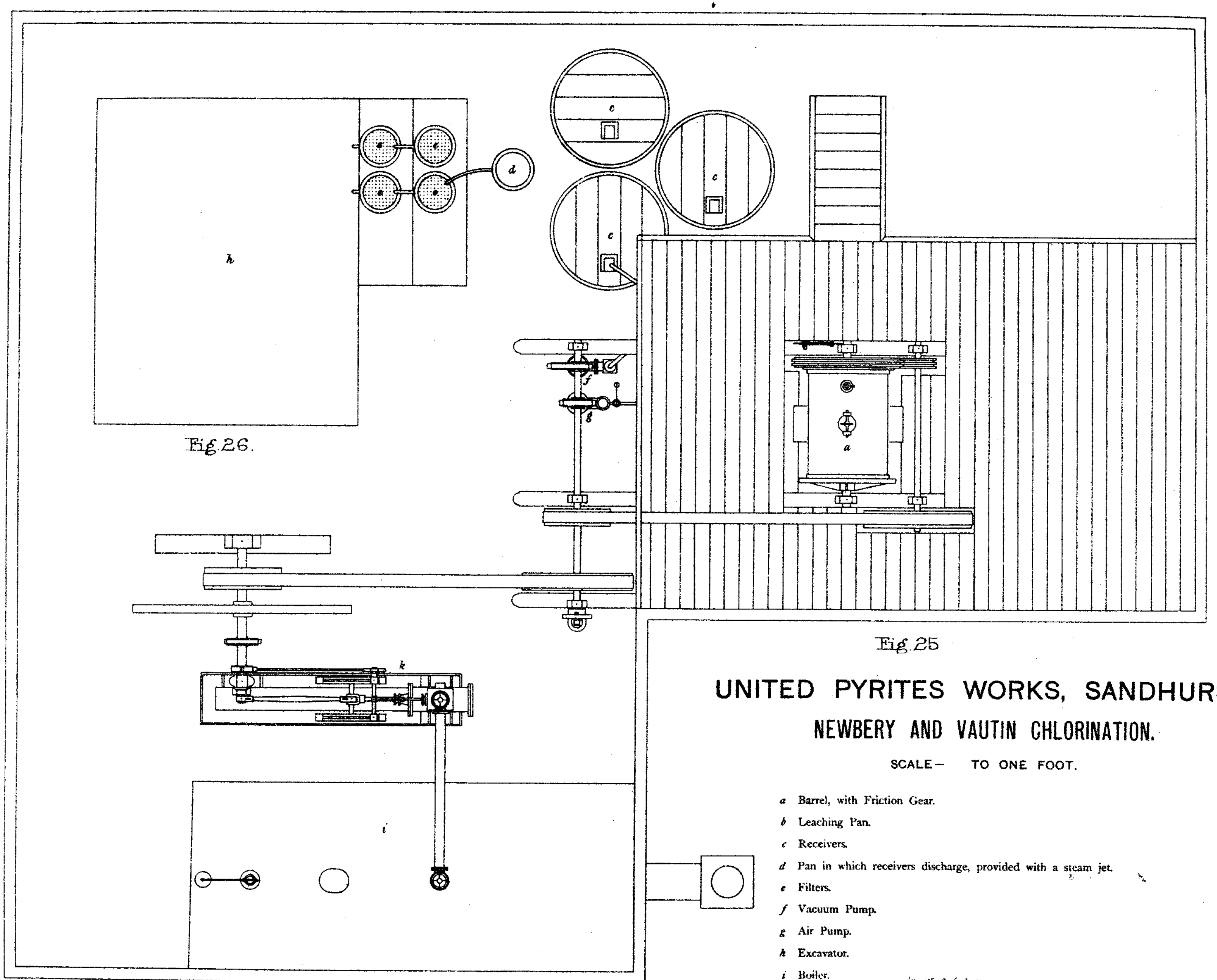


Fig. 26.

Fig. 25

UNITED PYRITES WORKS, SANDHURST.

NEWBERY AND VAUTIN CHLORINATION.

SCALE— TO ONE FOOT.

a Barrel, with Friction Gear.
b Leaching Pan.
c Receivers.
d Pan in which receivers discharge, provided with a steam jet.
e Filters.
f Vacuum Pump.
g Air Pump.
h Excavator.
i Boiler.
k Engine.

UNITED PYRITES WORKS, SANDHURST.

NEWBERY AND VAUTIN CHLORINATION.

a Barrel, with Friction Gear.
b Leaching Pan.
c Receivers.
d Pan in which receivers discharge, provided with a steam jet.
e Filters.
f Vacuum Pump.
g Air Pump.
h Excavator.
i Boiler
k Engine.

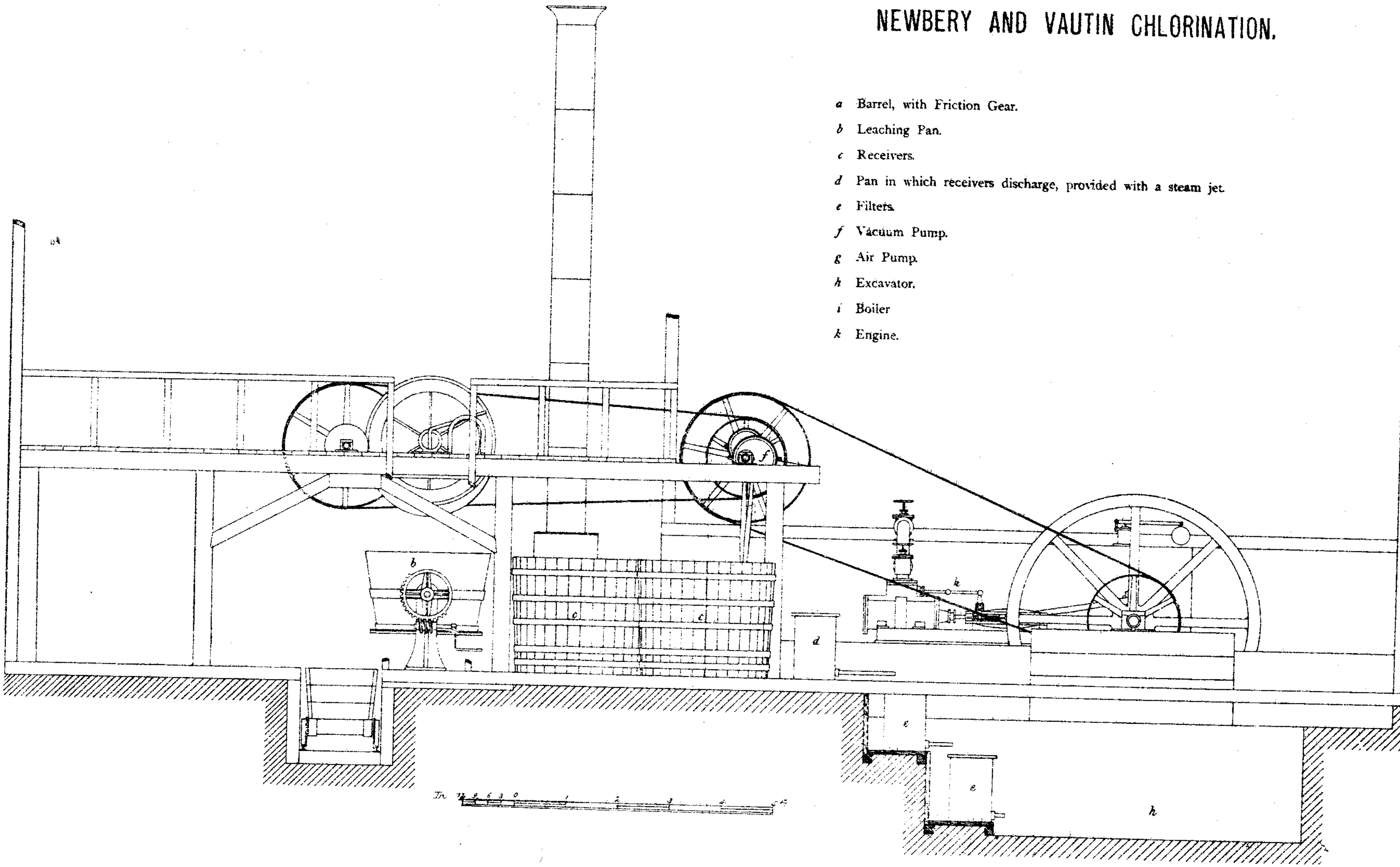


Fig 26.

Newbery-Vautin plant, and which is found to be of no service by the United Pyrites Company. They do not use it, as they find it of no advantage to the process of chlorination. The only improvement in this plant from that of the Mears process, which Mr. Rothwell described as used in North Carolina in 1873, is a vacuum-pump to extract the liquor from the sand after the chlorinating-barrel is emptied, which was not referred to by Mr. Rothwell; but it does not seem to be a new process.

The pyrites is brought to the United Pyrites Company's works from the several crushing-batteries on the field, also from Dalesford and Stawell, to be treated. It is operated on, chlorinated, and the gold extracted at a cost of £3 per ton. The company also purchase pyrites by assay, and treat it on their own account. The process is as follows:—

The pyrites is first roasted in reverberatory furnaces the same as for the amalgamating process, and then taken to the chlorinating plant, which consists of a revolving cylinder or barrel made of iron, and lined with sheet-lead about ¼in. in thickness. The iron cylinder is about 4ft. 6in. in diameter, with flanges at both ends. The lead is made in a cylinder and slipped inside the iron cylinder. The ends are then hammered down against the flanges of the iron cylinder, thus forming a lead flange. The ends of the cylinders are covered with a sheet of lead and bolted on to the flanges of the cylinder. There is a hole on one side in the centre, and the lead is hammered up to cover the edges of this opening in the iron, so that none of the liquor in the cylinder touches any part of the iron. Directly under this revolving cylinder there is a leaching-pan also lined with sheet-lead; but a wooden vat coated with asphaltum would answer the same purpose. There is a false bottom in this pan about 1in. above the main bottom, with perforated holes. On the top of the false bottom there is a filter-bed of quartz and coarse sand about 5in. in thickness. This is covered on the top with a very coarse cloth, the object being to preserve the filter-bed from being damaged when running out the waste sand. There are also two liquor-tanks, each about 5ft. in height and 4ft. 6in. in diameter. Besides these there are a number of earthenware jars filled with charcoal, through which the liquor filters, the gold being deposited on the charcoal.

The revolving cylinder is charged first by putting in water for about 1ft. in depth and about 34lb. of sulphuric acid. The ore is then put in—about 25cwt. in one charge—and to this is added about 30lb. of chloride of lime; but the quantity of sulphuric acid and chloride of lime depends entirely on the quantity and chemical properties of the ore to be treated—some ores require a larger quantity than others. The cover, having an indiarubber joint, is now screwed on in the same manner as on a gas-retort, and the cylinder set in motion. There is a valve attached to the cylinder to admit of air being forced in; but the person in charge of the work at the time of my visit informed me that they had disregarded forcing in air, as they found it of no advantage in the process of chlorination. The amount of gases generated by the mixture of sulphuric acid would be considerable—probably about from 25lb. to 30lb. to the square inch—that is, if the ore has been perfectly roasted before being operated on, but, if not, the pressure inside the cylinder would be greater.

The revolving cylinder is driven by a belt and pulley on an intermediate shaft from the engine. This shaft also works two pumps—one for air if required, and another for creating a vacuum. The cylinder is kept revolving at a speed of about seven revolutions per minute for about three hours, more or less, according to the fineness or coarseness of the gold, when the whole of the gold is in solution. The cylinder is now stopped and the cover removed. Afterwards the cylinder is again set in motion, and the ore and liquor falls into the pan or vat with the false bottom already described, and when empty another charge is made, and the same operation gone through.

A vacuum-pump is connected with the main bottom of the pan or vat already described, and the liquor is drawn through the sand and goes into the other two receiving-vats. Water is added, and the liquor tested from time to time as it comes from the leaching pan or vat, and when, by adding sulphate of iron in the test, the liquor remains perfectly clear, this operation ceases. After all the gold-liquor has been removed the sand is cleaned out ready for the next charge. In the vats which contain the liquor there are placed coils of perforated leaden pipes, which are attached to the air-pump, and the action of this pump forces out any gases in the liquor. The liquor-vats stand about 2ft. above the first row of earthenware jars which contain the charcoal. There are three rows of these jars, one below another, so that the liquor filtering through one passes through the one below it. These jars are of different diameter, but it can be said that the liquor filters through a column of charcoal about 4ft. in height. As the solution passes through the charcoal the gold is precipitated in a metallic state. The charcoal is then burned, and the ash mixed with borax and melted. The water in passing through the charcoal is tested from time to time to ascertain if the waste liquor contains any gold.

In the pamphlet published by Messrs. Newbery and Vautin some time ago the cost of this process was to be about 12s. per ton; but this statement is not borne out by those that are using it. Mr. S. McGowan, the legal manager of the company, assured me that the cost of treatment is as much as with the old chlorination-vats, but that the Newbery-Vautin process gave a higher percentage of gold. It is estimated that an average of 93 per cent. of the gold is obtained by this process. Annexed is a plan of this plant at the United Pyrites Company's works at Sandhurst. (See Figs. 25 and 26.)

The Newbery-Vautin process is, however, too costly to deal with low-grade ores. There are chlorination-works erected at the Butters Ore-milling Works, Kennett, on the banks of the Sacramento River, on the line of the California and Oregon Railroad, and now in operation, and if a description of the results of these works is to be depended on they will supply a want which is greatly needed in the treatment of low-grade ores. The following description is taken from the *Mining and Scientific Press* of San Francisco:—

"The works consist of a roasting-house containing a reverberatory furnace 43ft. outside and 12ft. 8in. wide, containing three hearths each 10ft. square, a dust-chamber, and two alternate cooling-pits for chlorination on cooling-floor. The lixiviation-house contains two ore-tanks 14ft. in diameter and 5ft. high, with a capacity of 20 tons to each tank. The tanks have counterbalanced lids fitting in water-channels on the outer edge of tank when chlorine gas is used. The two precipitating-tanks are 10ft. in diameter and 6ft. high, and the lower or copper precipitation-tanks are 12ft. in diameter and 4ft. high. The tanks and foundations are all built on inclined watertight floors leading to the waste-tanks. The generator for the production of chlorine is a very satisfactory machine, made, after the designs of the builder, to run by power, and heated directly by a wood fire, being practically a lead-lined kettle of 120 gallons capacity, arranged with valves and safety overflows, the connection with the vats being made with water-stopped lead valves, no hose being used in the works. These works are not only the most recent, but undoubtedly the model gold-chlorination works on the Coast."

"They are now handling 1,000 tons of gold tailings, assaying from $4 to $6 [16s. 8d. to £1 5s.] to the ton, at a cost of about $1 75c. to $2 [7s. to 8s. 4d.] per ton, by the chlorination process. The fact that such low-grade material, in which not a trace of gold is visible, and from which but 30c. [1s. 3d.] per ton, even after roasting, could be obtained by amalgamation, can be treated by the chlorination process at a profit is one that, to say the least, deserves notice. The capacity of these works is from 300 to 400 tons per month, of which only about one-tenth of th material operated on is roasted."

HEDLEY'S PATENT ELECTRIC ORE AND METAL CONVERTER.

This is a process that has been invented and patented by Mr. John Hedley, of Heathcote, Victoria, for crushing and treating ores, separating and refining metals, by the aid of electricity; but, as it has not yet been tried on a large scale, nothing can be said as to its success or otherwise. However, as it is on an entirely new principle, a description of it may be interesting, as inquiries were made by mine-proprietors in New Zealand, and instructions forwarded to me in Melbourne to inquire particulars respecting it. Mr. Hedley describes the process thus (taken from the last Victorian Mining Report) (see Figs. 27 to 36 of annexed sketches):—

" My apparatus has for its object the complete, economic, and speedy extraction of metals from high- and low-class ores, and the subsequent separation and refining of the same, and is particularly suitable for the treatment of those ores containing the more valuable metals—namely, gold, silver, and bismuth, and their combinations.

" It will be seen by the description hereinafter contained that two metallic baths are employed, one for the treatment and extraction of ores which do not contain a sufficient percentage of fusible material for smelting, and another for those ores which permit of the fusion of the whole mass, and the separation into slag and metallic regulus.

" My process of refining, hereinafter described, refers more particularly to the deposition of lead and antimony, and provides for the separation of the valuable metallic constituents from an antimonial regulus or base bullion by the deposition of the antimony or lead, as the case may be, by electro-chemical agency.

" Before describing the several parts and construction of my metallic baths, and the separation and refining process, I wish it to be understood that before the treatment of the more valuable ores I find it necessary to reduce the latter to a finely-divided state, which is accomplished preferably by my improved pulsating crushing or pulverising machine, illustrated at Fig. 27 in the accompanying drawings as a combined half-vertical section and front view. The construction and mode of operation of said machine are as follows: The frame consists of two longitudinal timbers A, supported and secured by four vertical standards B. Said timbers carry the transverse pieces B^1, upon which I secure two pillow-blocks or other bearings C, as shown, to receive the horizontal shaft D, having two grooved eccentrics or cams E keyed upon it to impart a pulsating motion to the conoidical-shaped casting F, which is suspended by two metallic bands or straps G, connected by a double shackle or other clevis H at four equidistant points to the circular wall of the conoidical-shaped casting F. Between the said bands or straps G and the grooved eccentric sheaves E, I insert removable brasses I. Each of the latter is provided with a set-screw J and jamb-nut as shown. By this means the vertical pulsating motion given to the casting F may be varied, to reduce the ore or material to any required degree of fineness for after-treatment in the metallic bath shown at Fig. 28. A conical-shaped body K, of a less angle than the aforesaid casting F, is secured by any suitable means to the cone-shaped bolster L affixed to the vertical shaft hereinafter described. A rotary motion is imparted to this body K, through the medium of gearing and a driving pulley M affixed as shown upon the horizontal shaft D. The crushing faces or surfaces of the casting F and the body K have furrows formed at an oblique angle on each and in opposite directions, and which die or lead into a plane surface of about 4in. from the skirt of each face. These faces present an extensive crushing-surface, and may be hardened or chilled to reduce the wear-and-tear to a minimum. The vertical shaft N, kept perpendicular by the affixed bearing N^1, as shown, has a bevelled spur-wheel O secured near its top end, to engage with the pinion P keyed upon the horizontal shaft D. To prevent the latter from moving laterally when revolving I affix a collar Q (provided with a set-screw) preferably against the outer sides of the aforesaid pillow-blocks or other bearings C. The bottom or foot of the vertical shaft N revolves within a footstep R, held in position by the two set-screws S (and jamb-nuts) projected from an outer receptacle T. To guide the casting F in a vertical line when in motion, I affix the two blocks U to the supports of frame. These blocks have concave faces to correspond with convexed-faced ears V cast upon the periphery at the bottom edge of aforesaid casting F. The ore or other material to be crushed or pulverised is fed in any convenient manner into the circular wall of the casting F, thence by its own gravity falls into the wedge-shaped space or opening W, where it is crushed between the furrowed and plane surface or face of the pulsating casting F and the body K, both of which afford sufficient vent at the bottom for the passage of the crushed material.

"Having thus explained the method of reducing the ore to a comminuted or finely-divided state, I will now distinguish the remaining figures upon the four accompanying sheets of drawings illustrated for treating ores, separating and refining metals, by the aid of electricity: Fig. 28 shows a vertical section of a metallic bath and furnace, with condensing-flue, adapted for the treatment of low-class ores of gold, silver, and bismuth. Fig. 29 illustrates a plan of Fig. 28 upon the horizontal line A^1. Fig. 30 is a vertical section taken upon the vertical line A^2 upon plan, Fig. 29 showing the furnaces with grate-bars. Fig. 31 represents a side vertical section of a set of flues for condensing fumes and increasing the draught of my apparatus shown at Figs. 28 and 32, and which flues may be adapted to calcining, smelting, or other furnaces where it is found necessary to arrest the volatile products from an economic point of view. Fig. 32 illustrates an alternative arrangement of a metallic bath and furnace adapted for the treatment of those high-class ores which permit of the fusion of the whole mass and the separation into slag and metallic regulus. Fig. 33 illustrates a half-top plan and a section upon the horizontal line A^3 shown at Fig. 32. Fig. 34 represents a plan of the arrangement of four vessels or receptacles adapted to my apparatus for the deposition of antimony or lead. Fig. 35 is a longitudinal section of the same upon line A^4, and Fig. 36 a transverse section through two vessels or receptacles.

"Similar letters refer to similar or corresponding parts where they occur in the several views. For the treatment and extraction of valuable metallic products from thoroughly-calcined and finely-divided ores which do not contain sufficient percentage of fusible material for smelting, I employ the apparatus shown at Fig. 28, consisting primarily of a metallic bath, which is further heated by the aid of an electrical current sufficiently powerful to increase the temperature of said bath to beyond the fusing-point of the metal to be extracted, from which latter, as it rises by its lighter specific gravity from the bottom to the surface of the bath, I extract 95 per cent. of the metallic constituents, a matter of great consideration.

"The consistency of the metal in the metallic bath aforesaid will be determined by the base metal contained in the ore to be treated, provided the fusing-point of such metal be not greater than 1,000° Fahr.—that is to say, should the ore under treatment be antimonial, the bath would preferably consist of antimony, because if lead were used it would only increase the number of constituents of the regulus drawn from the bath, which would consequently increase the cost of subsequent separation.

"In all cases where molten lead can be used as a bath without the disadvantage above stated, it is preferable, so far as the subsequent separation of the base metal is concerned; but any metal of low fusion may be used with equally good results.

"The construction of the aforesaid treating-apparatus, shown at Fig. 28, is such as to allow the ore under treatment containing base metal in addition to the gold, silver, and other constituents, to separate from such made metal, which will flow from the bath into suitable receptacles placed beneath.

"The combination of parts and mode of treating the ores in the metallic bath above stated will be understood from the following: Within a casing of brickwork or masonry *a* I arrange two covered iron vessels *b* and *c*, each being lined internally with refractory fireclay *d* to contain the aforesaid metallic bath. Said vessels have communication with each other by the connecting-pipe *e*, and are heated by the furnaces *f* formed in the casing *a* as at Fig. 30, in such a manner as to permit of a flue to partially surround the vessels and open into a condensing-flue leading into the main one. The vessel *c* is adapted to tap the larger one *b* by the discharge-pipe *g* affixed in its bottom as shown, and provided with a fireclay plug. A surplus-metal discharge-pipe *h* is secured upon the outer side, near the top of said vessels *c*, to insure one uniform height of the metallic bath by discharging through it any surplus metal it may contain.

"One of the essential features of this apparatus consists in suspending a rotary cone-shaft *i* to communicate with the bottom of the metallic bath. This cone-shaft *i* is in two flanged sections, the lower section *j* being made of fireclay, and the upper half *k* of iron. These sections are secured to each other by bolts passed through the said flanges. The neck or top end of the upper half *k* has a flange or turntable *l* cast or attached to it to revolve upon metallic balls *m* or rollers placed loosely in a grooved collar formed upon the top face of the bed-plate *n*, which is secured to the frame *o* carried up from the brickwork *a*—or any support may be used. To revolve the said cone-shaft *i* (provided with a funnel at its top) I form bevelled teeth upon the periphery of the turntable *l* to engage with a pinion *p* affixed upon the spindle or shaft *q* arranged in the bearings *r*, and carrying a pulley *s*, to which motion is imparted to revolve the cone-shaft *i*.

"To effectually and uniformly diffuse the comminuted ore into the column of molten metal I construct any number of fireclay ribs *t* upon the inner periphery of the cone-shaft. These ribs hold and revolve the material with the said cone-shaft until it reaches the bottom, where it is diffused by a dividing blade t^1 of fireclay or other suitable material, affixed in the centre upon the bottom of the vessel *b*.

"After the diffusion of the ore in the metallic bath aforesaid it immediately rises to the surface. From there it is removed (to be consigned to the division of waste or slag) by any suitable rake or scraper inserted through any openings made in the conical hood or cover *u*, and provided with one or more removable doors U, one being shown with catches.

"In order to first convey, and then extract, the volatile products from the fumes arising from the metallic bath and the furnaces I connect a conducting-pipe U^0 to the conical-shaped hood or cover *u*, and also to the top of the cemented or iron-lined vertical flue U^1, having two sieve-discs U^2 affixed therein as shown, both of which are connected respectively with the positive and negative poles of an electrical battery. When these discs are electrized they assist the precipitation of the metallic particles carried by the fumes.

"By the aid of a force-pump a continual stream or jet of water is ejected from the nozzle of the pipe U^3 into the funnel U^4, which latter discharges it from its perforated bottom or its equivalent into the said condensing-flue U^1. It is by these means that the volatile products are condensed and precipitated into the funnel-shaped outlet U^5 of flue, thence into a suitable receptacle placed beneath. The water, in its passage through the flue, not only arrests and condenses the volatile products by the aid of the electrical current, but also increases the draught in and from the furnaces to the main flue U^6, which conveys the smoke and the like to the chimney U^7, shown disconnected from its stack. Should the draught in said condensing-flue U^1 at any time be insufficient, a vent-hole may be formed or cut either in the end or top surface of the aforesaid conducting-pipe U^0.

"I wish it to be understood that I do not confine myself to the combination and use of one condensing-flue U^1, as before described and illustrated, as Fig. 31 shows a set of flues, marked U^8, which may be interposed between the furnaces and the chimney, with a stream or jet of water directed and distributed into each, and discharged in a similar manner to that described in Fig. 28. These flues U^8 may be advantageously adapted not only to one or any number of my metallic baths and furnaces, but to other furnaces, such as those used for calcining, smelting, or roasting ores or other materials.

"Figs. 32 and 33 illustrate an alternative arrangement of the treating-apparatus shown at Fig. 2, which will be seen to consist of a metallic bath and furnace adapted for the complete fusion of high-class ores, and the separation into slag and metallic regulus. The construction and mode of action of this arrangement are as follow: A longitudinal iron vessel a^1 lined with fireclay to contain a metallic bath a^2, which vessel is surrounded by the casing of brickwork a^3, having two furnaces such as the one shown at b^1, communicating with the flues b^2, arranged as shown at each side of said vessel, to heat and keep the metallic bath in a molten state, the temperature of which is further increased by the application of an electrical current, as described in apparatus Fig. 28. The treatment and separation of metals from ores in the alternative arrangement is effected by affixing a fireclay partition b^3 in the electrical metallic bath as aforesaid, a transverse opening b^4 being left at the bottom of the latter for free communication between the two divisions. In one side, and above the ordinary level of the bath aforesaid, I place a pipe c^1, through which the slag is discharged, and for the withdrawal of the made or base metal I attach an outlet pipe c^2 (provided with a fireclay plug) to the bottom of said vessel. The pipe c^4 affixed on one end of vessel, as shown, permits of the discharge of surplus metal from the metallic bath. The cover c^3 of the latter, slightly arched in shape, is formed of fireclay of any suitable thickness, through which I introduce the double bent pipe d^1, to convey, preferably, a gas-flame identical with that used for the fusion of silica at glassworks. The ore, in cubes of about 2in., when fed into the hopper d^2 is completely fused by the gas-flame at the surface of the electrolysed bath, the heat of which is also communicated to the ore.

"By this method the slags and metal are separated within the furnace. Or, in lieu of the gas-flame aforesaid, any suitable fuel may be introduced with the ore in its descent through the funnel d^2. By this means the whole mass would be fused in the same manner and with results previously mentioned. Should either the high- or low-class ores be charged with sulphides, thorough calcination will be necessary before treatment in the electrical metallic bath shown at Fig. 2, or the alternative arrangement, Figs. 32 and 33, last described.

"After the treatment and extraction of the metals from the ores before described, the metal from either of the apparatus shown at Figs. 28, 32, and 33 is withdrawn, to be moulded or rolled by any suitable means into slabs, the thickness of which may vary from 1½in. to 2½in., to be conveyed to the separating and refining division, where the base metal is decomposed and deposited by electro-chemical agency, as hereinafter described.

"Fig. 34 represents an apparatus designed for economically and effectually separating and refining the valuable constituents from antimonial or base-bullion regulus obtained after the treatment of the ores in my metallic baths.

"After the formation of either the antimonial or base-bullion slabs, I subject and expose them to the action of an electro-chemical solution contained in a series of vessels or receptacles hereinafter described. I wish it to be distinctly understood that for the deposition of lead a specially-prepared solution is required as much as for the deposition of antimony.

"In order to deposit, say, for instance, lead, I would alternately suspend an even number of slabs of the metallic regulus in its respective electro-chemical solution to become the soluble anode marked e^2, along with an equal number of plates of sheet-lead, also suspended in the same solution, to become the cathode marked as e^3. Now, for the deposition of antimony the antimonial metallic-regulus slabs would be suspended as the soluble anode, along with antimony plates as the cathode (as in the former case), in a solution prepared for its deposition hereinafter described. The anode in the case of either antimony or lead is suspended within a muslin bag, which retains the unsolved metallic constituents such as gold, silver, and bismuth after the decomposition of the said anode.

"Before describing the various parts in detail and mode of operation of the apparatus containing the solution for the partial immersion of the anodes and cathodes I prefer to describe a chemical solution for depositing lead. Such solution may be proportionately increased or reduced in quantity, according to size of apparatus, and consists of plumbic sulphate and sodic acetate: when simplified the solution would consist of 4 per cent. of acetic acid and 10oz. of sulphate of lead per gallon. The solution I prefer for the deposition of antimony would be composed of sulphate of antimony and potassic carbonate, consisting of ½lb. sulphate of antimony and 1lb. potassic carbonate per imperial gallon of solution, which is heated.

"For easy reference to the drawings without having to refer to the figures previously mentioned I will now describe them as follows: Fig. 34 represents a plan of the arrangement of four vessels or receptacles adapted to my apparatus for the deposition of antimony or lead, Fig. 35 is a longitudinal section of the same upon line A^4, and Fig. 36 a transverse section through two vessels or receptacles.

"One arrangement of vessels or receptacles, marked e^1, will thus be seen from the figures stated, but I do not confine myself to this, as several variations of form or number of vessels may be

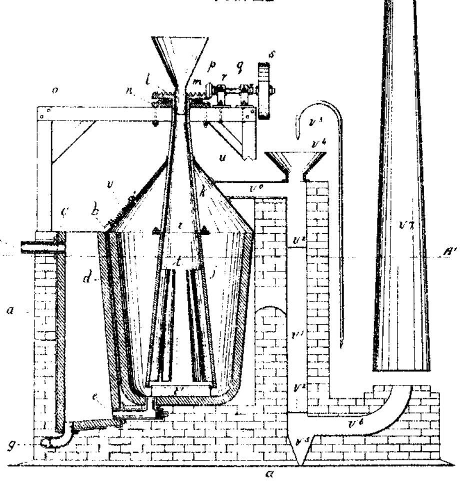


FIG. 28

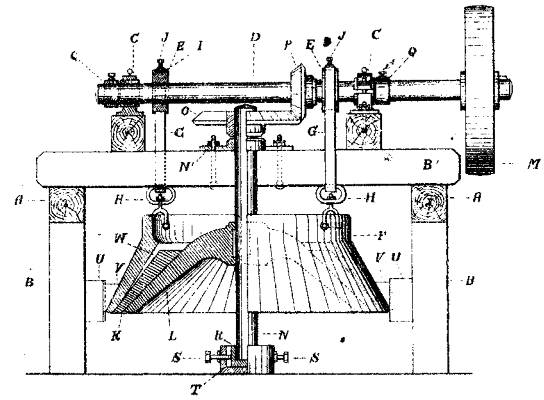


FIG. 27

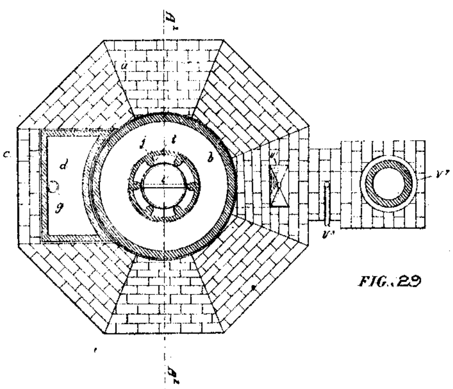


FIG. 29

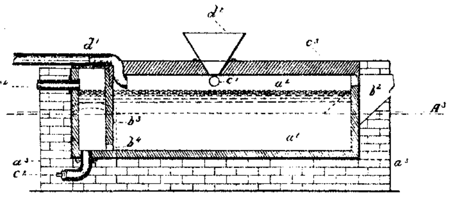


FIG. 32

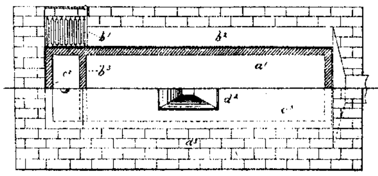


FIG. 33

HEDLEY'S

PATENT ELECTRIC ORE CONVERTER.

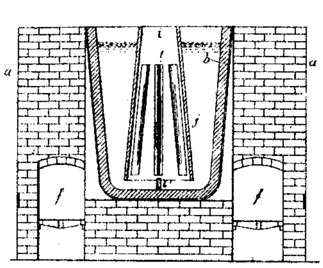


FIG. 30

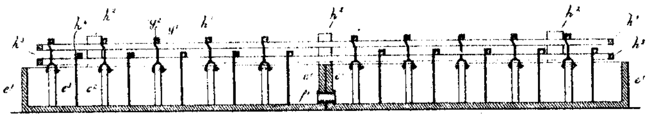


FIG. 35

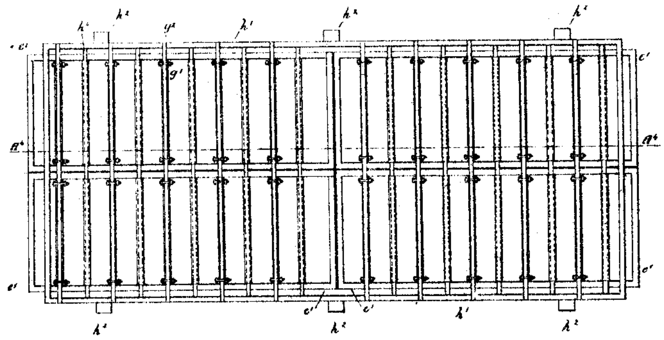


FIG. 34

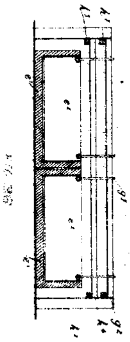


FIG. 36

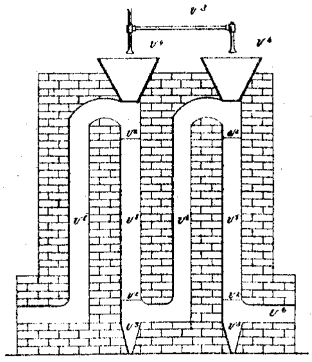


FIG. 31

adopted with equally good results to that illustrated. Each of these vessels or receptacles as shown is arranged to communicate with each other at their ends by an earthenware pipe such as f^1, through which either of the aforesaid solutions is made to circulate constantly by the aid of a force-pump constructed preferably of glass, from which an induction and eduction pipe is connected to and from two of the vessels or receptacles shown, or from two series of the latter of not less than twenty each.

"The soluble anodes herein described and illustrated, after being formed from the antimonial or base-bullion regulus, are suspended by the hooks g^1 from the transverse bars g^2, the outer ends of which rest upon the runners of a longitudinal parallel metallic frame h^1, supported by any number of standards h^2, insulated. The latter are also used to support another frame hereinafter described. The longitudinal parallel frame h^3 has also metallic bars h^4 laid transversely across said frame h^3, the ends only resting upon its side runners for the support of the cathodes, the top edges of which are simply bent over said transverse bars g^2. Said frame h^3 is arranged directly beneath the aforesaid one, and is likewise attached to the standards and insulated from the latter. I do not confine myself to the way in which I support the said metallic frames, as several variations may be substituted for that shown. In combination with my apparatus I may use a tank for the preparation of the solution; it may also be formed into a reservoir from which the solution in the vessels or receptacles may be replenished.

"With reference to the electrical current—which is derived from any suitable dynamo machine—used in connection with my apparatus, the positive pole of said current is connected to one end (in the centre) of the upper frame h^1 supporting the anode e^2, and the negative pole in the same manner and position to the lower metallic frame h^3 suspending the cathode e^3.

"By the action of the electrical current upon the solution through the electrodes, I am enabled to decompose rapidly the soluble anode of either lead or antimony, and deposit whichever metal the solution has been prepared for. By this means the desired metal is made to pass from the anode e^2 to the cathode e^3, upon which it is deposited, after which any valuable metallic constituents will remain in the muslin bag surrounding the skeleton anode, which may be again treated.

"Having now particularly described and ascertained the nature of my invention, and the manner of performing the same, I would have it understood that what I desire to claim and secure by Letters of Registration is—

"(1.) The combination and arrangement of the several parts or devices, supporting, connecting, and engaging with each other as herein described, and as illustrated in Fig. 27, for the effectual reduction of ores to a finely-divided state, for special treatment in the metallic bath shown at Fig. 28, as substantially set forth in the accompanying specification, and as fully illustrated in the drawings herewith.

"(2.) The employment and adaptation of a crushing-machine, such as shown at Fig. 27, in and to my improved apparatus for treating ores, separating and refining metals, as hereinbefore described, and as illustrated in the drawings attached.

"(3.) In the treatment and extraction of valuable metallic products from various ores, the combination and arrangement of the parts of my apparatus, shown at Figs. 28, 29, and 30, consisting essentially of two covered iron vessels b and c, connected with each other by the pipe g, and lined with fireclay d, to contain a metallic bath heated by an electrical current, and the furnaces f, communicating with a flue surrounding said vessels and leading into the condensing-flue U^1, all arranged and formed within the outer casing of brickwork a as and for the purpose hereinbefore set forth with reference to the drawings.

"(4.) In an apparatus shown at Figs. 28, 29, and 30, consisting of two vessels partially surrounded by a flue communicating with another for condensing the fumes and the like; the combination with said vessels, containing a metallic bath heated by furnaces f and an electrical current, of the pipes e and h; the rotary cone-shaft i, provided with ribs t and funnel as shown; the dividing blade t^1; turn-table l; the bevel gearing with respective parts, such as p, q, r, and s; the balls m, in combination with said turntable l and bed-plate n affixed upon the frame o; pipe U^3; the hood or cover u, with conducting-pipe U^0 and funnel U^4; the condensing-flue U^1, having an outlet U^5 at its bottom, and having arranged therein any number of sieve-discs U^2, electrized, all combined and arranged as and for the purpose hereinbefore set forth with reference to the drawings.

"(5.) In the alternative arrangement, described as an apparatus, and shown at Figs. 32 and 33, for the treatment of high-class ores, the arrangement of the fireclay partition b^3 having a transverse opening b^4 at its bottom, with the electrical bath a^2 provided with a fireclay cover c^3 to receive a bent pipe d^1 for the introduction of a gas-flame as described for the complete fusion of the whole mass; the arrangement of the discharge- and outlet-pipes c^1, c^2, and c^4, all combined and arranged as hereinbefore set forth and as illustrated.

"(6.) In the treatment of ores, the employment and adaptation of a set of flues such as U^8, Fig. 31, to one or more of my apparatus, as shown at Figs. 28, 32, and 33, or where it is necessary to arrest the volatile particles or condense the fumes or smoke from furnaces of various descriptions substantially as and for the purpose hereinbefore set forth and illustrated.

"(7.) In an apparatus, shown at Figs. 34, 35, and 36, for separating and refining the valuable constituents from the moulded or rolled slabs of antimonial or base-bullion regulus, for the deposition of lead or antimony, the use and arrangement with my other apparatus hereinbefore described and illustrated of four or a series of vessels or receptacles e^1 communicating with each other and containing an electro-chemical solution kept circulating constantly by the agency of a force-pump, preferably of glass, and also in combination with a supply tank or reservoir, all combined and arranged substantially as and for the purpose hereinbefore set forth and as illustrated.

"(8.) In the apparatus, as shown at Figs. 34, 35, and 36, for refining, separating, and depositing lead or antimony, the combination and arrangement of the metallic frames h^1 and h^3, connected respectively with positive and negative poles of an electrical battery, and supported by standards h^2 carrying the transverse bars g^2 and h^4 to suspend the slabs described as the anode e^2 within a muslin bag, and the plates as the cathode e^3, in a solution such as may be prepared for use for the deposition of lead or antimony, all combined and arranged as hereinbefore set forth, and as illustrated in the accompanying drawings."

ROCK-DRILLS.

There is hardly a mine of any note in the mining centres in Australia where there are not rock-drills employed. In sinking shafts, driving adits, and uprises, and in stoping out the lodes the rock-drill is in general use, and those who use it state that after a long experience in working it they are satisfied that its use reduces the cost of working very considerably. Indeed, many of the managers state that the mines could not be profitably worked were it not for the use of the rock-drill.

The rock-drills exhibited at the Melbourne Exhibition were the Eclipse, Slugger, Little Giant, Economizer, and Naylor and Thornton. It is a difficult matter to say which is the best description of drill, as each seems to have some points to recommend it. That is, each company using a certain pattern of drill has always something to commend it above the others. The National drill is greatly used and highly spoken of, as well as the Eclipse and Naylor and Thornton drills. The Slugger, Little Giant, and Economizer are but little known in the Australian Colonies.

Eclipse Drill.

This drill is manufactured by the Ingersoll Rock-drill Company, 10 Park Street, New York, in eight different sizes, and is made so that it can be fixed on a tripod or column-bar. It is only when they are used in open-quarry work, and in sinking very large shafts, that a tripod can be used; but these tripod-mountings, with heavy weights, make an excellent standard for the drill in open faces and places where there is plenty of room. (See Figs. 37 and 38, annexed sketches.)

Tripod.—The tripod supplied with the Eclipse drill is an adjustable one, made principally of steel, with three cylindrical legs, into the bottoms of which bevelled bars telescope, which are held and adjusted by means of set-screws. At the top these legs are fastened into sockets or hips, through which two hip-bolts pass, having right- and left-hand threads and long heads, and screw into a cup or saddle to which the drill is bolted. After the tripod is in place weights are hung on the legs to hold it steady. By loosening the hip- and saddle-bolts the drill can be pointed at any angle up and down or sidewise, and the legs, turning on the hip-bolts, can be moved backward or forward. By means of the telescope legs they can be lengthened or shortened, thus adapting the tripod to the most uneven surfaces, lying at any angle. The annexed sketch shows the drill mounted on tripod ready for work.

Shaft-columns.—For tunnelling, experience has proved that shaft-columns are the best mounting for drills. In large railway-tunnels a drift can be made of convenient dimensions along the line of the roof, and after this drift is made a tripod can be used for taking up the bottom. The column can be easily shifted when shots have to be fired, and again replaced, without much loss of time. For underground work in mines the columns are by far the best mounting. Annexed sketch shows the columns that are used.

A shows the clamp which fastens the drill to the columns. B shows a column for use in tunnels or drifts. C shows a plain shaft-bar.

The column for tunnelling, B, is a cylindrical bar on to which a malleable-iron base is shrunk. By means of two jack-screws working through the base the column is lengthened or shortened 12in., and held fast between the roof and floor of a tunnel or the walls of a shaft. It is also used in underhand and overhand stoping. Two machines are often worked on one column.

For tunnelling, drifting, or stoping, a lateral arm, D, is clamped to the column. The drill rests upon and is fastened to the arm by means of the clamp A. The drill can be moved in or out on the arm, and the arm moved up and down or around the column at will by loosening the clamp or back bolts. By this simple arrangement a large portion of the breast is commanded, making the frequent shifting of the column unnecessary, while the drill can readily be placed in any position to drill holes at any angle, up or down or to the right or left. These columns are made any desired length up to 8ft.; if longer there would be too much vibration.

For sinking, the arm is removed and the drill clamped directly to the double-screw column; or column C, with one screw, is often used. To insure rigidity the column should not be more than 8ft. to 10ft. long. In larger spaces the tripod would be best.

The price of these drills, Class B, mounted on tripod, is £65; but if columns are required the price is £72 delivered at Auckland. The following is a descriptive table of the Eclipse rock-drill:—

—	Class and Size of Drill.							
	H.	G.	F.	E.	D.	C.	B.	A.
	In.	In.	In.	In.	In.	In.	In.	In.
Diameter of cylinder	5	$4\frac{1}{4}$	$3\frac{1}{2}$	$3\frac{1}{8}$	3	$2\frac{3}{4}$	$2\frac{1}{2}$	$1\frac{3}{4}$
Length of stroke	7	7	$6\frac{1}{2}$	6	6	5	4	3
Extreme length of drill from end of crank to end of piston	60	60	53	42	40	36	34	26
Diameter of supply-pipe	1	1	1	1	1	$0\frac{3}{4}$	$0\frac{3}{4}$	$0\frac{1}{2}$
Approximate depth drilled in hard medium rock without changing bits	30	30	24	20	20	20	18	12
Diameter of holes drilled, as desired, from.. ..	3 to 6	2 to 4	$1\frac{1}{2}$ to $2\frac{1}{2}$	$1\frac{1}{4}$ to $2\frac{1}{2}$	$1\frac{1}{4}$ to 2	$1\frac{1}{4}$ to 2	1 to $1\frac{1}{2}$	$\frac{5}{8}$ to $1\frac{1}{4}$
Diameter of steel drill used	$1\frac{5}{8}$ & $1\frac{1}{2}$	$1\frac{5}{8}$ & $1\frac{1}{2}$	$1\frac{1}{4}$ & $1\frac{1}{8}$	1 & $\frac{7}{8}$	1 & $\frac{7}{8}$	1 & $\frac{7}{8}$	1	$0\frac{3}{4}$

	Class and Size of Drill.							
	H.	G.	F.	E.	D.	C.	B.	A.
	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.	Lb.
Weight of machine	670	605	345	250	230	195	155	100
Shipping-weight of drill, tripod, and weights complete	1,345	1,280	850	700	600	570	520	120
Approximate weight of blow delivered on rock at each stroke, with 60lb. pressure at drill	1,500	1,000	750	625	550	500	350	200
	Ft.	Ft.	Ft.	Ft.	Ft.	Ft.	Ft.	Ft.
Average depth drilled in ten hours in granite rock, including time lost in setting the drill and changing bits	70	70	70	70	60	60	50	[illegible]
Depth of vertical hole each machine will drill easily	40	30	16	12	10	7	4	2
Depth of horizontal hole each machine will drill easily	30	15	12	10	7	5	3	2
	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.
Best size of boiler to give plenty of steam at high-pressure	12	12	10	10	8	7	5	2

Although these drills can be worked by steam, that force could not be applied for rock-drills employed in underground mining; it could only be used to any advantage in open quarries.

Class H is adapted to submarine work, mounted on a barge or frame; to heavy tunnelling, mounted on tunnel-carriage; to deep rock-cutting, mounted on a tripod. It is made to feed automatically.

Class G is adapted for tunnelling and quarry-work where 15ft. to 30ft. holes are required from 2in. to 4in. in diameter in hard rock. It feeds automatically.

Class F is the most serviceable and useful size for general work. It is adapted to all kinds of rock-excavation except very heavy and very light work. It is principally used in quarries, railroad-tunnelling and cuts, sewer and cellar excavations, and for mining when the space is not too contracted. It is made to feed either automatically or by hand.

Class E is a lighter drill than Class F, and is generally used for the same character of work, where the rock is only of a medium hardness.

Classes D and C are especially adapted for general mining-work, in constructing large adits, or for light quarry-work. It is made to be fed by hand.

Class B.—This is the most useful class of drill for mining purposes, and the class that is generally used in quartz-mines in constructing shafts, adits, uprises, and in stoping out the lodes. It being short and much lighter than any of the others, makes it easily handled and shifted about when not required for work.

Class A is a very light drill, not suitable for general mining-work. It is used principally on a light tripod frame for drilling short holes for plug and feather work in quarries or for any very light work where the depth of the holes does not exceed 2ft. nor the diameter 1¼in.

The same firm—Parke and Lacey—exhibited a diamond-drill which could be used either on the surface or underground. This drill had all the latest improvements, and was of a very superior make. The drill was complete, with steam-engine attached, and cast-iron frame, but had no boiler or connections. The following is a list of the parts that were included in the price of the drill: 400ft. of rods, two extra core-lifters, one wire-rope, four blank bits, along with one bit set with diamonds, eight carbons, all necessary wrenches, &c., and diamond-setting tools. The bits were 2½in. in diameter, and the drill was said to be capable of boring 700ft. It had two steam-cylinders 6in. in diameter, and pressure-indicator showing the pressure on the drill while at work. Its total weight was 1,800lb., and its cost £1,200, delivered at Auckland on shipboard.

Rand Drill Company's Rock-drills.

This company manufactures three different patterns of rock-drills—namely, "The Little Giant," "The Slugger," and "The Economizer."

The Little Giant is a new machine made specially for mining-work. It has all the latest improvements, including split lower head to take up all the wear, and is constructed for either heavy or light work. Every part is made so that it can be taken up when worn, and it has the advantage of being all made of wrought-iron and crucible cast-steel. There is no cast-iron in its construction. Its cost, mounted on an adjustable tripod, is £65 f.o.b. (See Figs. 39 and 40 in annexed sketches.)

The cylinder slides in a shell or guide, which is in turn mounted on a tripod with a patented universal joint. The cylinder is fed towards the rock as fast as the steel penetrates it. The special feature of this drill is the positive valve-movement, which insures certain operation when steam or air is admitted, without depending upon close fits or clean parts. It allows of a variation in design between the up- and down-stroke which is said to economize steam or air, as the case may be, whichever is used, and increases the working-capacity of the machine. The valve is always moved in the same direction as the piston, which is one of the patents in this machine. Indeed, the whole of the parts of this machine are said to be secured by patents.

The greatest advantage this drill appears to possess, without seeing it in use for some time in a mine, is that there is no cast-iron in connection with it, and all the wearing-parts are made large and strong, and are made of hardened steel wherever possible.

The Slugger.—This drill is of a different pattern from the Little Giant. The casing is made of malleable cast-iron. The improvements in this drill consist in the outer shell containing bushes or bearing-blocks made in halves instead of in one piece, so that they can be easily removed when worn out, and replaced by new ones. Stuffing-box and packing are dispensed with. The escape of air or steam, as the case may be, whichever is used, is prevented by an automatic leather ring which is fitted on the upper part of the head. This ring is said to last from three to five months with the drill fully employed. A supply of these rings can be had with the drill. Its price (mounted on an adjustable tripod) is £65 f.o.b.

The manufacturers claim "that this drill is the crowning achievement in rock-boring machinery, as by its use an increase of 25 per cent. in cutting-capacity is realised." They state: "This arises from the fact that it is the first and only machine to introduce rational steam-distribution, and differs from all others in the following particulars: (1.) That it uses the steam or air expansively. (2.) That it strikes an uncushioned blow. (3.) That it is set with a late cut-off to the steam, and thereby realises the greatest drilling-power possible."

The Economizer is somewhat on the same pattern as the Little Giant, and the same price.

These drills have not been used much in the colonies, and nothing can be said of their capabilities beyond the information supplied by the manufacturers. However, they are drills that apparently will do good work, and are strongly made in proportion to their weight.

The columns supplied with these drills are different from those of the Ingersoll Rock-drill Company, as will be seen from the annexed sketch, Fig. 41.

The drills used with both the Rand and Ingersoll Rock-drill Companies have all cross-bits, which do more service and bore holes straighter and more cylindrical than the common bit which is mostly used with the rock-drills in the colonies. These cross-bits are more costly to make, and also more difficult to sharpen, and they require special tools to make them; but they are likely to do far more work, and there is not the same liability of the drills sticking in the holes as there is with the common bit when the machine is at work.

The following is a descriptive table of Rand rock-drills and accessories given by the manufacturers:—

—	Name and Number of Drill.			
	Little Giant No. 2. Slugger No. 12. Economizer No. 22.	Little Giant No. 3. Slugger No. 13. Economizer No. 23.	Little Giant No. 4. Slugger No. 14. Economizer No. 24.	Little Giant No. 5. Slugger No. 15. Economizer No. 25.
Diameter of cylinder—				
Little Giant	$2\frac{3}{4}$in.	$3\frac{1}{8}$in.	$3\frac{5}{8}$in.	$4\frac{1}{2}$in.
Slugger	$2\frac{3}{4}$in.	$3\frac{3}{16}$in.	$3\frac{5}{8}$in.	$4\frac{1}{2}$in.
Economizer	$2\frac{3}{4}$in.	$3\frac{1}{8}$in.	$3\frac{5}{8}$in.	$4\frac{1}{2}$in.
Length of stroke	$6\frac{1}{4}$in.	$6\frac{1}{2}$in.	$7\frac{1}{4}$in.	$7\frac{1}{2}$in.
Usual depth of hole	6ft. to 10ft.	10ft. to 14ft.	20ft. to 30ft.	30ft. to 40ft.
Depth drilled in eight hours—				
Little Giant	50ft.	60ft.	70ft.	70ft.
Slugger	70ft.	80ft.	90ft.	90ft.
Economizer	60ft.	70ft.	70ft.	70ft.
Diameter of hose	$\frac{3}{4}$in.	1in.	$1\frac{1}{4}$in.	$1\frac{1}{2}$in.
Diameter of steel	$1\frac{1}{4}$in. to 1in.	$1\frac{3}{8}$in. to $1\frac{1}{8}$in.	$1\frac{1}{2}$in. to $1\frac{1}{4}$in.	$1\frac{1}{2}$in.
Weight of machine without mounting ...	187lb.	260lb.	402lb.	560lb.
Weight of tripod without weights ...	145lb.	175lb.	382lb.	400lb.
Weight of the weights for tripod ...	286lb.	336lb.	510lb.	510lb.
Weight of column 6ft. high	305lb.	305lb.	350lb.	...
Weight of shaft-bar 8ft. long, with one arm	245lb.	245lb.	250lb.	...
Weight of steel to drill usual depth of hole	52lb.	120lb.	512lb.	930lb.

Sharpening and Tempering Drill-steels.—In making the bit it is necessary to hammer against it at the same time that it is being formed, in order to compress the steel. The steel should always be worked at as low a temperature as possible, and hammered till the colour leaves it. In tempering, the bit is heated to a cherry-red, taking care that this heat does not extend further down the steel than the cutting-edge—altogether from $\frac{1}{2}$in. to $\frac{3}{4}$in. from the extreme edge, according to the size of the steel. It is then plunged into water which has had the chill taken off it—water which feels about tepid to the touch—and the steel is left to cool in the water, when it will be ready for work.

Instructions for Running the Rand Rock-drills.—Set the mounting securely and firmly. If a tripod or quarry-bar is to be used, "spot" a place for each leg with a small hand-drill and hammer. Place the tripod-legs in these holes, and place the weights on the legs. If the column or shaft-bar is to be used, place a piece of planking or blocking between the rock and each end of the column or bar, as shown in the annexed sketch, Fig. 39. This must always be attended to, as the bar will not hold without it. Fasten the column or bar in place by setting out the jack-screws firmly, and with the shaft-bar tighten the lock-nut on screw. The mounting being in place, put the machine on it, and secure it by the bolts provided. Put the shortest drill-steel in the chuck, tightening the chuck-nuts firmly but evenly. Square off the place where the hole is to be, so the steel shall not strike a glancing blow. Draw the piston to the bottom of the cylinder, until it strikes the cylinder-head. Turn the feed-screw until the drill-steel just touches the rock, and then give the screw one more turn. Put oil in the nozzle of the throttle-valve and in the back head by removing the thumbscrew plug. Blow air through the hose to remove any dirt that may have

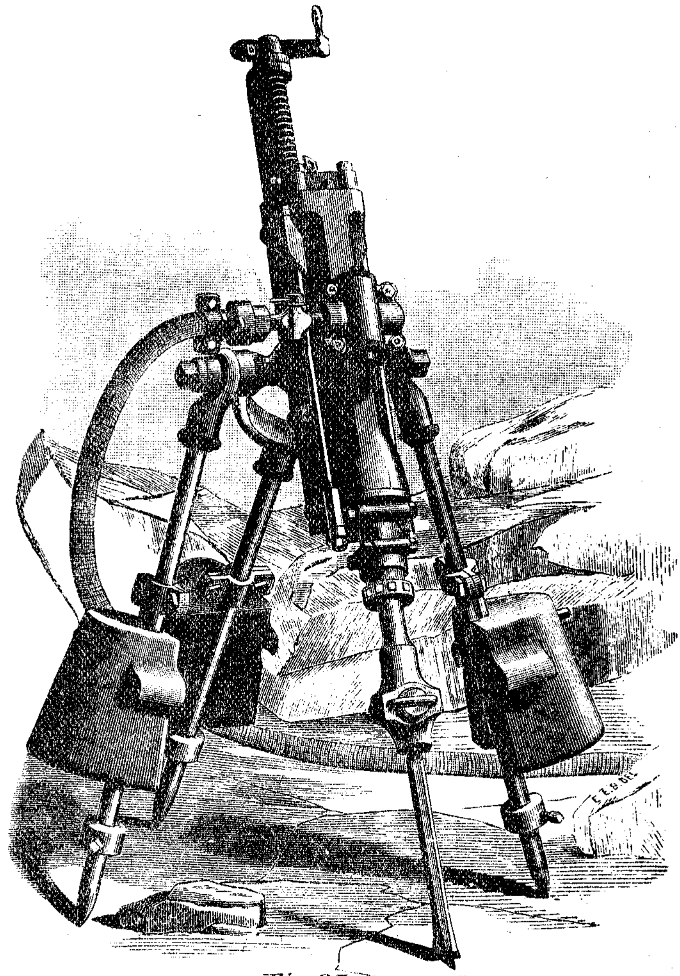
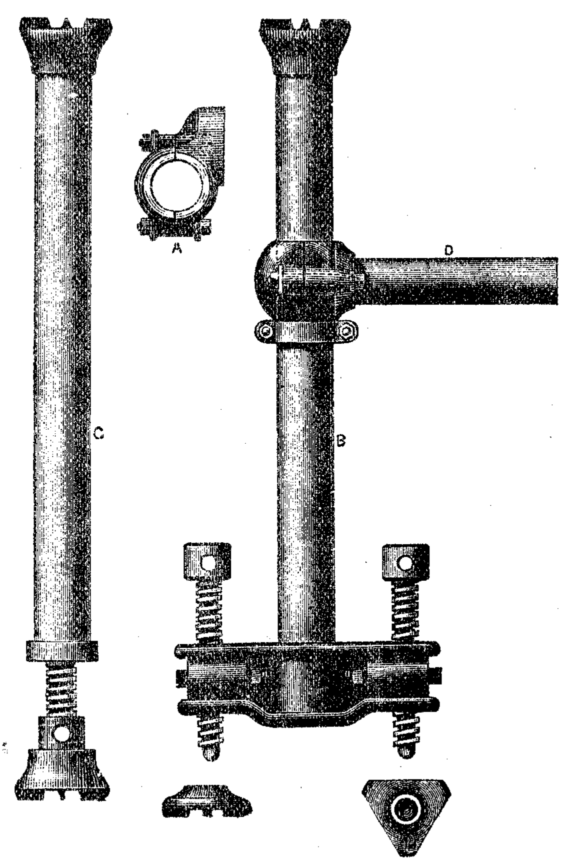


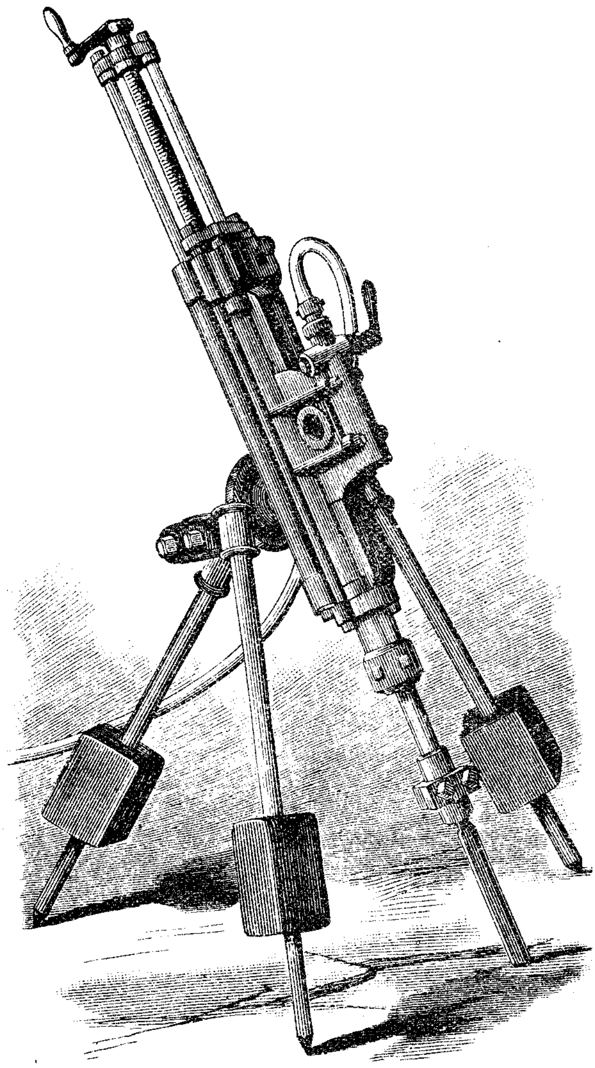
Fig 37.

--MOUNTED ON OUR PATENTED ADJUSTABLE TRIPOD FOR SURFACE WORK, STOPING.



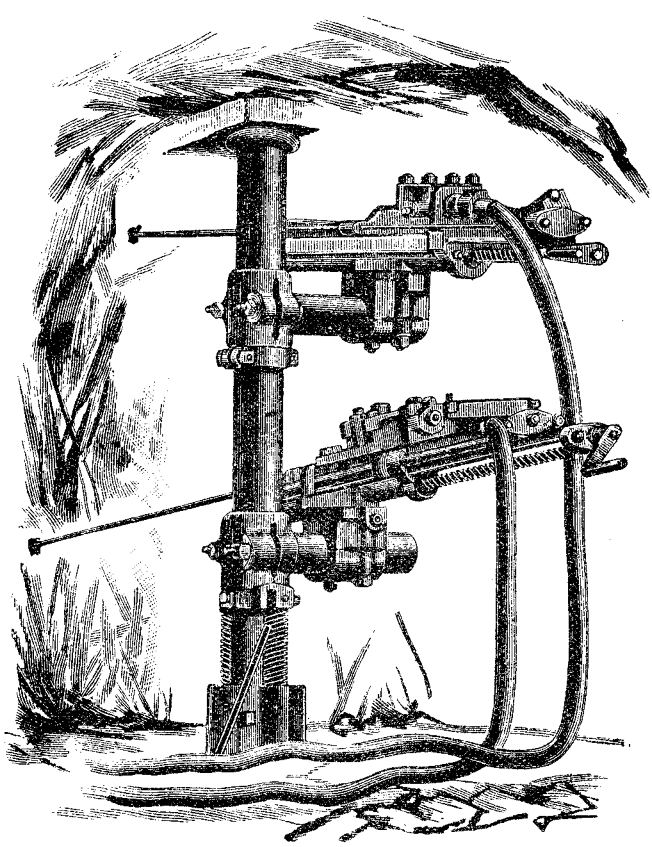
—CUTS REPRESENT TUNNELING AND SHAFT COLUMNS.

Fig 38.



RAND'S LITTLE GIANT ROCK DRILL.

Fig 39.



DRIVING COLUMN.

Fig. 40.

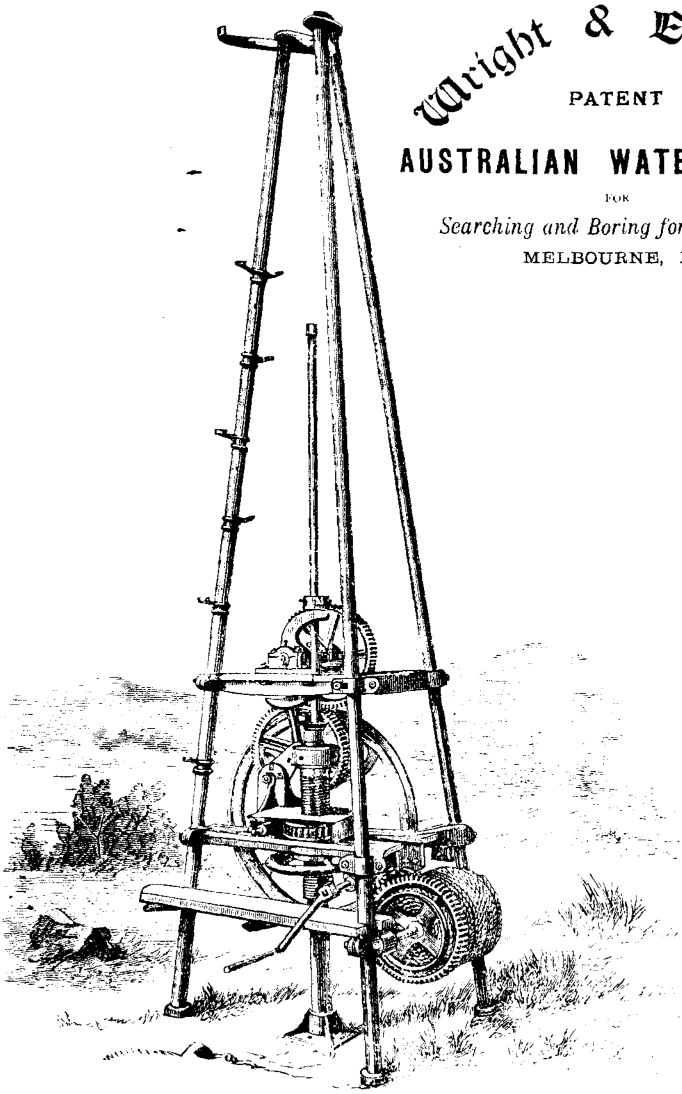
Fig 42.

Wright & Edwards' PATENT AUSTRALIAN WATER AUGER,

FOR

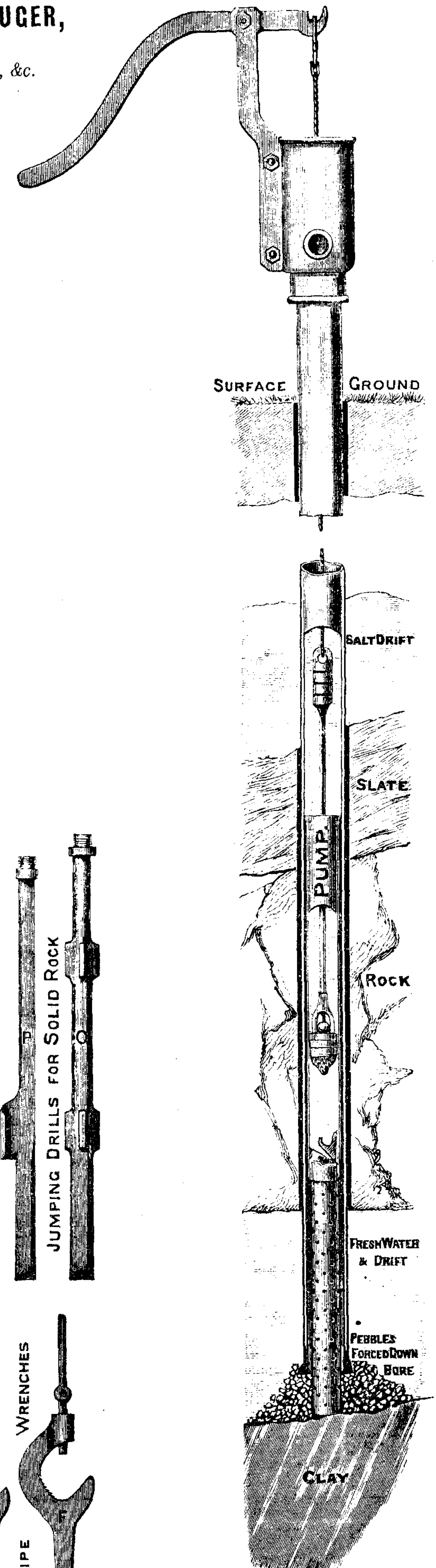
Searching and Boring for Water, &c.

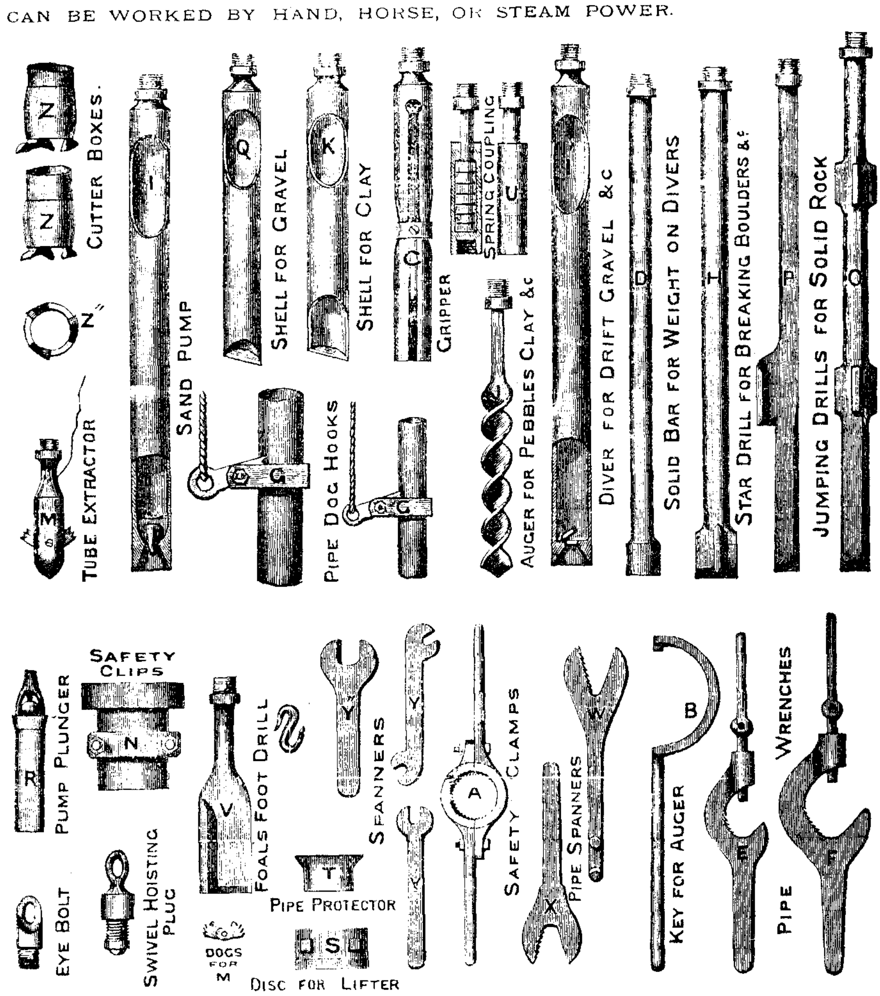
MELBOURNE, 1878.



ROTARY AND PERCUSSION MOTION COMBINED.

CAN BE WORKED BY HAND, HORSE, OR STEAM POWER.





accumulated. Couple the hose to the nozzle of the throttle-valve, being sure that the throttle-valve is closed. The machine is now ready to start. Turn air in the hose and open the throttle-valve half-way. The machine will start and immediately commence cutting the rock, and as it cuts its way into the rock the feed-screw must be turned to advance the machine correspondingly. When the hole is 4in. or 5in. deep turn the throttle-valve wide open and keep the hole half full of water. With some rocks the cuttings will at times form a stiff mud with the water, which will accumulate around the drill-steel and clog it. When this occurs remove the steel from the hole and clean out the hole with one of our "sand-pumps." When the short steel has cut as deep as it will reach, remove it and replace it by the next longer length, and so on. The above instructions apply when running the machines with compressed air. When a new machine is to be started with steam, trouble will be sometimes experienced from the refusal of the machine to start. This arises from the unequal heating of the cylinder, and will disappear as soon as the machine becomes uniformly heated. To hasten the heating and discharge the water of condensation, proceed as follows: Arrange the machine on the tripod and set the drill-point as usual, then loosen the nuts on the long cylinder side bolts so as to leave the cylinder-heads quite loose and slack on the cylinder. Now turn on steam. The water of condensation will run out between the cylinder and head, and presently steam will commence to blow through and warm the machine. Work the piston by hand up and down two or three times, so as to have the steam blow out first at one end and then at the other. Presently, when everything is well heated, the machine will start off all right, when the nuts at first loosened should be gradually tightened. When learning to run a machine, it is advisable to use air or steam at low pressure—say 30lb. per square inch. When accustomed to it the pressure can be increased to 60lb. It is also an excellent plan to run a new machine with a blunt-pointed steel—simply striking the rock without cutting it—until it has got well "limbered up," and the gum of the old oil thoroughly removed. In cold weather do not leave water in the cylinder: unscrew the stuffing-box and let it out. Never strike the piston-rod or chuck. Take care of your machine and use it properly, and it will last for years with few repairs.

The Naylor and Thornton Drill is well known to be one of the best in use in the colonies. It is used extensively in the Reefton district, and gives great satisfaction. It is lighter than any other drill, and therefore more easily handled, but it is more expensive in the first instance, owing chiefly to its being manufactured in the colonies.

Electric Drill.

One of the newest American inventions is a drill worked by electricity, patented by J. E. Story, which if as successful as is claimed by the patentee, will almost revolutionise mining, or any work which has to be done in hard rock. *The Mining and Scientific Press* gives the following description of it:—

"We had the pleasure of watching the drill last night. It is run by a motor which gets its power from electricity furnished by one of the large dynamos. The motor itself is about 8in. long by 7in. in diameter; the armature is about 4in. in diameter. The shaft which the armature runs on is hollow, so that the water can pass through it and through the diamond-drill. The motor works easily on bearers, and as the drill forces itself into the rock it is moved by a screw. At the rear end of the machine a brass tube is put over the hollow shaft, and this is connected with the water-pipe, so that the water is let into the drill as desired. When the drill is at work in a mine the water is forced into the drill by means of a small rotary pump. This is attached to the end of the revolving-shaft. The faster the drill works the faster the water is pumped in; when the drill is standing still no water flows, as it has to come from a tank or pool at some distance from the drill.

"The drill used last night was $1\frac{1}{8}$in. in diameter. By the movement of a lever, similar to that used in turning on incandescent light, the electricity is attached, and the drill begins to whirl and the water to flow. Full pressure was not put on the drill when we saw it, yet in one minute and one second it had bored a hole $1\frac{3}{4}$in. in depth into the very hardest granite found on our hills. The drill, if crowded, can run 2in. a minute. Mr. Story estimates that one of the 400-light incandescent dynamos will run twenty drills easily, each one with a capacity of boring on the average 2in. a minute in the hardest description of rock."

If such a drill prove a success it can be applied for the same purposes as rock-drills, the wires for conveying the electric power being much more convenient to handle and shift about than compressed-air pipes.

WATER-AUGER.

This is a machine that is used to a great extent in the Australian Colonies for boring for water, and could be used with advantage in New Zealand in boring through drift to prospect for gold. It can also be used for boring through rock. Although termed a water-auger it is really a machine which can be used for boring through any strata, and is more useful for boring through schist rock than the diamond-drill. It is suitable for boring in any place where common boring-rods can be used. (See Fig. 42 in annexed sketch.)

The machine is manufactured by Messrs. Wright and Edwards, engineers, Melbourne, chiefly for the purpose of boring for water through gravel-drift, clay, sand, and rock. When the machine was first made considerable difficulty was experienced by the makers in boring successfully through different strata met with in going down. In soil free from stones these machines were fairly successful; but as obstructions frequently existed, and had to be overcome before water was reached, something more was necessary to complete the machine. Such was the experience of the makers when attempting to sink a tube-well near the site of the new Law-courts in Melbourne. The rock inclined at a steep angle, with soft and hard seams, which caused the pipes or tube to be thrown to one side, when they were soon bent and rendered useless. The idea then suggested itself to the makers that a tube might be constructed to bore or cut its way through the rock and any other strata if the bottom of the tube were constructed with a cutting action and revolving motion. After a considerable amount of experimenting a machine was constructed to overcome all difficulties, and it is now perfected, so that any one with a fair amount of intelligence can work it satisfactorily.

A tripod of tubing is set up over the place where boring operations are to be carried on, and the machinery fixed on the tripod about 3ft. above the surface of the ground, as will be easily understood from the annexed sketch. The auger is made to revolve by means of a large worm-screw on the outside of the auger-case driven by a small worm or spiral on the axle of the fly-wheel, which is turned by hand. The other handle is used in lowering the cleaner to the bottom of the tube and in lifting it to remove the loose material from the boring. The cutting portion of the tubing or auger is represented in the annexed sketch marked Z. It consists of three steel-cutters kept in their places by a band or ferrule, and these are capable of going through granite or any harder stratum. The pipe or tube is 4in. in diameter, consequently anything smaller than this is brought up whole by the cleaner when boring through solid rock. In boring through bluestone or any hard compact substance a core 4in. in diameter is lifted up by the cleaner.

The rate of motion may be regulated at pleasure, according to the nature of the stratum bored through, but when worked at the slowest speed the power exerted is forty times that of the person employed at the machine. The tubes are made in convenient lengths, and are screwed and coupled together as each length is added. If water is not struck, the whole of the tubing can be lifted and removed to another place, and used again; but if water is found, the soil and stones are removed by means of the cleaner, and the working-barrel of the pump, with bucket, leather foot-valve, &c., is then lowered, and pumping commenced, the working-barrel being kept in position by the pressure. It may appear that the revolution of the tubing—the whole length of the pipes in the ground, which is the driving-shaft of the auger—would involve a considerable power; but this is not found to be the case: a boy can work it, and to a man the work is easy. In going through drift-sand a great deal more power is required to work it than when boring in solid strata. In order to deal effectively with any strata six different tools are sent with each machine. The tools are changed, to suit the strata to be gone through, without drawing the tubes, the tools being merely for extracting the core, and can be changed at pleasure in a few minutes.

When solid rock has to be gone through there are rods and drills attached. These have a rotary and percussive action, and are lifted by means of a disc fitted on to the rods with set-screws, and a revolving shaft on which there is a double cam similar to that used for lifting and revolving stamps, as will be seen by reference to the annexed sketch.

From several bores that have been put down between Wilcannia and Broken Hill the details of boring have been given in the mining records of New South Wales. Depths have been reached with this auger over 400ft. below the surface. In this district a great deal of the boring was done in soft rock. The usefulness of this machine in New Zealand would be that it could be used to test wet gravel-drifts and sand-leads for gold, as well as boring through rock for any other purpose. In boring through sand and clay, W. Lucas, manager for J. Kitchen and Sons, Melbourne, states, in reference to boring a well, that on commencing to bore a bed of coarse sand 19ft. thick, full of water for 16ft., was first met with; then a bed of hard, dry clay for 45ft. The depth of 64ft. was reached in seventeen hours' work. A second layer of fine drift-sand was got below the bed of clay, this drift coming up through the auger along with water, with considerable force, sending the water 2ft. above the surface of the ground. On proceeding 11ft. further clay was again struck, and the cores of the auger became dry as before; this bed proved 10ft. thick, when 2ft. 6in. of coarse gravel and pebbles was met with, which rested on the solid rock. Also, a bore was made at Laen by a party employed by the Government. At a depth of 250ft. the bore passed through a tree for a distance of 6ft. This shows that it is a machine capable of boring through any substance met with. The following are the makers' instructions with regard to fixing and working the machine, and also the method of fixing the pump if the bore is required for a well, together with the cost of the machine and accessories:—

"*To fix the Machine.*—The position of bore having been determined upon, and the surface of the ground levelled, the guard-ring L is sunk in the ground, or secured to timber, and puddled round to prevent any surface-drainage going down the bore. The tripod stand, having been fixed together according to the marks and numbers, is now placed with the hollow screw in line with the centre of hole in the guard-ring; a timber sleeper may be placed under each leg of tripod, and the base or shoe secured to it, the rope of the crab-winch rove over the sheave and the swivel hoisting-plug properly fastened at the end, and the machine oiled, special care being taken that the machine is perfectly plumb through cap of tripod and centre screw.

"*To commence Boring.*—Place one of the 7ft. lengths of auger-tube, with coupling end up, through the hollow screw, and screw cutter-head on to bottom end of same. Tallow should be placed on all the screws, and machine well oiled, care being taken that the tray where worm is running in is kept filled with oil. Raise the hollow screw by means of the hand-wheel under table; tighten the three dogs that secure the tube by means of the key B, which tightens the three at once; turn the fly-wheel in the direction shown by arrow on same, so that the auger-joints screw tighter in driving, and the auger will commence to rotate. The rate of its descent is regulated by means of the hand-wheel, and if the hand-wheel is held fast the auger will sink ½in. in each revolution; if allowed to revolve with the screw no descent will take place, so any intermediate rate can be obtained by slightly checking it, while in soft drifts the auger can be forced down at a great rate by turning the hand-wheel in the contrary direction to the screw. When the auger has bored to the depth of the screw the dogs are released, and the hollow screw raised as before for another cut. The 3ft. 6in. length of auger is screwed on the end of the first length when it descends below the dogs; this is removed, and a 7ft. length substituted for it when it has sunk 3ft. 6in. The object of this is to have the top end of auger a convenient height for extracting the cores. This process is repeated until the requisite depth is reached.

"*To extract Cores.*—As the auger bores down into the earth the core which is formed in the inside requires extracting continuously by means of core-gripper C, attached to solid bar D, which is then screwed to swivel on end of rope. (This tool is applicable to all solid strata.) This gripper on being lowered quickly grasps the core, which, striking the bolt, raises the belt and causes the tool to grip the core, which can then be raised by winch. For sand-drifts, gravel, sludge, and other liquid strata, use tools as per diagram applied as above. In the event of loose boulders being met with, couple solid bar D to star-drill H, and smash same, or drive it aside.

"*When water is struck,* and the core cleaned from the inside of auger, special care being taken to cleanse the tubes thoroughly from sand and drift by the use of pump, the pump-barrel, with outside leathers, and the suction-valve and perforated pipe screwed on, is let down the auger; the sand-bucket (with brass rings) is lowered attached to the rope, when the supply can be tested, the rope being worked by hand. The water at once secures the pump-lining, and rises in the auger. Should the strain on the rope show that the bucket is drawing, it is a sign that the supply is not sufficient, and the strokes should be slower for a time to allow the water to wash a passage. If the water has been struck in a sand-drift the bucket and pump-casing may have to be repeatedly withdrawn. The bucket is simply hoisted up, and the tool M, attached to the bar D, is lowered until the point rests on the bottom valve, which it will open, and the water flushed back will clear the auger. On hoisting this tool the small side dogs fasten on the inside of the pump-barrel, which can thus be hoisted up, and the drift at the bottom of the auger cleared out by the tool I, the pump-barrel again inserted, and pumping continued until sufficient sand is exhausted to allow the supply to come freer; the barrel should again be removed, and fine gravel forced down the auger by the drill to form a filter-bed. For this reason shallow drifts should be passed through, so that a firm bed is reached, and the auger backed about 6in. to let the gravel spread. Some pebbles should be placed above the gravel for the perforated pipe to rest upon, which should have horse-hair cloths or flannel tied around it to keep out sand. In the finer drifts, should the supply diminish at any time through choking the gravel filter-bed, the bucket can be withdrawn and the tool M lowered, with the side dogs removed, and on striking and raising the bottom valve the filter-bed can be flushed clear again. This is one of the most valuable features of this invention, and overcomes the cause of the failure of so many wells which choke with sand-drifts. If necessary, water can be forced down the auger from the surface while the valve is open, to force back the finer drifts and assist the flow. When the permanent power, if manual, horse, wind, or steam, for working the pump is fixed, the leather bucket sent should be used attached to the steel-wire rope, which simply wants a vertical movement of about 2ft. When a leather on the bucket requires renewing, the rope and bucket can be hauled up and the leather changed, and lowered in a few minutes without a single joint or rod having to be loosened.

"*To draw the Tubes.*—Should water not be procured, the auger can be withdrawn for removal to fresh site by the use of the extractor M, which is coupled to end of drill-rods and lowered down 4in. tubes until the middle of bottom pipe is reached, care being taken to keep the joints of rods 6in. above those of auger-tubes, otherwise serious trouble will arise in uncoupling. Then couple end of drill-rods to swivel hoisting-plug attached to blocks (chain or pulley-blocks, or any other purchase in use). Whilst lifting, the main screw may be utilised as a jack by supporting under side of table close to screw with wood blocks and affixing clips N to 4in. tube, and using dog end of spanner W for increased leverage, giving at the same time a slight forward motion with the fly-wheel to facilitate free withdrawal.

"*Fixing and working Jumping-motion.*—Fix the top table securely by clips, as per marks stencilled on same, to legs of derrick; fit wheels on bracket of lower table, and gear, same as per marks thereon, to the wheel that is used on worm-shaft necessary to the required speed of cam-shaft, according to the nature of stratum that is being drilled. This done, couple to drill the 7ft. solid bar and the spring coupling-box, which is intended to relieve the jar on rods, and then attach drill-rods according to the depth required, bolting to same the disc (which is in halves) securely, with the flange downwards, regulating same according to the desired lift or weight of blow necessary to penetrate the stratum under operation.

"NOTE.—If using the eccentric drills, it will be necessary to observe the following: Couple, as above mentioned, the drill O and drill to a depth of 10in., then withdraw drill and raise tubes at least 2ft., and suspend same by clips supplied; then substitute for the drill O the eccentric drill P, lower same carefully through the cutter-box into bore made by previous drill, and proceed by turning the fly-wheel in the opposite direction to that in which you would whilst boring, taking care to keep the main screw run down to head, and leave the pipe-chuck slack.

"*Cutters.*—Two sizes are supplied with each machine. Use the larger size when commencing to bore, to give more clearance in the descent of tubes when passing through expanding clays. When renewing cutters use the smaller size, allowance being thus made for wear-and-tear of larger ones.

"In boring, keep correct measurements of the auger and depth of each stratum, so that the length of auger is always known from the top of the guard-ring or machine. Keep the machine clean and well oiled. In deep bores, to save time in hoisting cores, the rope can be run over the brake-wheel of winch and drawn by a horse."

The following is the price-list of Wright and Edwards's Patent Improved Water-auger Boring-machine (22nd September, 1884):—

	£	s.	d.
Complete apparatus for working the auger, and which can be used for any number of wells	150	0	0
A.—Safety clamps, each	3	10	0
B.—Key for auger, included in price of machine.			
C.—Core-gripper for solid cores	2	10	0
D.—Two solid bars, required for weighting sand-pumps and grippers (each £1 10s.) ..	3	0	0
E and F.—Two pipe-wrenches (each £2 10s.)	5	0	0
G and G'.—Two pipe dog-hooks (each £2) ..	4	0	0
H.—Star-drills for breaking boulders, each ..	2	10	0
I.—One diver, for water and sand	2	10	0
I'.—One slush-pump with flap-valve ..	2	10	0
I'.—One gravel-pump with ball-valve ..	2	10	0
J.—One twist-auger	2	10	0
K.—One shell-auger	2	10	0

	£	s.	d.
M.—Tube-extractor, with extra dogs and relieving-wire	3	10	0
N.—Safety clips	2	10	0
O and P.—One pair of patent eccentric rock-drills	10	0	0
Q.—Shell for gravel	2	10	0
R.—Pump-head, lever, barrel and pump complete, with spare bucket and rose ..	10	0	0
S.—Disc for lifters, included with machine.			
T.—Tube-protector	0	12	6
U.—Patent spring coupling	2	10	0
V.—Foal's-foot drill	3	10	0
W and X.—Two pipe-spanners (each £1) ..	2	0	0

	£	s.	d.
Y.—Spanners, included in price of machine.			
Z.—Cutter-box and cutters complete ..	5	0	0
Twelve extra cutters (each 10s.)	6	0	0
Hoisting-plug, £2 10s.; eye-bolt, 10s. ..	3	0	0
200ft. of 4in. auger-tubes $\frac{1}{4}$in. thick and coupled with steel (6s. 6d.)	65	0	0
200ft. of $1\frac{1}{2}$in. tubes or clearing-rods (2s. 6d.) ..	25	0	0
200ft. of Manilla rope	1	0	0
200ft. of steel rope for pump	1	0	0
Total cost and weight of machine and all tools and tubes for 200ft. bore: Weight, $2\frac{1}{2}$ tons ..	£327	2	6

For surface-boring this drill will be useful for boring in any hilly, broken country, such as New Zealand, where the several pieces can be conveyed on packhorses to the place where operations are proposed to be carried on, and, as their action is somewhat similar to that of diamond drills by leaving a core inside the tube, when this core is lifted the nature of the strata gone through can be clearly ascertained. It is a machine that recommends itself, and can be worked by any intelligent labourer.

STEAM-BOILERS.

The Babcock and Wilcox Boiler.

The principle on which this boiler is constructed is calculated to meet the requirements of a perfect steam-boiler for land purposes—namely,—

(1.) That it should be simple in construction, and made of the best of materials.

(2.) It should have a constant and thorough circulation of water throughout the boiler, so as to maintain all parts at an equal temperature.

(3.) It should have a combustion-chamber so arranged that the combustion of the gases commenced in the furnace may be completed before they escape to the chimney.

(4.) That all parts should be perfectly accessible for cleaning and repairs.

(5.) That the heating-surface of a steam-boiler should be arranged as nearly as possible at right angles to the current of heated gases, and so break up the current as to extract the available heat therefrom.

(6.) The boiler should have a large water-surface for the disengagement of the steam, in order to prevent foaming or priming.

(7.) A boiler should have large and free passages between the different sections, to equalise the water-line and the pressure in all.

(8.) A boiler should have no joints exposed to the action of the fire.

(9.) It should have a great excess of strength over any legitimate strain, and should be constructed so as not to be liable to be strained by unequal expansion.

(10.) The water-space should be so arranged and divided into sections that should any section give way no general explosion can occur, but the destructive effects be confined to the simple escape of the contents.

(11.) It should be proportioned for the work to be done, and be capable of working to its full rated capacity with the highest economy.

(12.) It should have a large heating-surface in proportion to the quantity of water used, so as to produce the maximum effect in the saving of fuel.

(13.) All the inside portions should be easily cleaned out, and there should be a mud-chamber at the bottom, whence the mud and other solid substances can easily be removed.

(14.) All boilers should be provided with the very best gauges and safety-valves, and also provided with the most approved appliances for injecting feed-water at a high temperature.

The Babcock and Wilcox boilers are constructed so as to combine the above-mentioned requirements, and for stationary land boilers they are superior to any other I have seen where fuel is a consideration. They are eminently suited for generating steam for engines employed for reduction of mineral ores, or for driving the machinery connected with manufactories of all articles where coal and fuel becomes a large item in the expenditure. They would not apply with the same force to engines used about collieries, where coal is of very little consideration. They, however, require constant attention, and careful watching to see that the water is kept up to something like a uniform height in the boiler. The small amount of water-space in proportion to the large heating-surface necessarily involves this attention, and requires careful and trustworthy men to be employed; but even any accident that may occur through carelessness can only affect one particular section, and will not injure the whole boiler should any portion of the boiler burst, and this in itself is a great recommendation in favour of the Babcock and Wilcox boiler. (See Figs. 43 to 46 in annexed sketch.)

Description of Boiler.—The boiler is composed of lap-welded wrought-iron tubes, placed in an inclined position, and connected with each other and with a horizontal steam-and-water drum by vertical passages at each end, while a mud-drum connects the tubes at the rear at the lowest point of the boiler.

The end-connections are in one piece for each vertical row of tubes, and are of such form that the tubes are in a zigzag position—each horizontal row covers over the spaces of the previous row. The holes are accurately sized, made tapering, and the tubes fixed therein by an expander. The sections thus formed are connected with the steam-and-water drum and also with the mud-drum, the latter being connected by short tubes expanded into bored holes, thus doing away with all bolts, and leaving a clear passage between the several parts. The openings for cleaning opposite the end of each tube are closed by hand-hole plates, the joints of which are made by milling the surfaces to accurate metallic contact, and are held in place by wrought-iron clamps and bolts.

THE BABCOCK & WILCOX

WATER-TUBE BOILER.

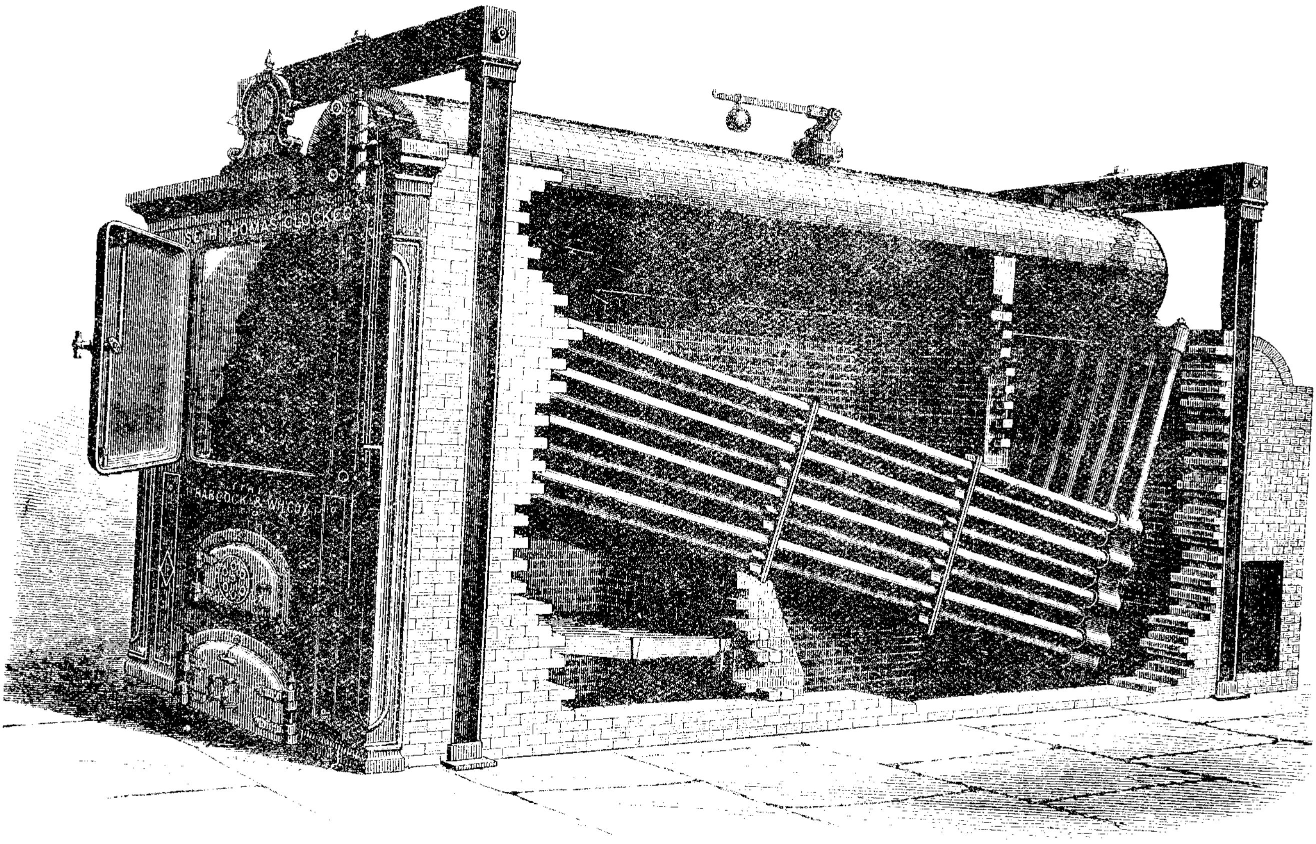


Fig 43.

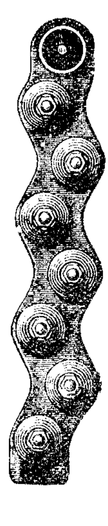


Fig. 46.

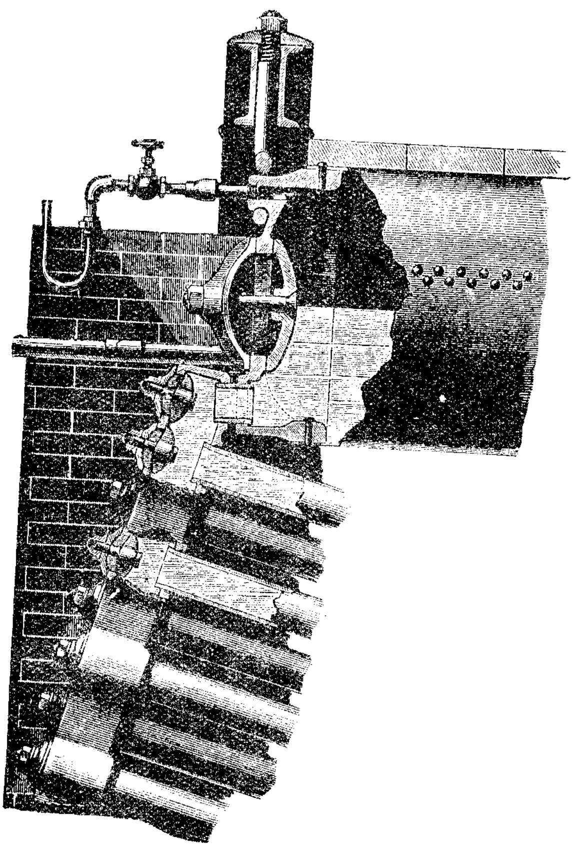
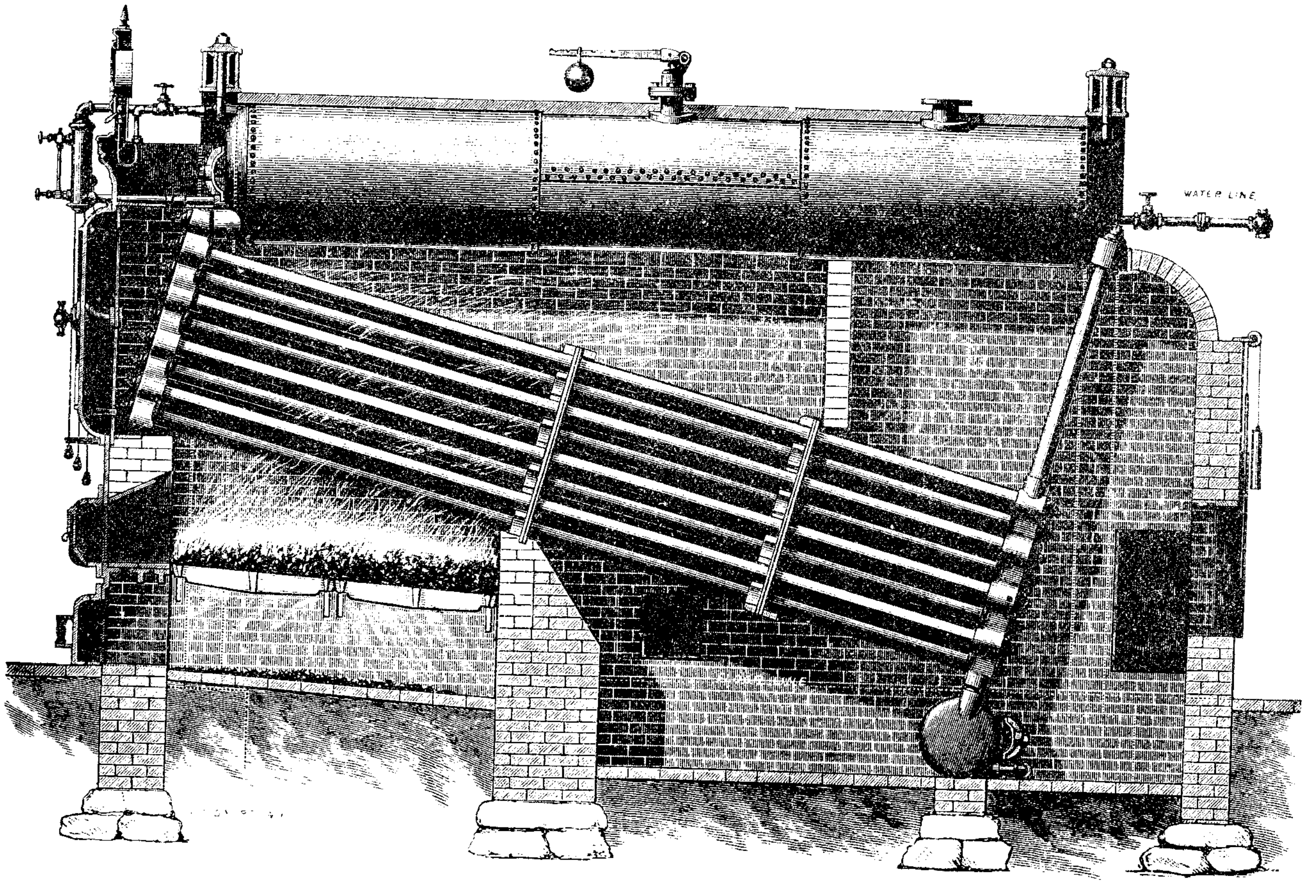


Fig 45.



SIDE ELEVATION.

Fig. 44.

They are tested and made tight under a hydrostatic pressure of 300lb. to the square inch, no india-rubber packing or other perishable substances being used.

The steam-and-water drums are made of flange iron or steel, of extra thickness and double riveted. They can be made for any desired working-pressure, but the agent informed me that they are always tested to 150lb. per square inch unless otherwise ordered.

The mud-drums are made of specially-prepared cast-iron, as being the best material to withstand corrosion, and are provided with ample means of cleaning. Annexed are elevational sections of these boilers, shown in Figs. 43 and 44.

Fig. 45 shows the vertical passages at each end into which the tubes are placed.

Fig. 46 shows the connection of the passages, with tubes placed in, and the steam-and-water drum of the boiler, and also portion of the brickwork in front.

Erection of Boiler.—In erecting this boiler it is suspended, entirely independently of the brick-work or fire-front, from wrought-iron girders resting on iron columns. This is done to avoid any straining of the boiler from unequal expansion between it and its enclosing walls, and permits the brickwork to be repaired or removed if necessary without in any way disturbing the boiler.

The makers describe the operation in working these boilers as follows: The fire is made under the front and higher end of the tubes, and the products of combustion pass up between the tubes into a combustion-chamber under the steam-and-water drum; hence they pass down between the tubes, then once more pass up through the spaces between the tubes and off to the chimney.

The water inside the tubes, as it is heated, tends to rise towards the higher end, and as it is converted into steam the mingled columns of steam and water, being of less specific gravity than the solid water at the back end of the boiler, rise through the vertical passages into the drum above the tubes, where the steam separates from the water, and the latter flows back to the rear and down again through the tubes in a continuous circulation. As the passages are all large and free this circulation is very rapid, sweeping away the steam as fast as it is formed, and supplying its place with water, absorbing the heat of the fire to the best advantage, causing a thorough commingling of water throughout the boiler, and a consequent equal temperature, and preventing to a great degree the formation of deposits or incrustation upon the heating-surfaces, sweeping them away and depositing them in the mud-drum, whence they are blown out.

The steam is taken out at the top of the steam-drum, near the back end of the boiler, after it has been thoroughly separated from the water.

Advantages claimed by the Makers.

1. *Thin Heating-surface in Boiler.*—The thick plates necessarily used in ordinary boilers in the furnace, or immediately exposed to the fire, not only hinder the transmission of heat to to the water, but admit of overheating and burning the side next the fire, with consequent strains resulting in the loss of strength, cracks, and tendency to rupture. Water-tubes, however, admit of thin envelopes for the water next the fire, with such ready transmission of heat that even the fiercest fire cannot overheat or injure the surface.

2. *Joints removed from the Fire.*—Riveted joints, with their consequent double thickness of metal, in parts exposed to the fire give rise to serious difficulties. Being the weakest parts of the structure, they concentrate upon themselves all strains of unequal expansion, giving rise to frequent leaks and occasionally to actual rupture. The joints between the tubes and the tube-sheets also give much trouble when exposed to the direct fire, as in locomotive and tubular boilers. These difficulties are wholly overcome by the use of the lap-welded water-tubes, with their joints removed from the fire.

3. *Large Draught-area.*—This, which is limited in fire-tubes to the actual area of the tubes, in this boiler is the whole chamber within which the tubes are enclosed, giving ample time in the passage of the heated gases to the chimney for thorough absorption of their heat.

4. *Complete Combustion.*—The perfection of combustion depends upon a thorough mixture of the gases evolved from the burning fuel with a proper quantity of atmospheric air; but this perfect mixture rarely occurs in ordinary furnaces, as is proved by chemical analysis, and also by the escape of smoke upon the introduction of any smoke-producing fuel. Even when smoke is not visible a large percentage of combustible gases may be escaping into the chimney in the form of carbonic oxide or half-burnt carbon. Numerous attempts have been made to cure this evil by admitting air into the furnace or flues to burn the smoke; but, though this may allow so much air to mingle with the smoke, as to render it invisible, and at the same time ignite some of the lighter gases, it in reality does little to promote combustion, and the cooling effect of the air more than overbalances all the advantages resulting from the burning gas. The analysis of gases from various furnaces shows almost uniformly an excess of free oxygen, proving that sufficient air is admitted into the furnace, and that a more thorough and perfect mixing is needed. Every particle of gas evolved from fuel should have its equivalent of oxygen, and must find it while hot enough to combine, in order to be effective. In this boiler the currents of gases after leaving the furnace are broken up and mingled together as they pass between the zigzag tubes, and they also have an opportunity to complete their combustion in the triangular chamber between the tubes and the drum.

The makers of these boilers claim that complete combustion is effected in them. This was tested by Dr. Behr in analysing the escaping gases from a stack of these boilers erected at Mattheissen and Weicher's sugar-refinery, he having made many separate analyses at different times; and in no case was there more than a trace of carbonic oxide, even when there was less than 1 per cent. of uncombined oxygen.

5. *Thorough Absorption of the Heat.*—There are important advantages gained in this respect in consequence of the course of the gases being more nearly at right angles to the heating-surface, impinging thereon instead of gliding by in parallel lines, as in the fire-tube boilers. The currents,

passing three times between the zigzag tubes, are brought intimately in contact with all parts of the heating-surface, rendering it much more efficient than the same area in ordinary tubular boilers.

The experiments of Dr. Alban, of the United States navy, have proved that a given surface arranged in that manner is 30 per cent. more efficacious than when in the form of fire-tubes as usually employed.

6. *Efficient Circulation of Water.*—As all the water in the boiler tends to circulate in one direction the steam is carried quickly to the surface, and all parts of the boiler are kept at nearly an equal temperature, thus preventing unequal strains, and by the rapid sweeping current the tendency to deposit sediment on the heating-surface is materially lessened.

7. *Quick steaming.*—The water being divided into many small streams in thin envelopes passing through the hottest part of the furnace, steam may be rapidly raised in starting, and sudden demands on the boiler may be met by a quickly-increased efficiency.

8. *Dryness of Steam.*—The large disengaging-surface of water in the drum, together with the fact that the steam is delivered at one end and taken out at the other, secures a thorough separation of the steam from the water, even when the boiler is forced to its utmost. Most tubular or locomotive boilers make wet steam, "priming" or "foaming," as it is termed, and in many "superheating-surface" is provided to "dry" the steam; but such surface is always a source of trouble, and is incapable of being graduated to the varying requirements of the steam. Hence a boiler which makes dry steam is to be preferred to one that dries steam which has been made wet.

9. *Steadiness of Water-level.*—The large area of surface on the water-line and the ample passages for circulation secure a steadiness of water-level not surpassed by any boiler.

10. *Freedom for Expansion.*—The triangular arrangements of the parts, forming a flexible structure, allows any member to expand without straining any other; the expanded connections being also amply elastic to meet all necessities of this kind. The rapid circulation of the water, however, in this boiler, by keeping all the parts at the same temperature, prevents to a large extent unequal expansion.

11. *Safety from Explosions.*—The freedom from unequal expansion avoids the most frequent cause of explosions, while the division of water into small masses prevents serious destructive effects in case of accidental rupture. The comparative small diameter of the parts secures, even with thinness of surface, great excess of strength over any pressure which it is desirable to use. So powerful is the circulation of the water that no part will be uncovered to the fire until the quantity of water in the boiler is so far reduced that if overheating should occur no explosion could result.

12. *Durability.*—Besides the important increase of durability due to the absence of deteriorating strains and of thick plates and joints in the fire, there is no portion of the boiler exposed to the abrasive action which so rapidly destroys the ends of the fire-tubes, or to the blowpipe action of the flame upon the crown of the sheet, bridge-walls, and tube-sheets or plates, which is so destructive frequently to ordinary, particularly to locomotive, boilers. Neither is there any portion of the surface above the water-level exposed to the fire. For these reasons these boilers are considered durable and less liable to repairs than other boilers under the same circumstances and having the same care.

13. *Accessibility for Cleaning.*—This is of the greatest importance, and is secured to the fullest extent. Hand-holes, with metal joints, opposite each end of each tube permit access thereto for cleaning; and a man-hole in the steam-and-water drum, and hand-holes in the mud-drum, are provided for the same purpose. All portions of both the exterior and interior surface are fully accessible for cleaning. The occasional use of steam, through a blowing-pipe attached to a rubber hose, operated through doors in the side walls, will keep the tubes free from soot, and in condition to receive the heat to the best advantage.

14. *Least Loss of Effect from Dust.*—The ordinary fire-tube or flue, receiving the dust from the fire on the interior, is quickly covered for about one-third of its surface: the water-tube, however, retains but a small quantity on its upper side owing to its circular shape, and beyond this quantity no more can be deposited; whereas with the fire passing through the inside of the tube, as in ordinary tubular and locomotive boilers, the tube would in time get completely filled with dust if not regularly cleaned out.

15. *Ease of Transportation.*—Being made in sections, which are readily put together with a simple expanding-mandrel, these boilers may be easily and cheaply transported where it would be impossible to place a boiler of the ordinary type, unless by riveting it on the ground. These boilers can be made, if required, in sections capable of being packed on horses to the place of erection.

16. *Repairs.*—As now constructed, these boilers, the maker states, seldom require any repairs; but should from any cause repairs be required, any good mechanic can execute them with tools found in any boiler-shop. Should a tube require to be renewed, it can be removed and a new one substituted, the same as in a tubular boiler.

17. *Capacity.*—This is a point of the greatest importance, and upon it depends in a large measure the satisfactory performance of any boiler in several particulars. Unless sufficient steam-and-water capacity is provided there will not be regularity of action—the steam-pressure will suddenly rise and as suddenly fall, and the water-level will be subject to rapid changes; and if the steam is drawn rapidly from the boiler, or the boiler crowded, wet steam will be the result.

The makers of this boiler claim that its proportions have been adopted after numerous experiments with boilers of varying capacity, and that experience has established that this boiler can be driven to the utmost, carrying a steady water-level and steam-pressure, and always furnishing dry steam.

The cubical capacity per horse-power is equal to that of the best tubular boilers of the ordinary construction. The fire-surface being of the most effective character, these boilers will, with good fuel and a reasonably economical engine, greatly exceed their rated power; but it is not economy to work any boiler beyond its nominal power. The space occupied by this boiler in setting is equal to about two-thirds that of the same power in tubular boilers.

Annexed are sketches of boilers showing the general arrangement.

The following are the prices of the different-sized boilers, delivered f.o.b. at Glasgow, together with the number of bricks required in setting them: Forty-five horse-power boiler, weight 7 tons, price £211; common bricks required for setting, 9,000; fire-bricks, 1,500. Sixty-three horse-power boiler, weight 9 tons, price £255; common bricks required in setting, 12,000; fire-bricks, 1,700. Eighty-four horse-power boiler, weight 11 tons, price £306; common bricks required in setting, 13,000; fire-bricks, 1,700. The smallest boiler that Babcock and Wilcox manufacture is forty-five horse-power.

Walther and Co.'s Water-tube Boilers.

These boilers are on the same principle as those of Babcock and Wilcox, but the latter boiler has a different arrangement for the fire passing through among the tubes, and also in the tube-plates. This company's works are at Kalk, near Cologne, Germany, and they manufacture three different descriptions of water-tube boilers—namely (see Figs. 47 to 52 in annexed sketches): (1) The Root system, with nothing but tubes; (2) the Root system improved, with steam-chest on top; (3) the Petry-Walther system, with wrought-iron tubular water-chambers.

Description of Boilers.

Class I.—The water-tubes, which are 4in. in diameter, run in alternate rows above one another, and are divided into vertical zigzag rows, the tubes of each row being connected with one another by means of heads and caps at each end, so that a connecting-chamber is formed running up at each end, into which the tubes open out.

Fig. 47 annexed shows a front view and Fig. 48 a longitudinal section of one of these boilers. The feed takes place at the bottom back end of the boiler, marked S, and the water rises through the zigzag chambers of the back end in proportion as the evaporation takes place. The steam produced rises through the zigzag chambers of the front end into the steam-drum D, where it is said by the manufacturers to arrive free from water, owing to its having followed such a tortuous course, and having been superheated in the upper tubes, in which there is no water.

The connections at the ends of the tubes are made of cast-iron, and are tested up to a pressure of 1,500lb. per square inch, or to about ten times the ordinary working-pressure.

These boilers can be transported over extremely rough country on pack-horses, the heaviest part being only about 300lb.; and from the way in which the connections are made almost any handy man can put them together. The tube-end connections are shown in Figs. 49, 50, 51, and 52. They are fitted on to the tube-heads with conical metallic packing-rings. The ends of the caps and the corresponding openings in the tube-heads are bored out slightly conical, and between them are placed iron biconical packing-rings (see Figs. 49 and 52). This metallic packing is said to have given most satisfactory results in every way and under all conditions, and it allows the connecting-caps to be removed without having to renew the packing.

All the connecting-caps are curved, and can be examined and cleaned from either end. The connection of the back ends of the bottom row of tubes with the mud-drum permits of any tube of the bottom row being taken out singly without disturbing any of the tubes in the row above; and the connections with the steam-drum are equally simple.

A feed-water heater is supplied with these boilers, but is charged for as an extra. This heater is placed above the system of tubes, so that the waste products of combustion may circulate around it before passing out into the chimney. The valves V in the feed-pipe (see Fig. 48) are arranged so that the feed may go through the heater or straight into the mud-drum.

The fire from the furnace strikes against a metal plate which rests on the centre row of tubes in the boiler, and returns at the end and passes through again amongst the upper line of tubes, and again returns into flue leading into the furnace, so that the heating-surface of this description of boilers is very great.

The following are the prices of these boilers delivered f.o.b. at Hamburg, Germany (discount of 10 per cent. allowed):—

Horse-power.	Water-heating Surface.	Total Heating-surface.	Water evaporated per Hour.	Width, including Masonry.		Length, including Masonry.		Approximate Weight, including Case.	Price of Boiler in Hamburg.	Price of Heater, extra.
	Sq. ft.	Sq. ft.	Lb.	Ft.	in.	Ft.	in.	Cwt.	£	£
*5	54	80	130	3	4	5	0	59	95	10
5	54	80	130	4	11	6	0	47	90	10
6	81	108	195	4	11	7	4	49	95	14
8	108	144	260	5	4	7	4	54	105	14
10	129	194	312	5	4	9	4	98	125	15
15	172	226	416	5	4	9	4	124	150	15
18	202	282	490	5	8	10	4	128	160	15
20	229	321	562	5	8	10	4	132	175	20
22	275	367	675	5	8	10	4	140	187	20
25	286	401	702	6	5	10	4	150	200	21
30	344	481	840	7	0	10	4	163	238	22
35	413	550	1,012	7	0	10	4	177	248	22

* Special construction in cast-iron casing, only requiring to be lined with 4½in. of fire-bricks.

Horse-power.	Water-heating Surface.	Total Heating-surface.	Water evaporated per Hour.	Width, including Masonry.		Length, including Masonry.		Approximate Weight including Case.	Price of Boiler in Hamburg.	Price of Heater, extra.
	Sq. ft.	Sq. ft.	Lb.	Ft.	in.	Ft.	in.	Cwt.	£	£
40	458	642	1,125	8	4	10	4	220	278	25
45	550	734	1,350	8	4	10	4	236	300	25
50	573	802	1,405	9	8	10	4	256	325	27
60	688	963	1,687	11	0	10	4	306	370	30
70	825	1,101	2,024	11	0	10	4	367	413	32
75	918	1,146	2,250	9	8	10	4	374	425	32
80	963	1,183	2,376	12	4	10	4	387	465	35
90	1,124	1,444	2,755	12	4	10	4	411	510	37
100	1,169	1,559	2,867	14	4	10	4	492	540	42
120	1,364	1,753	3,345	14	4	10	4	512	600	42
130	1,559	1,948	3,823	14	4	10	4	551	650	45
150	1,742	2,178	4,273	15	8	10	4	630	725	50
180	1,960	2,395	4,807	15	8	10	4	670	775	50
200	2,178	2,613	5,342	15	8	10	4	710	825	50

The connections with the boilers comprise a strong cast-iron frame, which has to be filled with bricks; all wall-plates, tie-rods, deflecting-plates, T-iron for furnace-roof; soot-, fire-, ash-, and safety-doors; fire-bars and frame, damper and frame, bolts and packing; safety-, stop-, and feed-valves; blow-off cock, water-gauge, test-taps, pressure-gauge and taps; all screw-keys and tube-cleaners, both inside and out; indiarubber tube, with nozzle, to blow off soot from outside, and a special steel instrument to remove scale from inside, of tubes; also a steam draining apparatus and everything complete necessary for the boilers, except the brick lining and masonry, and steam- and water piping not forming portion of the boiler itself.

Classes II. and III.—There is not any material difference between these two classes. The principles of each are the same—namely, that of having a steam-chest or receiver on the top of the tubes. In Class II. the fire goes in among the tubes on the same system as that of Class I., while in Class III. the flue is at the bottom of the tubes instead of at the top, as in the other two systems. The price of the boiler in Classes II. and III. is the same, and the articles supplied along with them are the same as that for Class I.

The following are the dimensions and weights of the inexplosible water-tube boilers, improved Root's system, with superincumbent steam-and-water drums and circulation (15 per cent. discount allowed):—

Horse-power.	Number of Steam-and-water Drums	Water-heating Surface.	Breadth of Boilers, including Masonry.		Length of Boilers, including Masonry.		Approximate Weight.	Price of Boiler at Port Hamburg.	Price of Water-heater.
		Sq. ft.	Ft.	in.	Ft.	in.	Cwt.	£	£
8	1	138	5	4	8	4	98	140	15
15	1	230	5	8	10	4	134	185	20
20	1	306	5	8	13	6	173	200	20
25	1	366	5	8	13	6	187	215	20
30	1	428	5	8	13	6	193	225	20
35	1	535	6	4	13	6	197	263	22
40	1	642	7	0	13	6	246	308	23
50	2	856	8	4	13	6	315	373	25
60	2	978	8	4	13	6	344	400	25
70	2	1,070	9	8	13	6	370	425	28
80	2	1,223	9	8	13	6	394	480	28
90	2	1,468	11	0	13	6	420	550	30
120	2	1,712	12	4	13	6	492	650	35
130	2	1,926	12	4	13	6	522	700	38
150	3	2,338	14	4	13	6	748	800	43
200	3	2,904	15	8	13	6	788	975	50
250	3	3,485	15	8	13	6	886	1,050	50
300	3	3,908	15	8	14	8	984	1,250	50

PUMPING MACHINERY.

Pumping machinery of different descriptions was exhibited at the Melbourne Exhibition—namely, centrifugal, rotary, rocker, chain, and duplex plunger-pumps. The latter is a class of pump most applicable to mining.

These duplex plunger-pumps were exhibited by Messrs. Parke and Lacey, of Sydney, who are agents for them. The pumps are manufactured at Knowles's Steam Pump-works at New York and Boston, and they have many points to recommend them for use in mines. These pumps are made for working both horizontally and vertically. (See Figs. 53 to 62, in annexed sketches.)

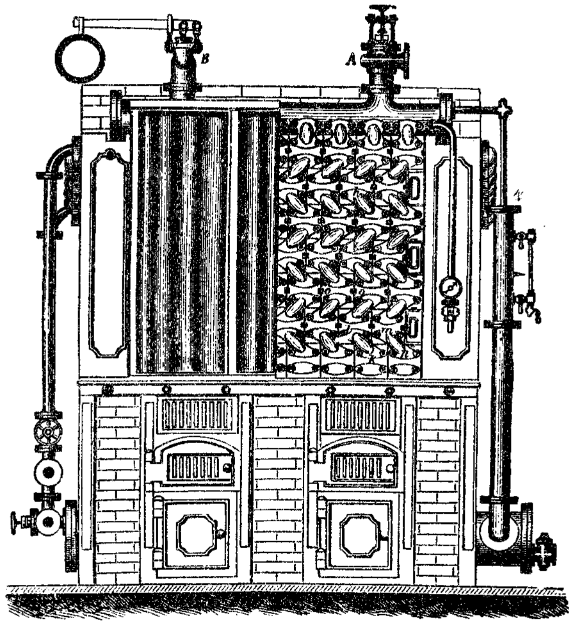


Fig. 47. Front view of boiler.

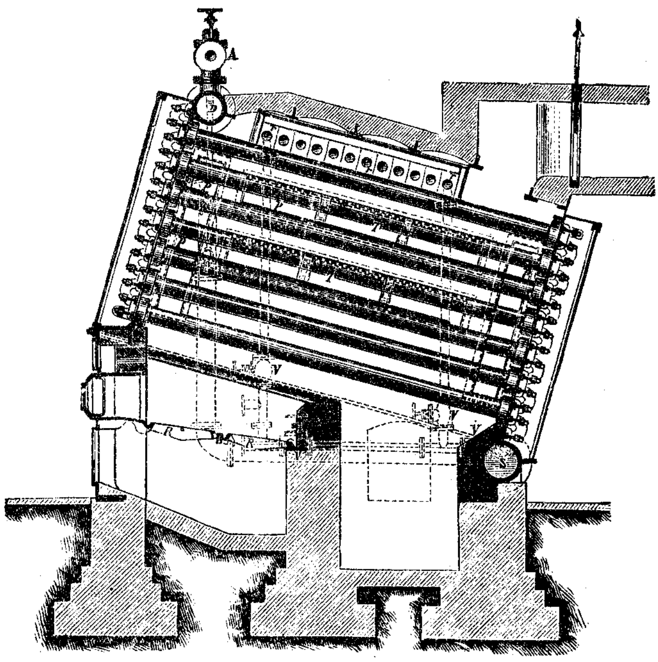


Fig. 48. Longitudinal Section of boiler.

Root's tube boiler connections.

Inexplosible Water tube boilers with rapid circulation

Improved Root's System

Front view.

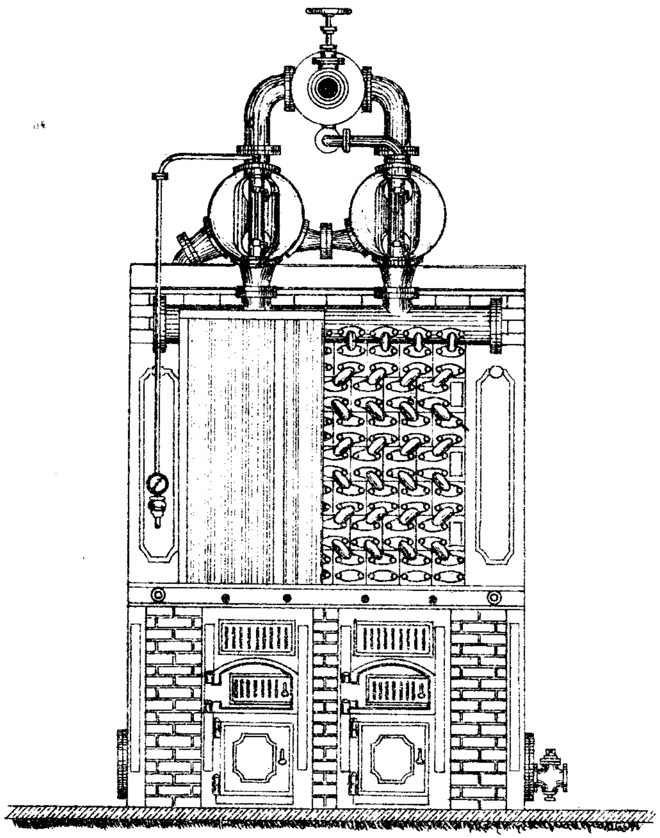


Fig 47.

Longitudinal Section.

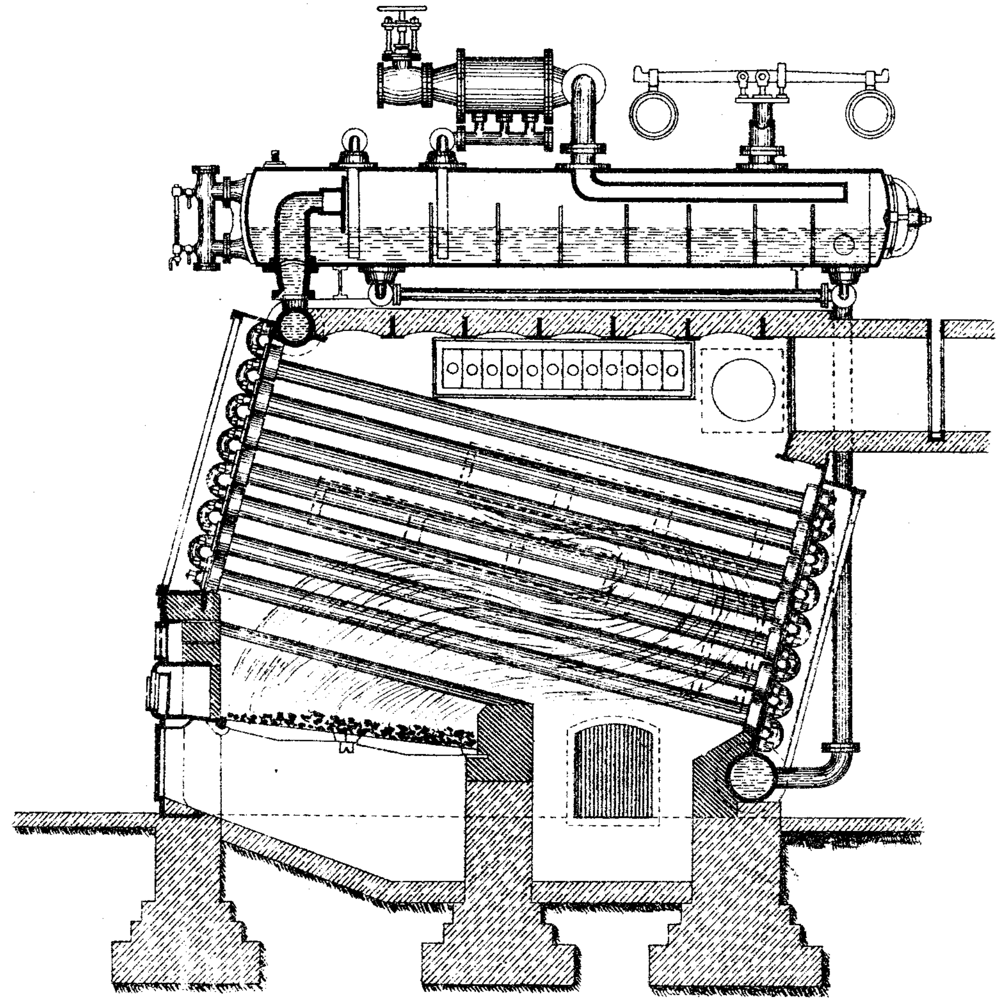


Fig 48.

III. -Inexplosible Water tube Boilers with circulation
- and wrought iron interchangeable tubular Water chambers.

Petry Walther System.

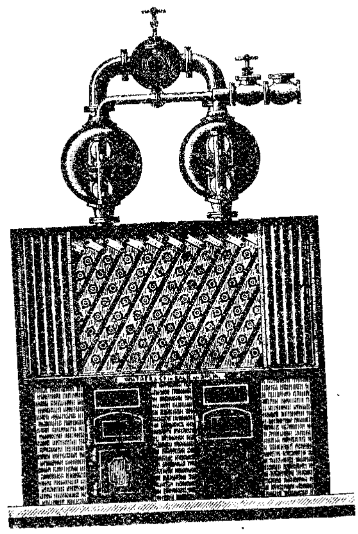


Fig. 49.— Front view.

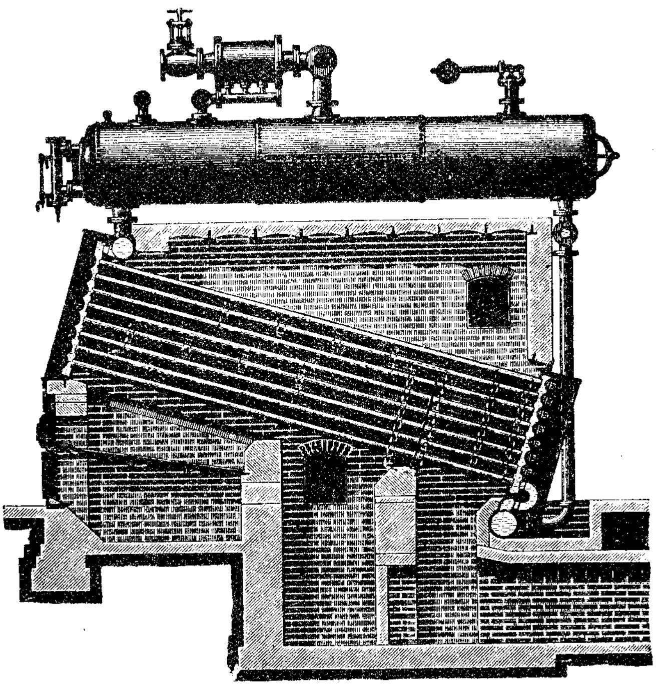


Fig. 50 — Longitudinal section.

II. -Inexplosible Water tube Boilers with circulation

Root's system **improved.**

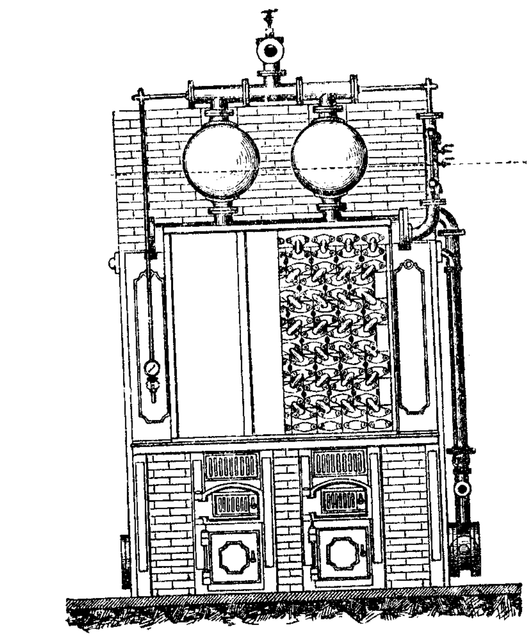


Fig. 51.— Front view.

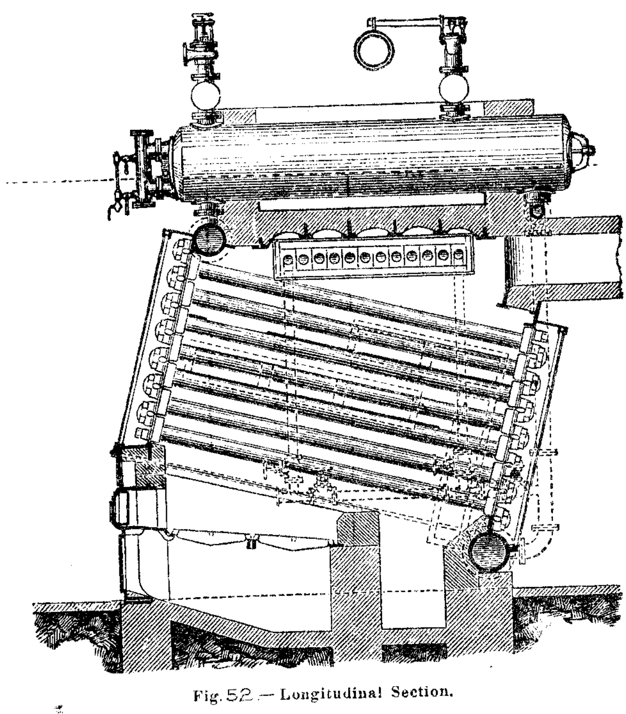


Fig. 52.— Longitudinal Section.

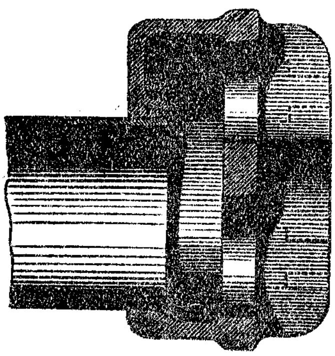


Fig. 49.— Section of tube nead.

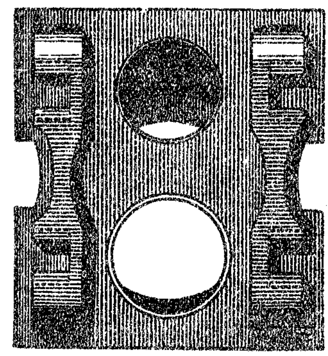


Fig. 50.— Front elevation of tube head.

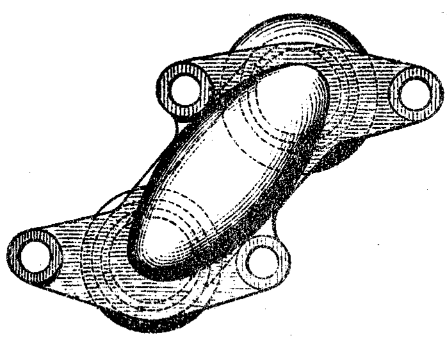


Fig. 51.— Front elevation of cap.

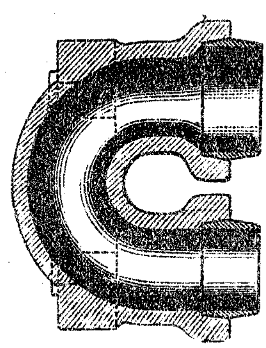


Fig. 52— Section of cap and rings.

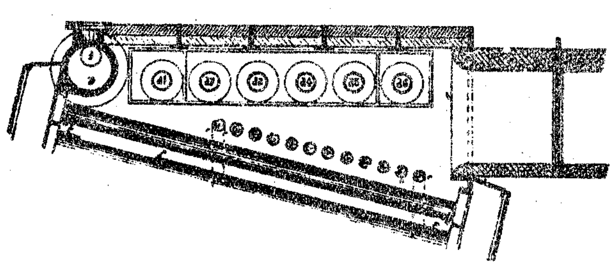


Fig. 52A— Feed-Water-Heater and Superheater.

Root's tube boiler connections.

Where the horizontal pattern is used, a chamber has to be constructed near the bottom of the shaft, or near the place where they lift the water, and they are manufactured so that the water can be forced up in one column to a height of 1,000ft., thereby dispensing with tanks and change of lifts at every 180ft. or 200ft., as is the case when the common lifts and plunger-pumps are used. They have a great advantage over the common pumps now in use in the colonies, inasmuch that they do not require so large a shaft to be constructed. Instead of having large columns and rods, with plungers and tanks at change of lifts in the shaft, there is only one column of pipes for the water and one small column of steam-pipes for supplying steam for the engine, which is erected in the chamber.

These pumps are made duplex for deep mines, and one is fixed in a chamber constructed at each side of the shaft, the water from both going up one column. With this arrangement the constant drainage of the mine is secured. One engine and pump may be stopped for a time to get any repairs done, while the other is at work. The manner in which these pumps are fixed is shown in annexed sketch (Fig. 53) of Knowles's duplex compound condensing mine-pump; connected complete, placed 800ft. below the surface.

Both pumps discharge into one column-pipe, and take steam from a single steam-pipe, which, when properly covered, does not affect the temperature in the shaft. Each pumping-engine takes its supply from a tank or sump, to which point the water is raised by the ordinary sinking pump, and from this same tank the air-pump draws the supply of injection-water necessary for condensing the exhaust steam from the pumping-engine. The warm water from the condenser flows back again into the same tank and enters at a point close to the suction-pipe of the main pumps, so that it is immediately taken up and delivered to the surface, and the coolest left available for condensing purposes. By means of a float in this tank, operating on a balance throttle-valve in the steam-pipe, the engine is stopped automatically should the tank become empty. The movement of this float up or down also regulates the speed of the pumping-engine, so that no power is wasted.

When these pumps are used to force up water from great depths, special-made wrought-iron pipes, lap-welded, are used, with cast-iron flanges on each end. The flanges are shrunk on and beaded; they are also faced off and counterbored with male and female joints, so that the gaskets between the joints cannot blow out. These pipes are tested to a pressure of 600lb. to the square inch. The agents could not give the prices of the pipes, but they have the great advantage of being light, which makes them easily handled, and they are less costly in freight and carriage where they have to be transported for any great distance. These pipes are made in any lengths up to 15ft., and in size from 6in. to 15in. outside diameter.

The steam-pipes are made of wrought-iron, in any length, having screwed couplings on the joints, and to prevent the steam from condensing in the pipes they are covered with a coating of asbestos sheathing, which is a good non-conducting substance. The covering is done in two ways, as will be seen by referring to the annexed sketches (Figs. 54 and 55), Class C and Class D. The only difference in the covering of the two classes is that Class C is covered entirely with asbestos fibre, with partitions of asbestos sheathing, and Class D has a covering of asbestos fibre next the pipe, with a covering of horsehair felt above, which makes the covering of the latter class a little less expensive. These two substances are both good non-conductors, of light weight, and will not crack under vibration, the layers being enclosed between double walls of non-porous sheathing.

Both classes of covering are made in sections of 3ft. in length, and of sizes to fit any diameter of pipe. Each section is cut longitudinally on one side, so as to easily slide on to the pipe. The joint is held together by staples, over which is pasted a flap, as shown in the annexed sketch, Fig. 56.

The following are the manufacturers' directions for covering the flaps: The paste for fastening down the flap should be made rather thick, and of good wheat flour. Before putting on the paste moisten the flap with water until it is pliable, then open a section of the covering at one end with both hands and slip it over the pipe to be covered; draw the edges together and paste the flap firmly in its proper position. On large sizes of pipes use a strap, or rope with a noose. Place the next section on the pipe, and push the abutting joints together, then paste around each abutting joint one layer of strips. The staples are for use only when the position of the pipe prevents a section staying in its place with the use of paste alone.

With these classes of covering there is no difficulty in maintaining the steam at a uniform temperature between the boiler and the engine. The prices of the two classes of covering, in 3ft. lengths, are as follow:—

—	Diameter of Inside of Pipe.									
	$1\frac{1}{4}$in.	$1\frac{1}{2}$in.	2in.	$2\frac{1}{2}$in.	3in.	$3\frac{1}{2}$in.	4in.	$4\frac{1}{2}$in.	5in.	6in.
	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.
Class C	2 0	2 1	2 3	2 7	3 0	3 4	3 6	3 10	4 2	4 8
Class D	1 7	1 8	1 10	2 1	2 5	2 8	2 10	3 1	3 4	3 9

When sinking shafts or pumping out water in mines that have been flooded, light vertical steam-pumps are used, of the pattern shown in the annexed sketch (Fig. 57). Adjustable wrought-iron dogs for hanging the pump to the shaft-timbers are bolted to the cylinders; but it works equally as well, and throws as continuous a stream of water, when hanging to the tackle as when hooked on to the timber in the shaft. The exhaust-steam pipe from the steam-cylinder is led into the pump or tank above where the water is discharged.

These pumps require but little more room in the shaft than a common lift-pump, and possess the advantage of continuous action, throwing a steady stream of water, owing to their being fitted with double plungers; and, being suspended by a tackle from the top, they are readily lifted, in the case of sinking in hard rock, when blasting has to be done, so as to prevent their being damaged when firing shots.

The lower plunger works in a removable cylinder of gun-metal or hard iron, as may be desired, and this can be easily replaced by a new one when worn out. The object in having this removable cylinder is, the ordinary cylinders get cut with grit, and also get worn, when the water—as in many cases, such as at the Thames—is strongly impregnated with sulphurous acid. When the removable cylinder gets worn it can be rebored if not too much worn, or it can be replaced by a new one, causing only a stoppage of not more than twenty minutes at the outside. The water-valves are made of vulcanised rubber, and work on seats of gun-metal. Swing-bolts admit of easy access to the water-valves and pump-barrel.

The following table is given by the manufacturers of the dimensions of the different pipes required for this class of pump, their weight and capacity, and also the prices for four different sizes. There seemed to be a strong objection amongst exhibitors of all mining machinery to quote the prices, as they said it afforded information to other manufacturers in the same line of business, which they declined to give; but if intending purchasers would write to their respective offices every information would be afforded them.

Steam Cylinder.	Lower Plunger.	Upper Plunger.	Stroke.	Gallons per Stroke.	Capacity per Minute at Ordinary Speed.	Steam-pipe.	Exhaust-pipe.	Suction-pipe.	Delivery-pipe.	Dimensions over All.	Weight.	Price.
In.	In.	In.	In.		Gall.	In.	In.	In.	In.	In.	Lb.	£ s. d.
6	5⅜	3¾	10	0·48	40	0¾	1	3½	2	70 by 18 by 13	800	83 6 0
8	5⅜	3¾	10	0·48	40	1	1¼	3½	2	71 by 18 by 13	900	88 10 0
8	6⅜	4½	10	0·69	80	1	1¼	4	2½	72 by 22 by 15	1,000	90 0 0
10	7	5	10	0·85	100	1¼	1½	4	3	72 by 22 by 15	1,200	93 15 0
12	8½	6	12	1·47	147	2	2½	5	4	80 by 24 by 17	2,000	...
12	10	7	16	2·66	200	2	2½	6	5	114 by 29 by 22	2,800	...
14	10	7	16	2·66	200	2	2½	6	5	114 by 29 by 22	3,000	...
16	10	7	16	2·66	200	2½	3	6	5	115 by 29 by 23	3,500	...
16	12	8½	16	3·92	300	2½	3	8	6	115 by 30 by 23	4,000	...
18	12	8½	16	3·92	300	2½	3	8	6	116 by 31 by 24	5,500	...
18	14	10	24	8·16	400	3½	4	10	6	152 by 40 by 30	...	...
20	14	10	24	8·16	400	3½	4	10	6	152 by 40 by 30	...	...

Another pattern of this description of pump was working in the Melbourne Exhibition. It is termed "Knowles's Patent Compound Condensing Mining-pump" (Fig. 58), and is said to be capable of forcing up water to a height of 1,000ft. The high-pressure cylinder was 8in. in diameter, and the low-pressure one 14in., with 16in. stroke. There were four plungers 4½in. in diameter, which were calculated to raise 400 gallons of water per minute to the height mentioned. A small steam vacuum-pump was attached to the condenser, the steam-cylinder being 8in. in diameter, with vacuum-pump connected on the end of the piston-rod. The condensing-pump, with vacuum and pressure-gauges was quoted £335 f.o.b. in Auckland, and its total weight was 21,000lb. The following table shows the sizes and capacities of these and the duplex pumps:—

Steam-cylinders.	Water-plungers.	Stroke.	Gallons per Stroke each Plunger.	Strokes per Minute of each Plunger, Ordinary Speed.	Capacity of both Plungers at Speed stated (per Minute).	Steam-pipe.	Exhaust pipe.	Suction-pipe.	Delivery-pipe.
In.	In.	In.			Gal.	In.	In.	In.	In.
4½	2	4	0·05	50 to 100	5 to 10	0½	0¾	1½	1
6	2½	7	0·15	45 to 90	13 to 27	1	1½	2	2
6	3	7	0·22	45 to 90	20 to 41	1	1½	2½	2
6	3½	7	0·28	45 to 90	25 to 50	1	1½	2½	2
7½	4	10	0·54	37 to 75	40 to 80	1½	1½	3½	2½
7½	4½	10	0·69	37 to 75	50 to 100	1½	1½	3½	2½
8	4	12	0·64	37 to 75	48 to 96	1½	1½	3½	2½
8	5	12	1·02	37 to 75	76 to 153	1½	1½	4	3
10	5	12	1·02	37 to 75	76 to 154	2	2	4	3
10	5½	12	1·22	37 to 75	91 to 183	2	2	5	4
12	5	12	1·02	37 to 75	76 to 153	2	2½	4	3
12	5½	12	1·22	37 to 75	91 to 183	2	2½	5	4
12	6	12	1·47	37 to 75	110 to 220	2	2½	5	4
12	7	12	2·00	37 to 75	150 to 300	2	2½	5½	4½
14	6	12	1·47	37 to 75	110 to 220	2½	3	5	4
14	7	12	2·00	37 to 75	150 to 300	2½	3	5½	4½
16	7	12	2·00	37 to 75	150 to 300	3	3	5½	4½

Steam-cylinders.	Water-plungers.	Stroke.	Gallons per Stroke each Plunger.	Strokes per Minute of each Plunger, Ordinary Speed.	Capacity of both Plungers at Speed stated (per Minute).	Steam-pipe.	Exhaust pipe.	Suction-pipe.	Delivery-pipe.
In.	In.	In.			Gal.	In.	In.	In.	In.
16	8	12	2·61	37 to 75	195 to 391	3	3	6	5
18	5	18	1·53	32 to 65	99 to 198	4	5	4	3
18	5	24	2·04	25 to 50	102 to 204	4	5	5	4
18	6	18	2·20	32 to 65	143 to 286	4	5	5	4
18	6	24	2·94	25 to 50	147 to 294	4	5	5½	4½
20	5	18	1·53	32 to 65	99 to 198	5	5	4	3
20	5	24	2·04	25 to 50	102 to 204	5	5	5	4
20	6	18	2·20	32 to 65	143 to 286	5	5	5	4
20	6	24	2·94	25 to 50	147 to 294	5	5	5½	4½
20	7	18	3·00	32 to 65	195 to 390	5	5	5½	4½
20	7	24	4·00	25 to 50	200 to 400	5	5	5½	4½
24	6	18	2·20	32 to 65	143 to 286	...	...	5	4
24	6	24	2·94	25 to 50	147 to 294	...	...	5½	4½
24	7	24	4·00	25 to 50	200 to 400	...	...	6	5
24	8	24	5·22	25 to 50	261 to 522	...	...	6	5
24	9	24	6·60	25 to 50	330 to 660	...	...	6	5
30	10	36	12·24	20 to 50	488 to 1,200	...	...	8	6

It will be seen from the above table there must be a great friction in the flow of water in both the suction- and delivery-pipes; but the manufacturers state that it is found to be more economical to use small pipes when the cost and room larger pipes would take up in a shaft is taken into consideration.

The manufacturers claim the following advantages for this class of pump over the ordinary plunger- and lift-pumps:—

(1.) The first cost is from 30 to 50 per cent. less than the ordinary pumps.

(2.) In transportation and erection the expense is less than half.

(3.) They do not require massive and expensive foundations or large buildings.

(4.) There are no pump-rods, guides, rollers, bobs, or other parts to produce friction, get out of order, and wear out.

(5.) They require less attendance, as there are fewer working-parts; also require less expense for oil, grease, waste, and similar supplies; and they take up considerably less room in the shaft.

(6.) As they are double-action the column of water is kept continually in motion, thereby relieving pipes and pumps from shock and jar, and avoiding leaky joints in column-pipe.

(7.) Smaller column-pipes can be used, as the water is not intermittent in its flow.

The annexed plans (Figs. 59 to 62) will give an idea of their construction. They are made from 3in. diameter of water-cylinder, with 7in. stroke, to 18in. in diameter and 36in. stroke; the former being calculated to lift 125 gallons of water per minute, and the latter 1,190 gallons per minute; but the manufacturers only give the prices of pumps that will lift 200 gallons per minute, which vary from £110 to £146. Of the larger sizes the manufacturers supply the prices on application.

Pumps of a similar pattern to these are at work at the Broken Hill Proprietary Company's mine for lifting the water from the dams to supply the water-jacket furnaces with a constant stream of cold water, and they give the greatest satisfaction.

As stated previously, there is considerable trouble in getting at the exact cost of American mining machinery of every description. It is not exhibited by the manufacturers, but by agents, who generally decline to state what it can be purchased for in America, but they are always willing to quote prices which they are prepared to supply it for, which in many instances are fully 50 per cent. more than it originally cost.

Useful Information in connection with Steam-pumps and Pipes.

Doubling the diameter of a pipe increases its capacity four times. Friction of liquids in pipes increases as the square of the velocity. See table of "Friction of water in pipes," on page .

The mean pressure of the atmosphere is usually estimated at 14·7lb. per square inch, so that with a perfect vacuum it will sustain a column of mercury 29·9in., or a column of water 33·9ft., high.

To find the pressure in pounds per square inch of a column of water, multiply the height of the column in feet by 0·484. Approximately, we may say that every foot elevation is equal to ½lb. pressure per square inch; this allows for ordinary friction.

To find the quantity of water elevated in one minute, running at 100ft. of piston-speed per minute: Square the diameter of the water-cylinder in inches, and multiply by 4. Example: Capacity of a 5in. cylinder is desired. The square root of the diameter (5in.) is 25, which, multiplied by 4, gives 100, the number of gallons per minute (approximately).

To find the horse-power necessary to elevate water to a given height, multiply the total weight of the water in pounds by the height in feet, and divide the product by 33,000. (An allowance of 25 per cent. should be added for water-friction, and a further allowance of 25 per cent. for loss in steam-cylinder.)

The area of the steam-piston, multiplied by the steam-pressure, gives the total amount of pressure that can be exerted. The area of the water-piston, multiplied by the pressure of water per square inch, gives the resistance. A margin must be made between the power and the resistance to move the pistons at the required speed—say from 20 to 40 per cent., according to speed and other conditions.

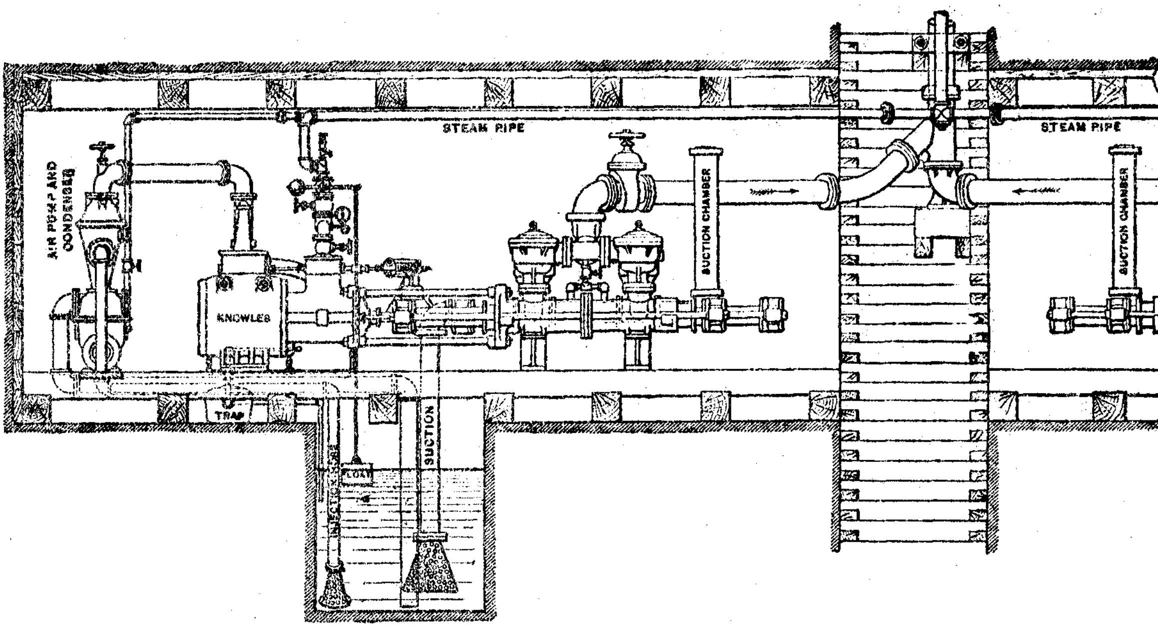
To find the capacity of a cylinder in gallons: Multiplying the area in inches by the length of stroke in inches will give the total number of cubic inches. Divide this amount by 277·274 (which is the cubical contents of a gallon in inches), and the product is the capacity in gallons.

Wrought-iron Welded Pipe.—Dimensions, weights, &c. (1in. and below are butt-welded, and proved to 300lb. per square inch, hydraulic pressure; 1¼in. and above are lap-welded, and proved to 500lb. per square inch, hydraulic pressure):—

Inside Diameter.	Outside Diameter.	External Circumference.	Length of Pipe per Sq. Ft. of Outside Surface.	Internal Area.	External Area.	Length of Pipe containing One Cubic Foot.	Weight per Foot of Length.	No. of Threads per Inch of Screw	Weight of Water per Foot of Length.
In.	In.	In.	Ft.	In.	In.	Ft.	Lb.		Lb.
⅛	·40	1·272	9·44	·057	·129	2500	·24	27	·005
¼	·54	1·696	7·075	·104	·229	1385	·42	18	·021
⅜	·67	2·121	5·657	·191	·358	751·5	·56	18	·047
½	·84	2·652	4·502	·304	·554	472·4	·84	14	·085
¾	1·05	3·299	3·637	·533	·866	270	1·12	14	·190
1	1·31	4·134	2·903	·862	1·357	166·9	1·6	11½	·349
1¼	1·66	5·25	2·301	1·496	2·164	96·25	2·25	11½	·527
1½	1·9	5·969	2·01	2·038	2·835	70·65	2·69	11½	·760
2	2·37	7·461	1·611	3·355	4·430	42·36	3·66	11½	1·356
2½	2·87	9·032	1·328	4·783	6·491	30·11	5·77	8	2·116
3	3·5	10·996	1·091	7·388	9·621	19·49	7·54	8	3·049
3½	4	12·566	·955	9·837	12·566	14·56	9·05	8	4·155
4	4·5	14·137	·849	12·730	15·904	11·31	10·72	8	5·405
4½	5	15·708	·765	15·939	19·635	9·03	12·49	8	6·851
5	5·56	17·475	·629	19·990	24·299	7·20	14·56	8	8·500
6	6·62	20·813	·577	28·889	34·471	4·98	18·76	8	12·312
7	7·62	23·954	·505	38·737	45·663	3·72	23·41	8	16·662
8	8·62	27·096	·444	50·039	58·426	2·88	28·34	8	21·750
9	9·68	30·433	·394	63·633	73·715	2·26	34·67	8	27·500
10	10·75	33·772	·355	78·838	90·792	1·80	40·64	8	34·000

Friction of Water in Pipes.—Friction-loss in pounds pressure per square inch for each 100ft. of length in different-size clean iron pipes discharging given quantities of water per minute (G. A. Ellis, C.E.):—

Gallons per Minute.	Sizes of Pipes (Inside Diameter).													
	¾in.	1in.	1¼in.	1½in.	2in.	2½in.	3in.	4in.	6in.	8in.	10in.	12in.	14in.	16in.	18in.
5 ...	3·3	0·84	0·31	0·12	...	...	...	...	...	...	...	...	...	...	...
10 ...	13·0	3·16	1·05	0·47	0·12	...	...	...	...	...	...	...	...	...	...
15 ...	28·7	6·98	2·38	0·97	...	...	...	...	...	...	...	...	...	...	...
20 ...	50·4	12·3	4·07	1·66	0·42	...	...	...	...	...	...	...	...	...	...
25 ...	78·9	19·0	6·40	2·62	...	0·21	0·10	...	...	...	...	...	...	...	...
30 ...	...	27·5	9·15	3·75	0·91	...	...	...	...	...	...	...	...	...	...
35 ...	...	37·0	12·4	5·05	...	...	...	...	...	...	...	...	...	...	...
40 ...	...	48·0	16·1	6·52	1·60	...	...	...	...	...	...	...	...	...	...
45 ...	...	...	20·2	8·15	...	...	...	...	...	...	...	...	...	...	...
50 ...	...	...	24·9	10·0	2·44	0·81	0·35	0·09	...	...	...	...	...	...	...
75 ...	...	...	56·1	22·4	5·32	1·80	0·74	...	...	...	...	...	...	...	...
100 ...	...	...	...	39·0	9·46	3·20	1·31	0·33	0·05	...	...	...	...	...	...
125 ...	...	...	...	...	14·9	4·89	1·99	...	...	...	...	...	...	...	...
150 ...	...	...	...	...	21·2	7·0	2·85	0·69	0·10	...	...	...	...	...	...
175 ...	...	...	...	...	28·1	9·46	3·85	...	...	...	...	...	...	...	...
200 ...	...	...	...	...	37·5	12·47	5·02	1·22	0·17	...	...	...	...	...	...
250 ...	...	...	...	...	...	19·66	7·76	1·89	0·26	0·07	0·03	0·01	...	...	...
300 ...	...	...	...	...	...	28·06	11·2	2·66	0·37	0·09	0·04	...	...	...	...
350 ...	...	...	...	...	...	...	15·2	3·65	0·50	0·12	0·05	0·02	...	...	...
400 ...	...	...	...	...	...	...	19·5	4·73	0·65	0·16	0·06	...	...	...	...
450 ...	...	...	...	...	...	...	25·0	6·01	0·81	0·20	0·07	0·03	...	...	...
500 ...	...	...	...	...	...	...	30·8	7·43	0·96	0·25	0·09	0·04	0·017	0·009	0·005
750 ...	...	...	...	...	...	...	...	...	2·21	0·53	0·18	0·08	...	...	...
1,000 ...	...	...	...	...	...	...	...	...	3·88	0·94	0·32	0·13	0·062	0·036	0·020
1,250 ...	...	...	...	...	...	...	...	...	...	1·46	0·49	0·20	...	...	...
1,500 ...	...	...	...	...	...	...	...	...	...	2·09	0·70	0·29	0·135	0·071	0·040
1,750 ...	...	...	...	...	...	...	...	...	...	...	0·95	0·38	...	...	...
2,000 ...	...	...	...	...	...	...	...	...	...	...	1·23	0·49	0·234	0·123	0·071
2,250 ...	...	...	...	...	...	...	...	...	...	...	...	0·63	...	...	...
2,500 ...	...	...	...	...	...	...	...	...	...	...	...	0·67	0·362	0·188	0·107
3,000 ...	...	...	...	...	...	...	...	...	...	...	...	1·11	0·515	0·267	0·150
3,500 ...	...	...	...	...	...	...	...	...	...	...	...	...	0·679	0·365	0·204
4,000 ...	...	...	...	...	...	...	...	...	...	...	...	...	0·910	0·472	0·263
4,500 ...	...	...	...	...	...	...	...	...	...	...	...	...	...	0·593	0·333
5,000 ...	...	...	...	...	...	...	...	...	...	...	...	...	...	0·703	0·408

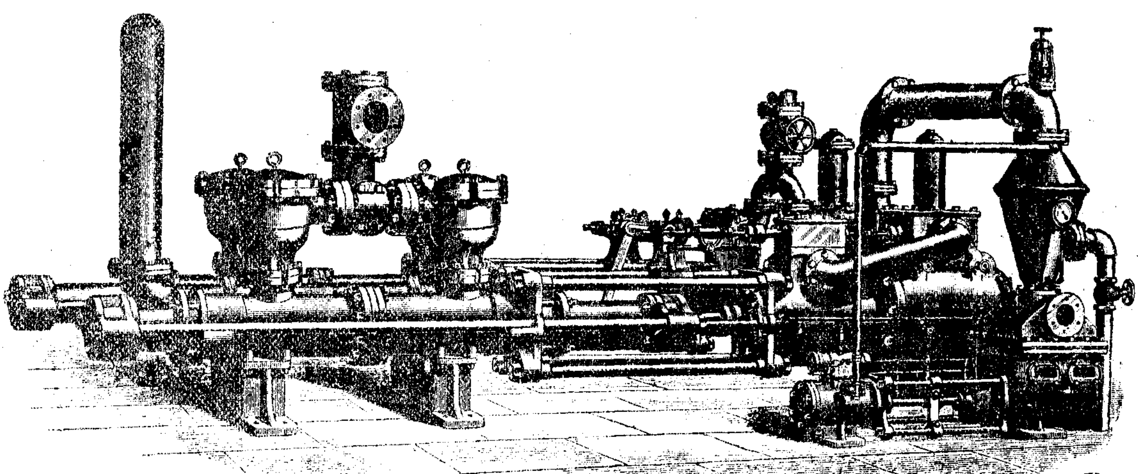


KNOWLES DUPLEX COMPOUND CONDENSING MINE PUMP.—CONNECTED UP COMPLETE,

"POT VALVE," DOUBLE PLUNGER PATTERN.

Located in Pump Station 800 feet below Surface.

Fig 58.

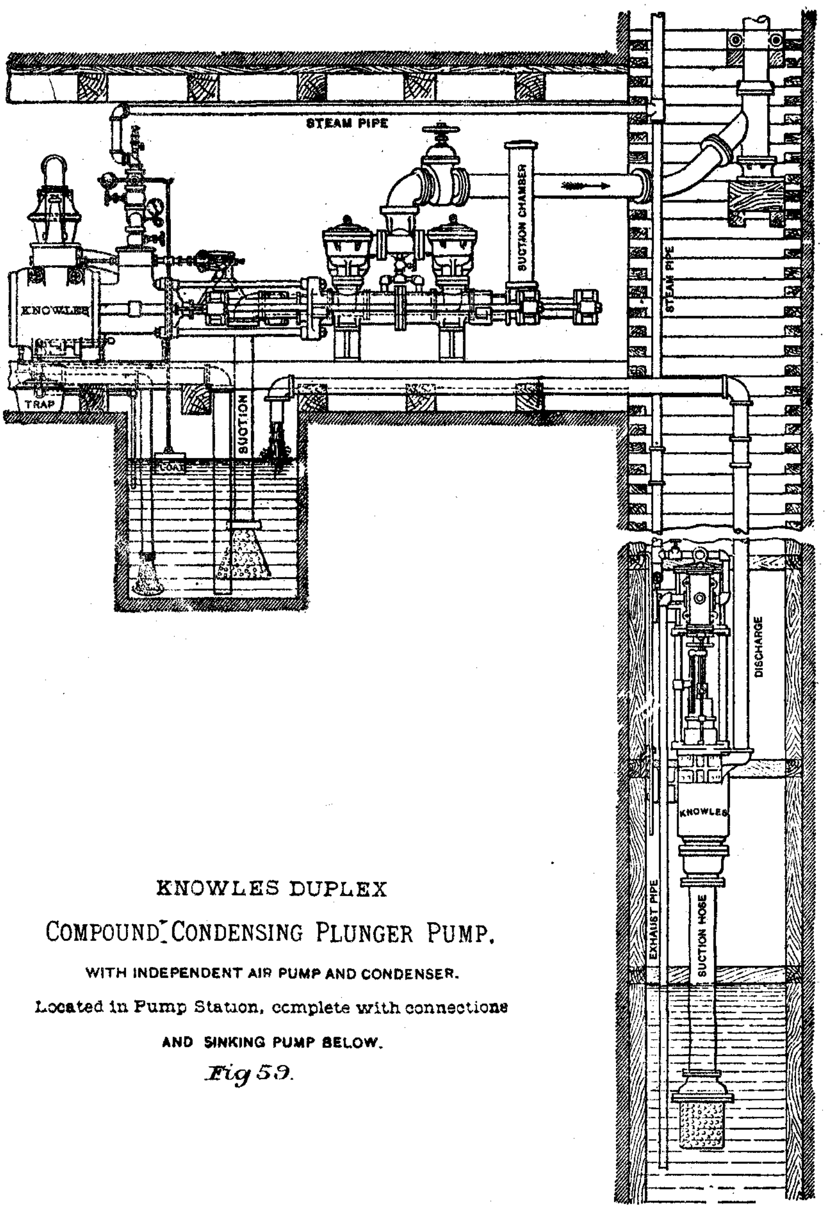


KNOWLES DUPLEX COMPOUND CONDENSING PLUNGER PUMP.

"POT VALVE" PATTERN.

With Independent Air Pump and Jet Condenser.

Fig 60.



KNOWLES DUPLEX

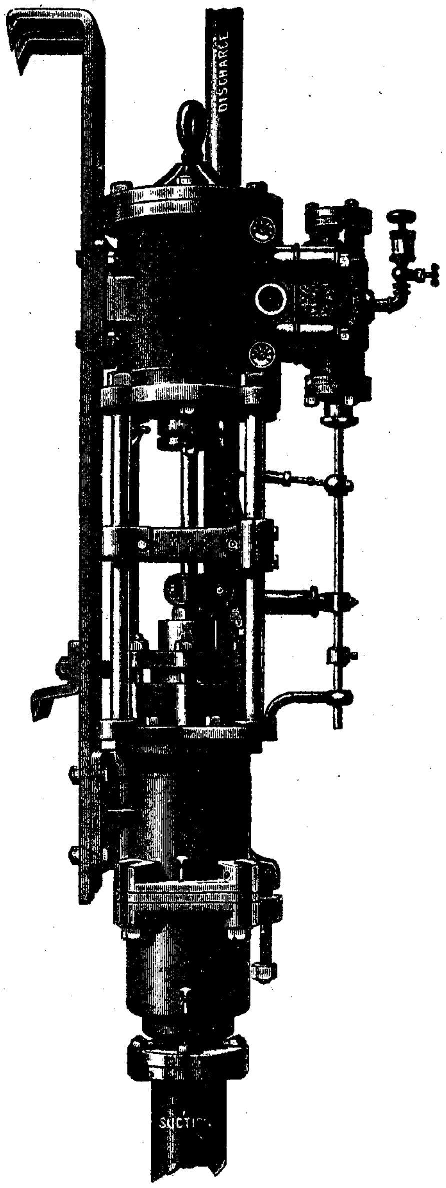
COMPOUND CONDENSING PLUNGER PUMP.

WITH INDEPENDENT AIR PUMP AND CONDENSER.

Located in Pump Station, complete with connections

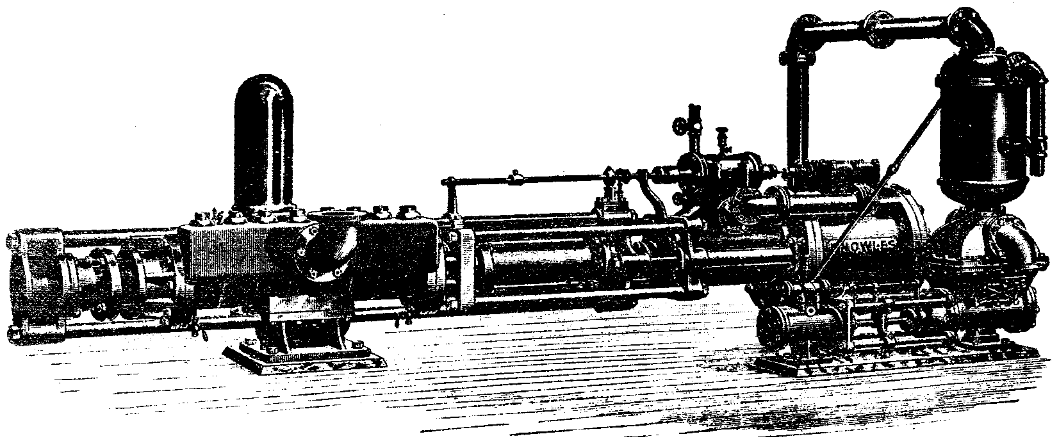
AND SINKING PUMP BELOW.

Fig 59.



KNOWLES PATENT VERTICAL SINKING PUMP

Fig 57.



KNOWLES PATENT COMPOUND CONDENSING PLUNGER MINING PUMP.

"SOLID CYLINDER" PATTERN.

With Independent Air Pump and Condenser.

Fig 58.

Fig 54.

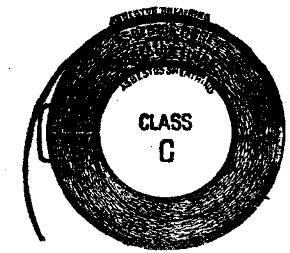
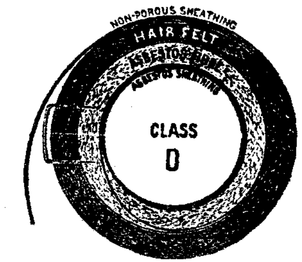
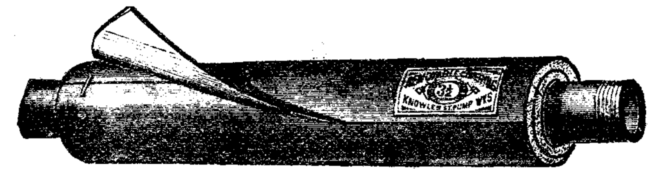


Fig 55.





PATENT REMOVABLE PIPE COVERINGS.

Fig 56.

SPECIAL WROUGHT-IRON COLUMN PIPE.

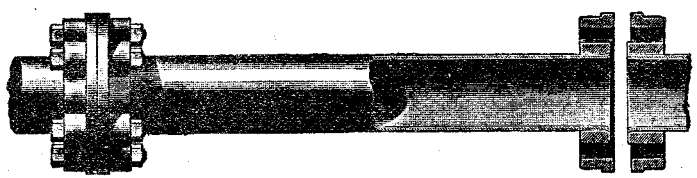
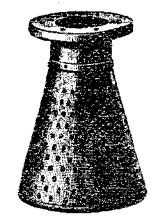
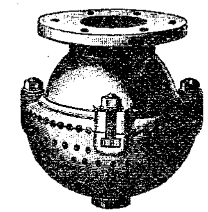


Fig 61.



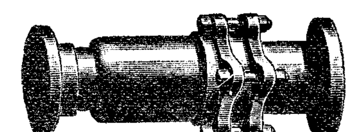
GALVANIZED SHEET-IRON STRAINER.

Fig 61A.



COMBINED STRAINER AND FOOT VALVE.

Fig 61B.



EXPANSION JOINT FOR STEAM PIPE.

Fig 61C.

SUCTION HOSE AND CONNECTIONS.

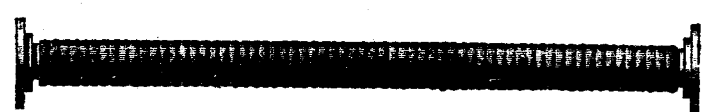
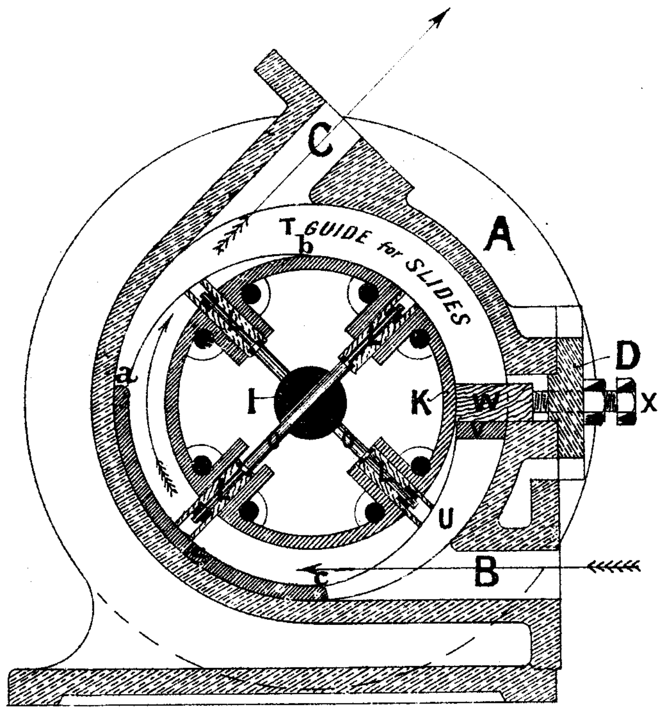


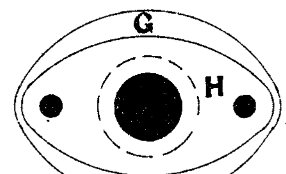
Fig 62.

ROTARY PUMP.

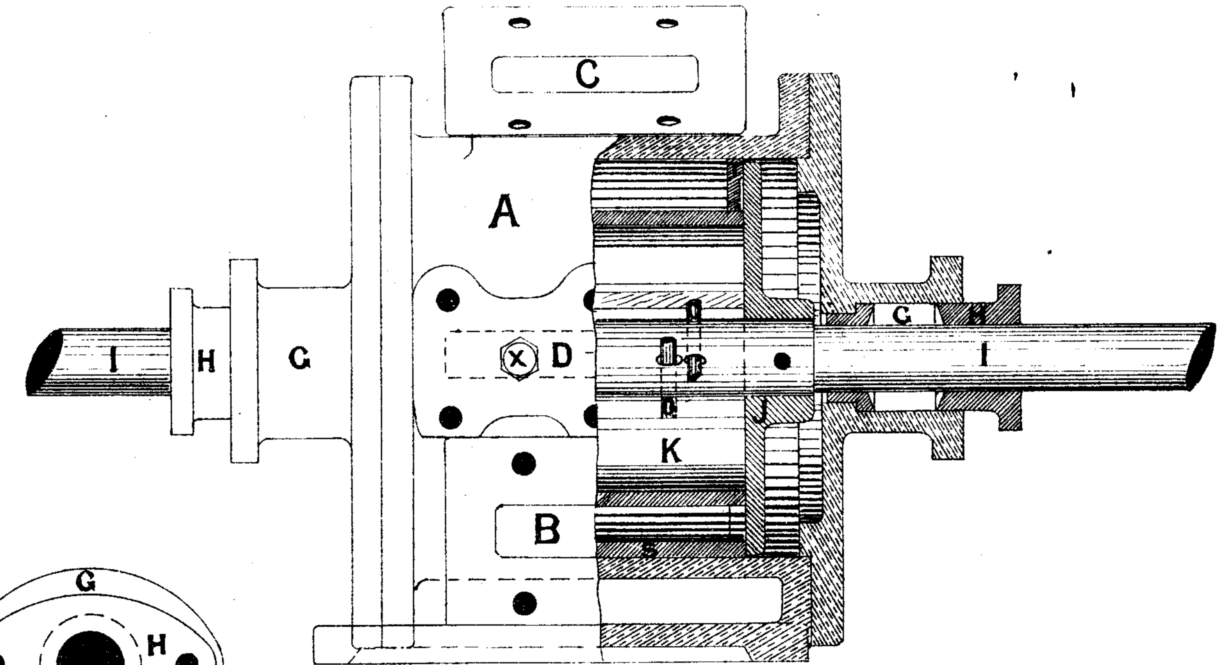


CROSS-SECTION AT CENTRE

Fig. 63.

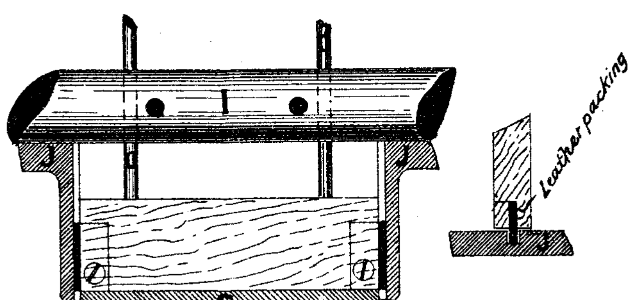


GLANDS

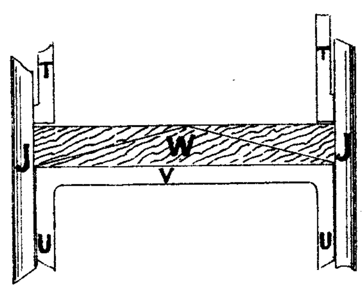


HALF LONGITUDINAL SECTION

Fig. 64.



SLIDE



BACK BLOCK

Rotary Pump.

A rotary pump was exhibited at the Melbourne Exhibition which deserves mention. It was designed and patented in the colonies by R. M. Simpson, of Wellington. It is a pump specially adapted for irrigation, or lifting water in which there is not much sand or grit, but its construction does not admit of it being used where there is thick, muddy water, as the grit would very soon cut the casing into grooves, and its usefulness would thereby be to a certain extent destroyed. It is on the principle of the centrifugal pump, and would lift and force water with good effect to a height of 40ft. or 45ft.—that is, the suction-pipe to be 20ft. and the discharge-pipe 20ft. to 25ft.; but for forcing water to a great height the power required to work it becomes too great for the quantity of water discharged. It is, however, a pump that can be highly recommended for irrigation purposes or for pumping water into reservoirs when the height to lift it is not more than that mentioned, as it will give a high percentage of useful effect for the power employed. The following is a description of its details, which can be easily understood by reference to the plan annexed (Figs. 63 and 64):—

The outside casing, A, of the pump is of metal, in form as shown in plan. Through the sides of the casing are the openings for suction, B, and delivery, C, of the pump. There is also an opening, D, for the purpose of obtaining access to the interior, more especially in case of need for removing and renewing the slides and back blocks, hereinafter to be described. This opening is provided with a cover, which is fastened to the casing. The ends of the casing are enclosed by covers. In the centre of each cover is a bearing, with stuffing-box, G, and gland, H, for carrying the shaft. The shaft I runs through the pump, and to it (within the casing) are attached the flange-plates, J, of the cylinder, K. The shaft runs through the centre of each of the flange-plates, and also through the centre of the cylinder. The cylinder is attached to the flange-plates.

The slides are formed as in plan. On each edge, where it runs against the flange-plate of the cylinder, is attached a strip of leather. This piece of leather projects into a groove made for its reception in the flange-plate of the cylinder. The object of this leather is to prevent as far as possible a passage between the edge of the slide and the flange-plate. On power being applied to turn the shaft, each slide in rotation traverses the working-face plate from the suction to the delivery, carrying in front of it water or air found there. After leaving the working-face plate the slide gradually retires into the cylinder, until at the point *b* it is quite within it. The slide remains within the cylinder until the back block is passed. After the back block is passed, the slide, following the guide-pieces U, gradually comes out again until the working-face plate is reached. From the point *a* the movement of the slide back into the cylinder is caused by the form of the guide-piece T, and by the pressure of the water or air which is being expelled by the pump. This pressure continues to be felt on the slide until it is shut in by the back block.

The cylinder is hollow, and has four slot-openings, L, equidistant from each other, cut from end to end of the cylinder between the flange-plates. Each slot-opening has bearings on which the slides or pistons work. The slides or pistons, N, are of wood or other material, and work on the bearings through the openings in the cylinder, being so confined to their position, and also by the flange-plates, which enclose and overlap the ends of the cylinder. The slides work in pairs, each two opposite being connected by means of rods, O, running from one to the other. These rods pass through the shaft, openings being made for that purpose. There are two rods to each pair of slides. The rods are of such a length as to maintain the slides at proper distance apart, with the edges parallel. The rods penetrate the slides and are fastened to them. The diameter of each rod is reduced where it enters the slide. Where each rod penetrates the slide a washer is let into the slide, against which the shoulder of the rod formed by the reduction rests, so as to prevent the rod wearing into the slide.

Inside the casing, and attached to it between the suction and the delivery, is a plate, S, the surface of which forms a working-face for the ends of the slides to traverse. This working-face is concentric with the cylinder, being in the form of a quarter, or thereabouts, of a circle. The working-face plate is placed preferably at a distance from the cylinder equal to about one-eighth of the diameter of the cylinder. The sides of the working-face plate are overlapped by the flanges of the cylinder, which work against them as closely as possible. Attached to the casing on the inside are the guide-pieces T. There are two of these, one on each side of the delivery. They join the working-face plate at the edge, *a*, nearest the delivery, and there have their surface flush with the working-face. From this point the surfaces of the guide-pieces gradually approach the cylinder, touch it at the point *b*, and so continue as far as the back block at the opening D, where the guide-pieces terminate.

There are two guide-pieces, U, which join the working-face plate at the edge, *c*, nearest the suction opening. Their faces are there flush with the working-face plate. From that point, passing one on each side of the suction opening, they gradually approach the cylinder until they touch it just below the opening D, where they terminate. These guide-pieces are placed so that the flange-plates of the cylinder work against them as closely as possible. The guide-pieces U are connected together just below the opening D by the connecting-plate V, which fills up the space at that point between the cylinder, the flanges of the cylinder, and the outer casing.

The connecting-plate forms a bed upon which rests the back block W. This is a piece of wood or metal, as shown in the plan. It extends from one flange of the cylinder to the other, and so far fills up the space between. It touches the cylinder, and its surface on that side is concave so as to conform with the surface of the cylinder.

Through the cover of the opening D there are adjusting-screws X, for the purpose of keeping the back block against the cylinder.

MINERAL RAILWAYS.

There was a mineral railway exhibited at the Melbourne Exhibition which is suitable for many of the coal-mines and even some of the gold- and silver-mines where the ore has to be conveyed a distance from the mine to the reduction-works. It was manufactured at the Boehmer Union Locomotive Works at Munich, and the agents are Shadler, Koeniger, and Arven, 160, Elizabeth Street, Melbourne, and Shadler, Koeniger, and Co., 8, O'Connell Street, Sydney, of which drawings of the different details are annexed. (See Figs. 65 to 100.)

The railway or tramway is constructed with steel-rails and sleepers, having a gauge of 2ft., the weight of the rails varying from 10lb. to 20lb. per yard. The latter weight was used in the one exhibited. The railway exhibited was laid down in the form of a link, the sharpest curve having a radius of 16ft., and in this link-railway the locomotive engine, hauling about eight trucks, was continually going round. The trucks are made in two sizes—one capable of holding one cubic yard, and the other capable of holding three-quarters of a cubic yard. There are trucks specially constructed for carrying rails and sleepers and also timber during the time the railway is being constructed, and, from the prices quoted by the manufacturers, one mile of railway per day can be laid down in ordinary country.

Cost of Railway.

The cost of rails and sleepers is as follows: Rails, 20lb. per yard (Fig. 65) with steel sleepers of pattern marked A (Fig. 66), 6s. per yard of railway, or £528 per mile; same rails with steel sleepers, pattern B (Fig. 67), 5s. 8d. per yard, or £498 13s. 6d. per mile. This price includes sleepers fish-plates, and bolts, &c. The same rail without sleepers but including fish-plates and bolts is 4s. per yard running, or £359 6s. 8d. per mile.

With 14lb. rails (Fig. 68) and sleepers, pattern A, the price is 4s. 9d. per yard, or £418 per mile; with sleepers, pattern B, the price is 4s. 6d. per yard, or £396 per mile. These prices include every necessary for making the railway in perfect working-order. Without sleepers the price is 3s. 2d. per yard, or £278 13s. 4d. per mile.

With 12lb. rails (Fig. 69) (these rails are only suitable for underground road, where there is light traffic), with sleepers of pattern A, the price is 4s. 6d. per yard, or £396 per mile; with sleepers of B pattern, 4s. 3d. per yard, or £374 per mile; with sleepers of C pattern (Fig. 70), 4s. per yard, or £352 per mile; without sleepers, 3s. per yard, or £264 9s. per mile.

With 10lb. rails (Fig. 71), with sleeper B, the cost is 3s. 9d. per yard, or £330 per mile; with sleeper C, 3s. 8d. per yard, or £322 12s. 6d. per mile; without sleepers, 2s. 6d. per yard, or £220 per mile.

To analyse the cost of these rails, to take the price without sleepers but including fish-plates and bolts, the cost is from £10 to £12 per ton.

Cost of Steel Sleepers.

Pattern A, for 20lb. rails, £168 13s. 4d. per mile; pattern B, £138 19s. 10d. per mile.

Pattern A, for 14lb. rails, £139 6s. 8d. per mile; pattern B, £117 6s. 7d. per mile.

Pattern A, for 12lb. rails, £131 11s. per mile; pattern B, £109 11s. per mile; pattern C, £87 11s. per mile.

Pattern B, for 10lb. rails, £110; pattern C, £102 12s. 4d.

All fish-plates, dogs, and bolts, if purchased separately, cost £16 per ton. The difference in the cost of sleepers is in proportion to the width: for instance, sleeper of A pattern is 5⅛in. wide, sleeper of B pattern is 4½in. wide, sleeper of C pattern 3⅝in. wide.

There is also every description of points and crossings, and crossings and turnouts (either bent or straight), and the steel sleepers are made suitable for each description of crossing, &c. The prices of these are as follow:—

B crossing with point-blades, Figs. 72 and 73, with straight track and curve to right, 16ft. 5in. in length, £6; the same crossing, 8ft. 2½in. long, £3 10s.

Compound crossing, Fig. 74, 16ft. 5in. long, £7.

Crossing for two even-curved tracks—to the right and left—Fig. 75, 14ft. 9in. in length, £6.

Crossings and turnouts, Fig. 76, 16ft. 5in. long, £6; Fig. 77, 8ft. 2½in. long, £3.

Portable turnout with curved span, Fig. 78, 16ft. 5in. in length, £3 10s.

Fixed crossings of track, Fig. 79, and right-angle crossings for wheels with one flange, £2 10s.; ditto, Fig. 80, ditto, £2 5s.

Bridge-crossing for wheels with one and two flanges, Fig. 81, £3.

Universal-joint span of rail, Fig. 82, £2 12s. 6d.

Road-crossing, 9ft. 10in. long, being portable in two pieces, Fig. 83, £3 5s.

Fixed turntable built in, Fig. 84, £7 7s.

Portable turntable of heavy construction, Fig. 85, £6 16s.; ditto, light construction, Fig. 86, £5 15s.

Portable bridge-turntable, Fig. 87, £9 15s.

Shifting-plate, Fig. 88, £2 15s.

Wagon-wheels and Axles.

The wagons are made in two sizes of steel plate: one carries a cubic yard of material, and the other three-quarters of a cubic yard. All the wagons are made to tip sidewise, the ends of the wagon being straight or vertical, and the sides are sloped in a triangular shape, but in place of having a sharp apex at the bottom they are rounded. The body of the wagon is hinged on a steel frame, and can be either tipped or held in any position by a handle and rack. When the wagons are empty they are placed on an angle towards the side the workman is engaged in filling them, and when they are about three parts full they are brought up to their vertical position and loaded up. The reason of their being made on this principle is that the workmen have not to lift the material to any height in loading them until they are nearly filled. The prices of these wagons, with steel frames, axles, and wheels, are as follow:—

For wagon holding 1 cubic yard, £10 16s.; ditto holding ¾ cubic yard, £9 12s.

Fig. 89 shows the wagon in position for loading—at least three parts full.

Fig. 90 shows the wagon when loaded.

Fig. 91 shows the position of the wagon when emptied or tipped.

The prices of the axles and wheels depend on the diameter of the wheels, the largest of which are 15½in. in diameter, and they are made with either one or two flanges as may be required. The prices are as follow:—

Fig. 92 shows a set of wheels with one flange fixed on round axle turned for outside bearings, 2ft. gauge, 12in. in diameter, £1 8s. 6d.; ditto, 15½in. in diameter, £1 15s.; ditto, 11in. in diameter, £1 5s.

Fig. 93 shows a set of wheels one of which has a flange on one side, and one a double flange and axles turned for inside bearings. Price for wheels and axles for 2ft.-gauge wheels 15½in. diameter £1 15s., and 12in. diameter £1 8s. 6d.

Fig. 94 shows a set of wheels with double flanges, one of the wheels being keyed on to the axle and the other revolving on the axle.

The wheels as shown in Fig. 94 are slightly more expensive than the quotations given; but the difference is so little that it is not worth mentioning.

Fig. 95 shows a set of wheels fixed on square axles, having hobs in the axle so that it can be bolted to the body of the truck; having the wheels loose, revolving on the axles. These, with wheels 11in. in diameter, cost £1 5s. 6d. per set.

When brakes are fitted on to the trucks it makes an additional cost of £3. Figs. 97 and 98 show the wagon without the brake and with a screw-brake fitted to it.

There are also timber-carriages constructed on the principle of the Pullman car, having a swivel truck-body at each end (see Fig. 99). These are constructed with iron or steel frames, having a carrying capacity of about 150 cubic feet of timber. One of these carriages costs £15 15s.

The Boehmer Union Locomotive Works, Krauss and Co. (Limited), Munich, where these specially mineral railways and trucks are constructed, make a specialty of making small tank-locomotives for working on these lines, having gauges from 18in. to 30in. The water for supplying the boiler is carried underneath on the tank-frames. The maker claims the following advantages for them as compared with other engines for the same purpose: Lower centre of gravity, less weight in proportion to the heating-surface and traction-force, greater strength and solidity of the whole construction, increased steam-pressure, suspension on three points.

The boilers are made of Krupp boiler-plates, with copper fire-box. Axles, tires, springs, connecting- and coupling-rods, guide-bars, piston- and valve-rods, and crank-pins are made of cast steel. All the bearings are of gun-metal lined with antifriction metal. The engines are fitted with spark-arrester chimneys, and are made to burn any description of fuel, and are supplied with one pump and one injector (see Fig. 100). The prices of these locomotives are as follow:—

Diameter of Cylinder.	Length of Stroke.	Diameter of Wheels.	Weight.	Horse-power.	Price f.o.b.
In.	In.	Ft. in.	Tons.		£
3½	6¼	1 3⅛	1½	5	300
4½	6¼	1 3½	3	10	350
5½	9½	1 11	4½	20	400
6½	11	2 1	6	30	450

The pressure of steam on the boiler on the engine that was working at the Exhibition was about 22lb. to the square inch.

This railway rolling-stock and locomotive has much to commend itself for a country like New Zealand, where material has to be hauled from the mine to the crushing-batteries for a considerable distance round steep sidelings, necessitating sharp curves. It is also useful for coal-mines where the coal has to be brought some distance to the railway.

Mining-plant of this description will come more into use in the future than it has in the past with regard to the haulage of auriferous and argentiferous ores from the mine to the reduction-works. In the past companies have striven to erect a stamping-battery as near the mine as possible; but the days of this class of machinery for reducing ores are drawing to a close. Plants will be erected capable of extracting a far larger percentage of the metal from the ore, but as these plants will involve a considerable outlay there will be fewer of them erected, and the ore will be brought from greater distances, and a railway such as described is the very thing required to convey the ores from the mines to the reduction-works. Such a railway and rolling-stock would have saved the County of Piako a considerable amount of money at Te Aroha, and in all probability the claims that were from time to time given up would have been working at the present time.

NEW EXPLOSIVES.

Rackarock.

This explosive was exhibited in the American Court at the Melbourne Exhibition. It is said to be more powerful than dynamite, and much cheaper. It is formed by the union of two substances, one of which is a solid and the other a fluid, both in themselves inexplosive until combined immediately before using. The exhibitors state that there are no deleterious fumes and very little smoke from the explosion of this compound, that it does not freeze in the coldest weather, and that it is equally effective in wet and dry holes.

Lieutenant George McC. Derby, in writing about this explosive in connection with the Hellgate explosion at New York, states: "The experiments with this explosive at Hellgate proved the efficacy of the means previously discovered at Flood Rock of firing long narrow charges of rackarock, an explosive so inert that a pistol-bullet can be fired into it at a short range with impunity. It was afterwards adopted for a large blast, its strength under water being greater than No. 1 dynamite, while its cost is but little more than one-half, and it need not be made explosive until such time as it is required to be taken into the mine." It is exploded by either an electric fuse or ordinary safety fuse and quintuple-force cap, in the same manner as dynamite.

There was no opportunity of seeing this explosive tested; but if it possesses the strength and safety that the manufacturers claim it will be a useful explosive in working mines—at least, it is well worthy of a trial. It is well known that if dynamite is not properly exploded—that is, if combustion is not produced—the fumes are highly deleterious to the health of the workman; and in a great many instances, and especially when dynamite is in a frozen condition, proper combustion is not produced. It would require a detonator about three times as strong to properly explode a charge when frozen as when it is in a soft state.

In order to render the cartridges of rackarack explosive they have to be dipped in the liquid for a few seconds. For this purpose a small wire basket is provided, and as many cartridges as required placed in it, and immersed in the fluid. The dipping of the cartridges, the manufacturers state, is an important operation, but it may be done by any miner of ordinary intelligence; yet they state that it is better that a reliable man should be selected for the operation, which only takes a few minutes, as if a careless man allowed the cartridges to remain too long in the liquid it would use more of the fluid than necessary, and thus increase the volume of smoke from the explosion. In this matter as well as in many others only ordinary intelligence is required to insure success. The merits claimed for this explosive by the manufacturers are that it is more powerful than dynamite, and never requires to undergo the process of thawing; that it is perfectly safe to handle or transport in its uncombined form; that it is cheap; and that its fumes are innocuous.

The solid is made up in 80lb. cases of cartridges, and the liquid in 40lb. tins, the proportion of liquid to solid being 20lb. of liquid to one case of 80lb. of cartridges. The holes are charged and fired in the same way as with dynamite and gelatine, with a detonating cap; but premier sextuple caps are used.

This explosive has been tried by several of the managers of the mines at Ballarat, in Victoria, and also at Gympie, in Queensland, and from what information could be obtained regarding these trials they were very satisfactory. The Ballarat *Evening Post* of the 17th August last states that "during this week experiments were conducted with this explosive at the Sulieman Pasha, Star of the East, and Prince Regent Mines, and in each instance produced satisfactory results. Some months previously a trial was made at the North Band and Barton Mine, and so satisfactory were the results there that it has been used ever since. The miners get out more stone by its use, while the company's expenditure is reduced."

The following is a copy of the directions given by the manufacturers for preparing the cartridges for use:—

"1. Pour the liquid from the tin into the dipping-can; place the cartridges (laid in tiers) in the wire basket; then lower the basket into the liquid and allow it to remain, in cool weather, for the following periods: For $\frac{3}{4}$in. cartridges, about 4 seconds; for 1in. cartridges, about 6 seconds; for $1\frac{1}{4}$in. cartridges, about 8 seconds; for $1\frac{1}{2}$in. cartridges, about 10 seconds. Lift the basket out of the liquid and hold it over the can a moment to drain. After saturating the required number of cartridges pour back into the tin any liquid remaining, and cork tightly. Enough may be dipped for a single blast, or two days' work, as required. As a test that the cartridges are properly soaked, a dry core, about $\frac{1}{16}$in. diameter, should be left in the centre immediately after soaking. In hot weather it is advisable to let the cartridges absorb more oil, as they are then less sensitive. Over-soaking increases the amount of smoke, but not the power of the explosive.

"2. The bore-holes should be about $\frac{1}{8}$in. larger diameter than the cartridge, so as to admit of the charge going to the bottom without ramming. A cartridge should never be forced into too small a hole.

"3. A wooden rammer must in all cases be used, and the cartridge pushed, not forced by tapping, into the bore-hole. Metal rammers must never be used.

"4. When two or more cartridges are used in a blast they should be pushed easily to the bottom of the hole, and then squeezed firmly without hitting. When this is done the cartridge containing the detonator, with fuse attached, should be pushed easily down, so as to rest on the charge. Sand, water, or clay tamping should be used, so as to confine the charge as much as possible. The more rackarock is confined the better the result of the blast.

"5. To prepare the detonator cartridge, put the fuse into the detonator, taking care to leave some space between the end of the fuse and the fulminate in the detonator; then take a soaked cartridge, untie one end, and push a stick the size of the detonator down through the centre of the cartridge to within about 2in. of the bottom; withdraw the stick, and insert in the hole thus made the detonator, with fuse attached; retie the end of the cartridge round the fuse. It is then ready to place in the blast-hole.

"6. The cases of cartridges must be kept dry. Should the ingredient get wet it must be thoroughly dried before being saturated in the liquid."

TASMANIA.

Mount Bischoff Tin-mine.

This mine is situate on the western slope of Mount Bischoff, near the township of Waratah, about forty miles from Emu Bay, at an elevation of considerably over 2,000ft. above the level of the sea. The tin-ore is mixed among a great depth of wash-drift, which appears to be

SECTIONS OF CAST-STEEL SLEEPERS.

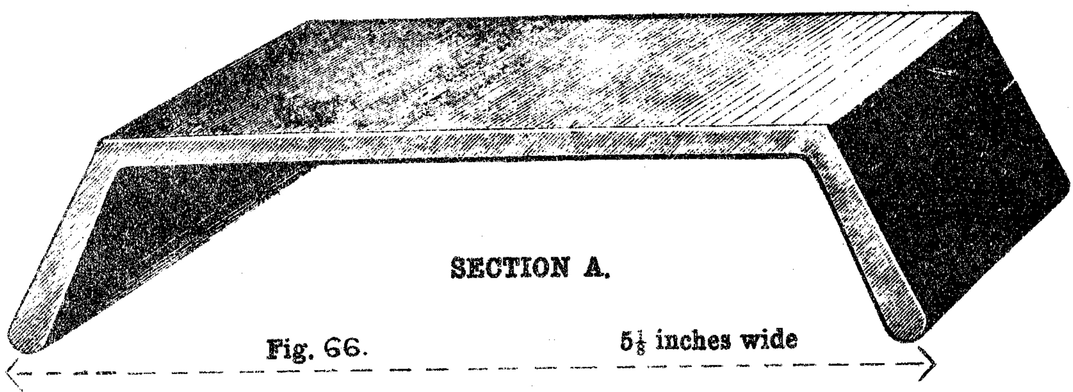


Fig. 66.

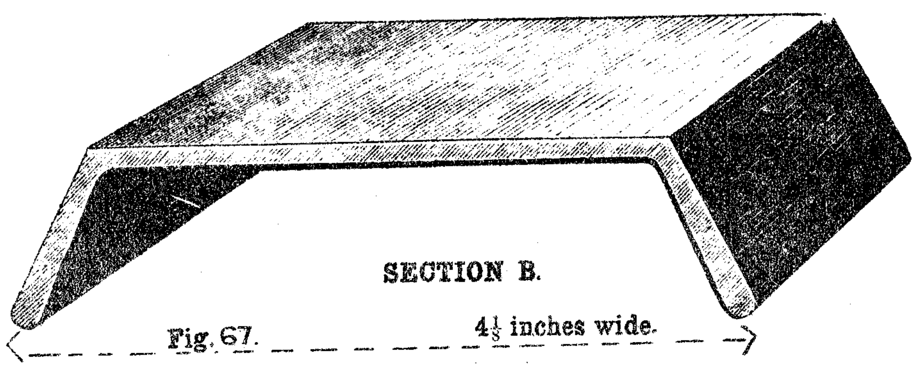


Fig. 67.

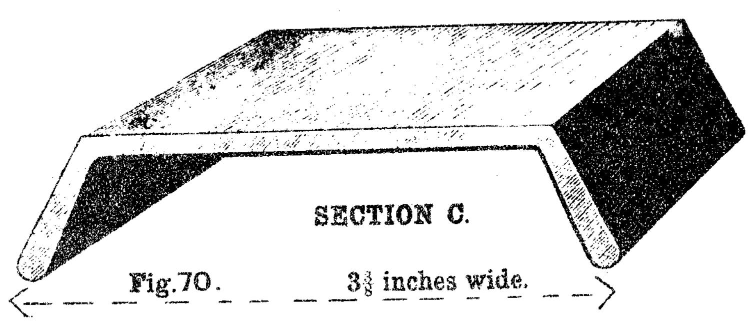


Fig. 70.

SECTIONS OF CAST-STEEL RAILS.

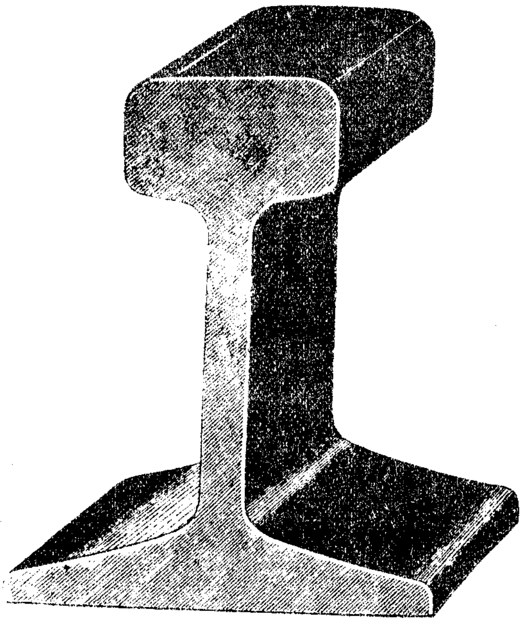


Fig. 65.
Section A, 3 inches high.

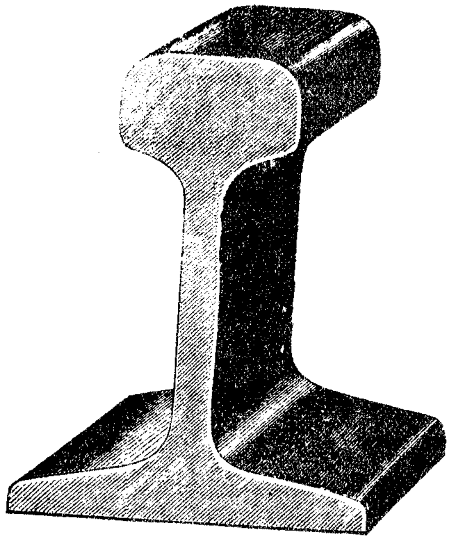


Fig. 68.
Section B, 2½ inches high.

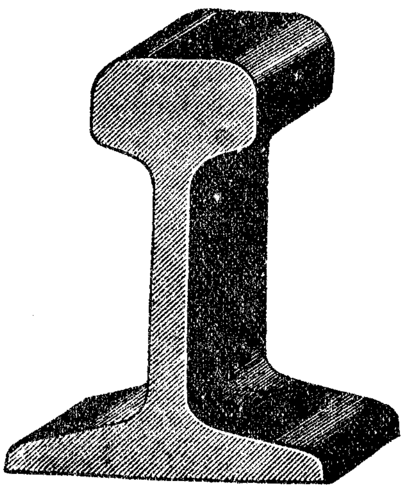


Fig. 69.
Section W, 2⅜ inches high.

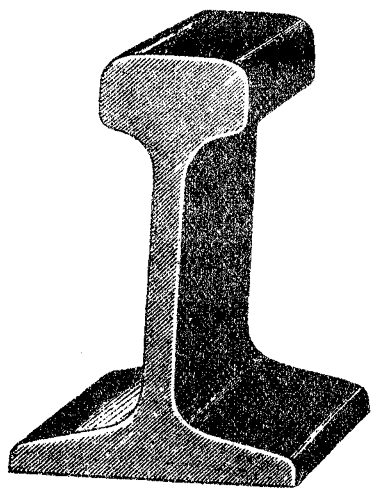


Fig. 71.
Section C, 2¼ inches high.

B. CROSSINGS WITH POINT BLADES.

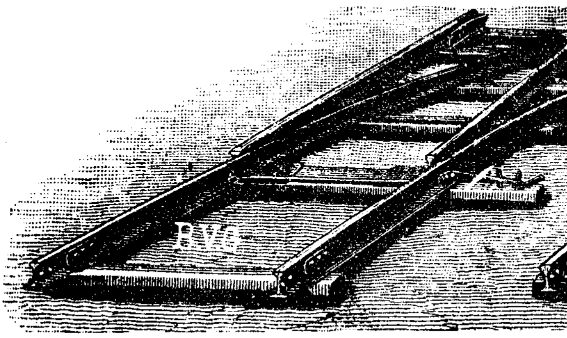


Fig. 72. **Straight track and curve to the right, 16 feet 5 inches in length.**

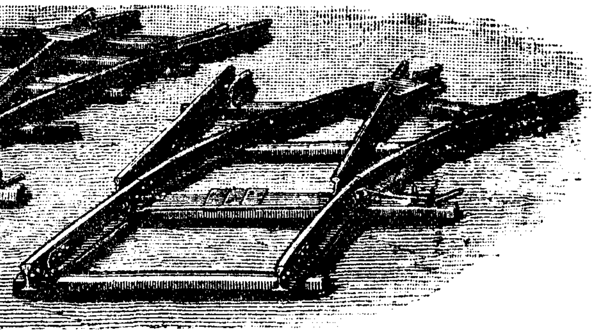


Fig. 73. **Straight track and curve to the right, 8 feet 2½ inches in length.**

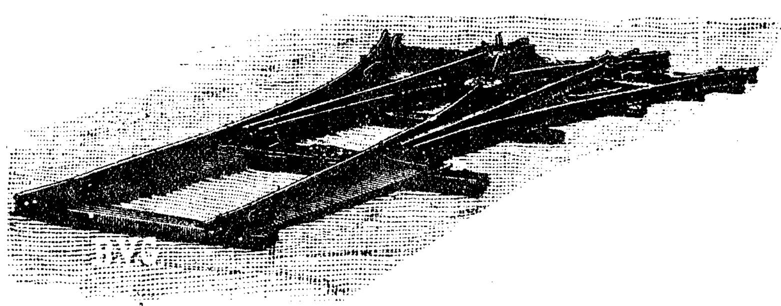


Fig. 74. **Compound Crossing, 16 feet 5 inches in length.**

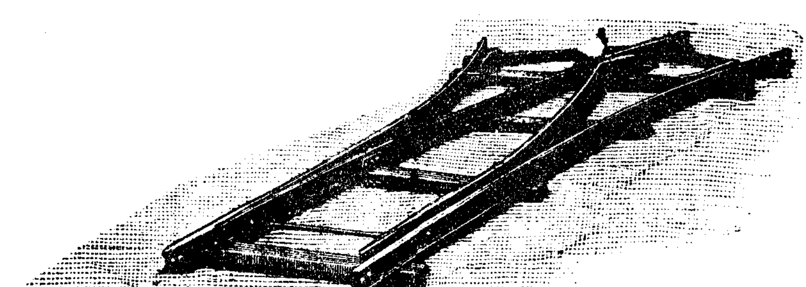
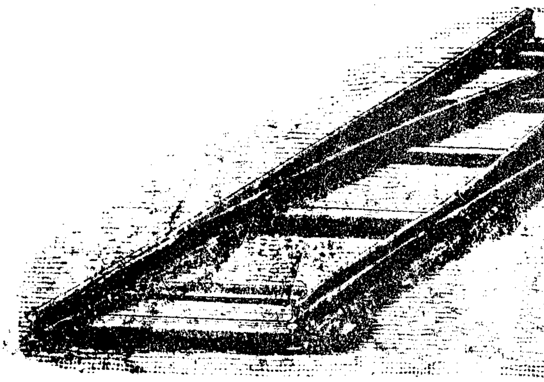


Fig. 75. **Crossing for two even-curved tracks (to the right and left), 14 feet 9 inches in length.**

CROSSINGS AND TURNOUTS.



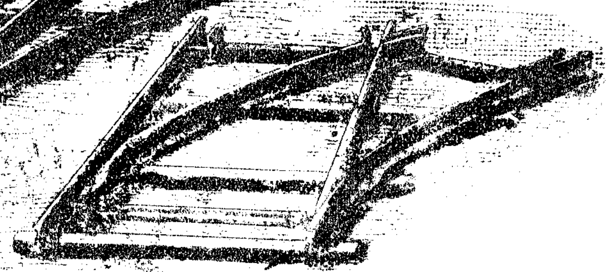


Fig. 76. 16 feet 5 inches long. **Fig. 77. 8 feet 2½ inches long.**

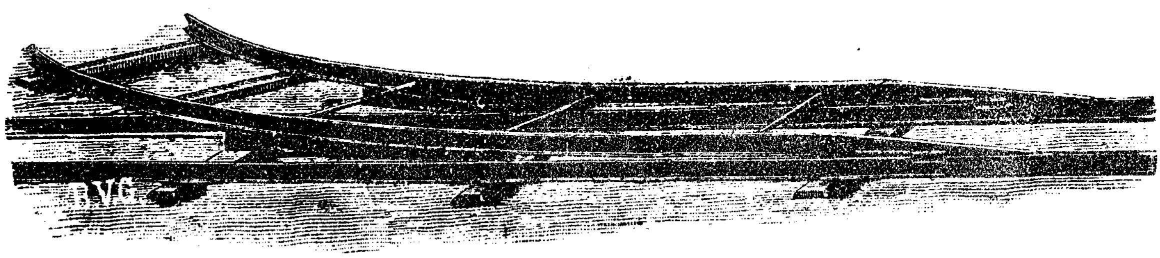


Fig. 78. Portable Turnout, with curved span, 16 feet 5 inches long. *(Very handy.)*

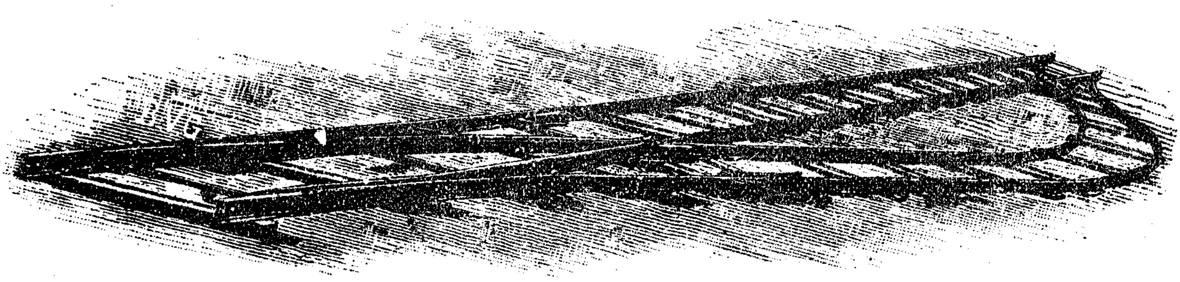


Fig. 78^A Turnout with Switches to the right and left.

C. FIXED CROSSINGS OF TRACK.

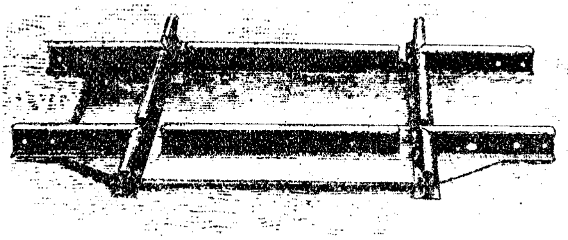


Fig. 79. Right-angle Crossing for wheels with one flange.

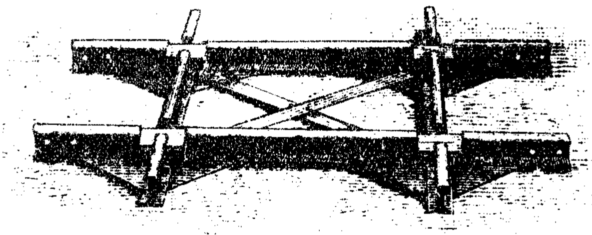


Fig. 80. Right-angle Crossing for wheels with flanges.

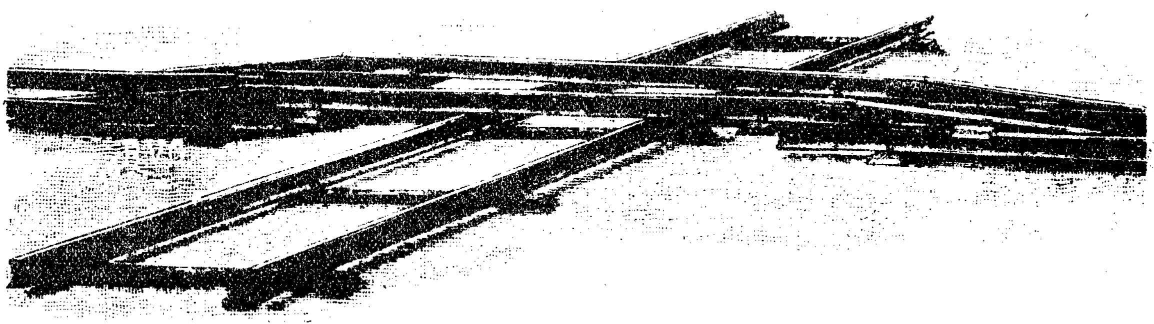


Fig. 81. Bridge Crossing for wheels with one and two flanges.

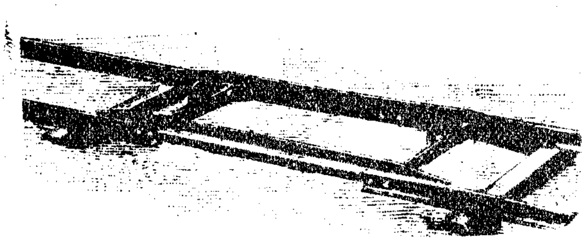


Fig. 82. Universal Joint Span of Rail.

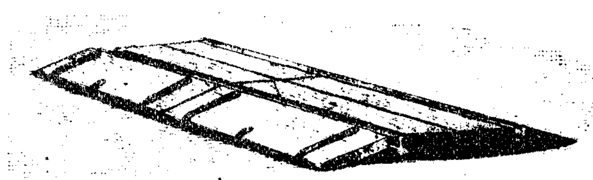


Fig. 83. Road Crossing, 9 feet 10 inches long. Portable, in two pieces.

D. TURNTABLES.

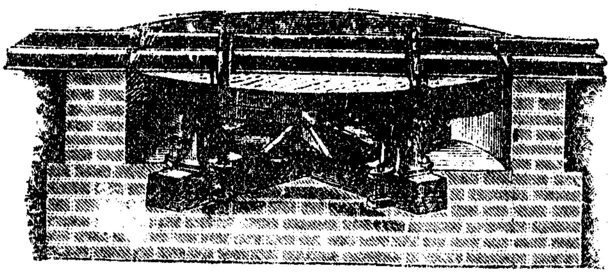


Fig. 84. Fixed Turntable, built in.

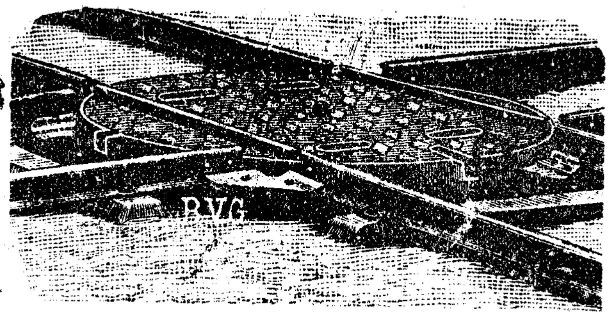


Fig. 85. Portable Turntable, of heavy construction.

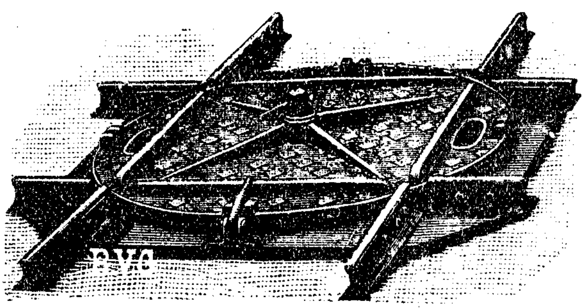


Fig. 86. Portable Turntable, of light construction.

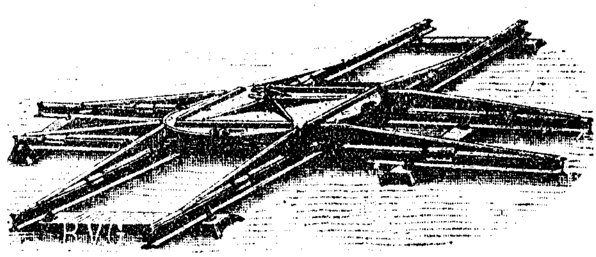


Fig. 87. Portable Bridge Turntable.

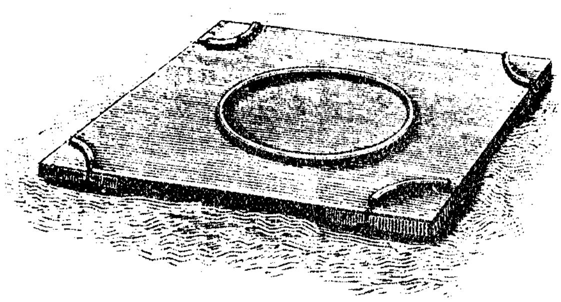


Fig. 88. Shifting Plate.

C. TRUCKS.

Steel Side-tip Waggons, with brackets rivetted to truck body.

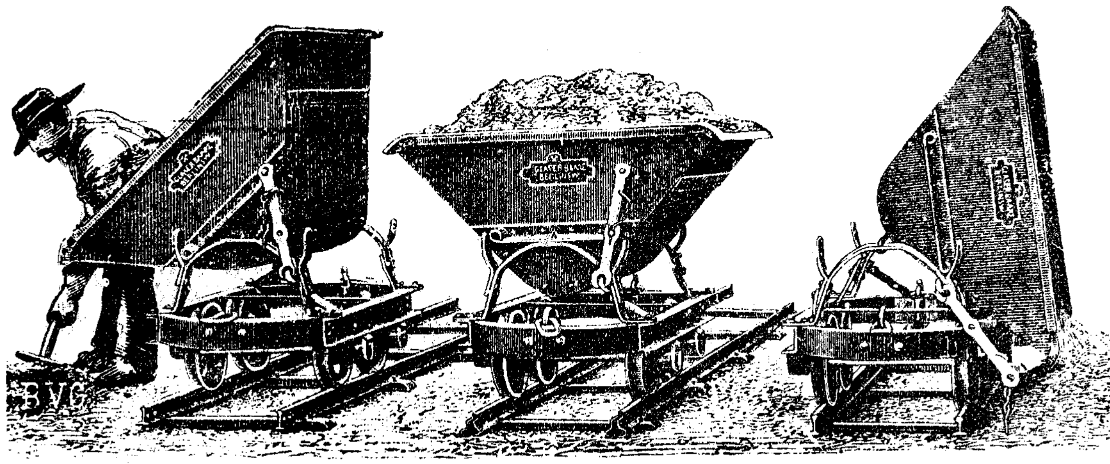


Fig. 89. **Position of waggon when loading.**

Fig. 90. **When loaded.**

Fig. 91. **When unloading**

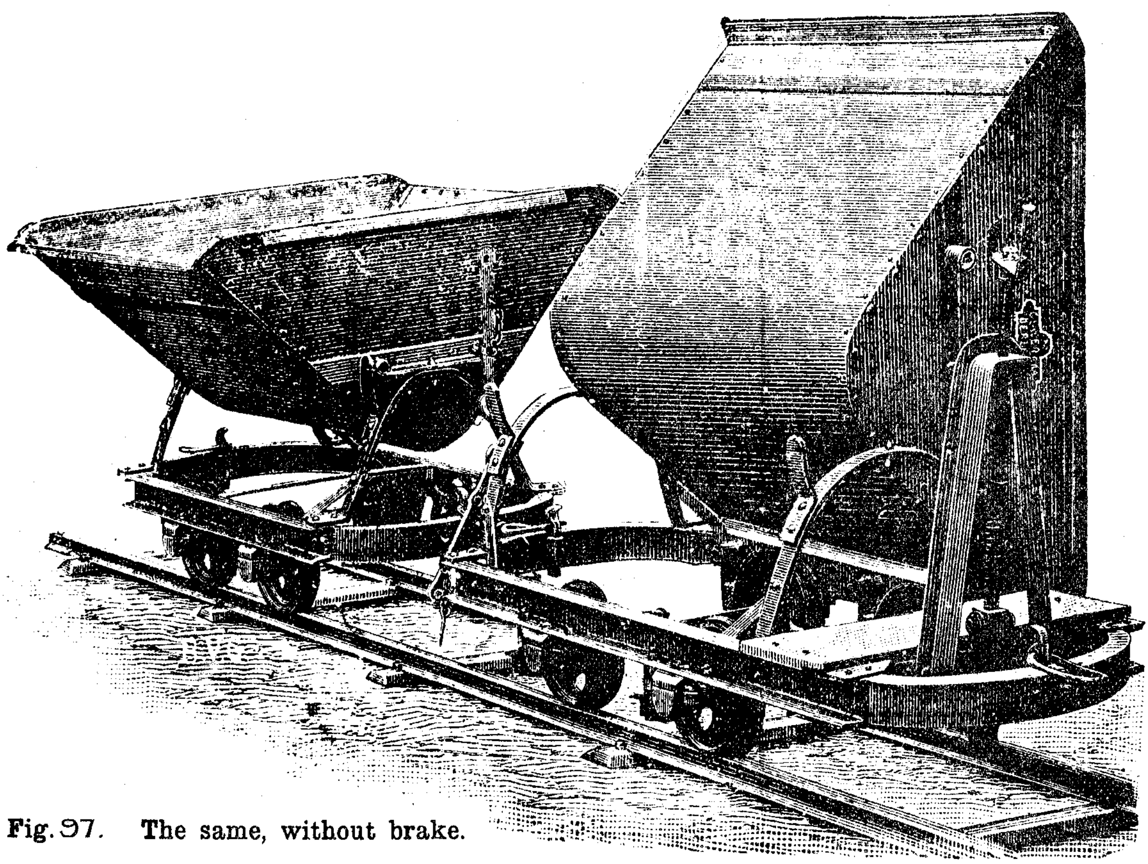


Fig. 97. The same, without brake.

Fig. 98. The same, with screw brake.

A. WHEELS AND AXLES.

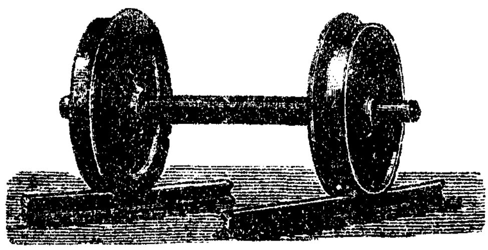


Fig. 92. Set of Wheels, with one flange, fixed on ○ axle, turned for outside bearings.

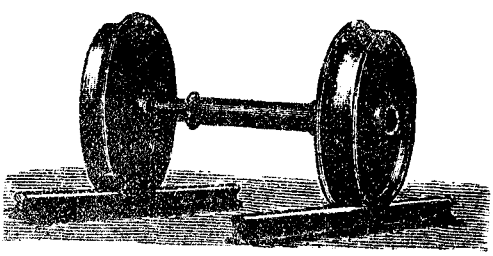


Fig. 93. Set of Wheels, with one flange, fixed on ○ axle, turned for inside bearings.

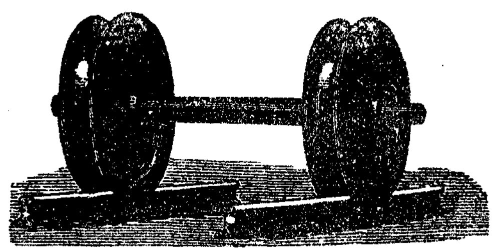


Fig. 94. Set of Wheels, with two flanges and ○ axle, turned for outside bearings. One fixed and one loose wheel.

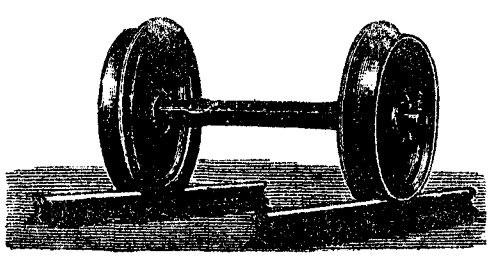


Fig. 95. Set of Wheels, with one flange and □ axle, with bolt holes for fastening to truck. Both wheels loose.

Tank Locomotives (Krauss System) for Portable Railways.

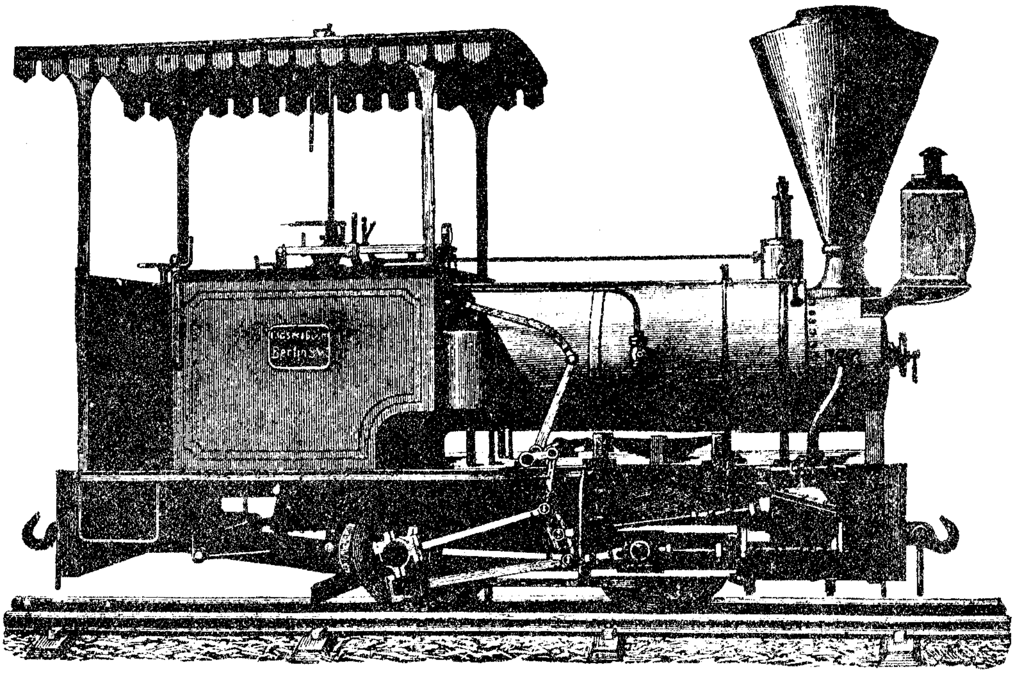


Fig 100.

confined to a basin having a diameter of from 1,300ft. to 1,400ft. The drift is full of very hard porphyry boulders and quartz, and in these, as well as in the wash-drift, the cassiterite or tin-ore is in places thickly disseminated, occurring generally in vertical layers or seams. The whole deposit assumes the form of an inverted cone, and has the appearance of an extinct crater or geyser. The ground has been proved to contain rich ore to a depth of about 200ft. below the surface, and the bottom of the basin has not yet been reached.

The whole of the deposit is worked in a face from the surface downwards, the fall of the mountain allowing levels being put in to enable this to be done. In many places the wash-drift contains large quantities of decomposed iron-pyrites, which, when mixed with the tin-ore, makes it difficult to treat properly. When this material is met with it is picked out from among the wash-drift as much as possible before sending it to the reduction-works.

The ground is taken up all around this basin which the Mount Bischoff Company hold; but, although there is a certain quantity of tin-ore to be got, no other ground has been found in the district that contains so rich a deposit. Some very rich ore is met with in this basin—indeed, large blocks of pure cassiterite are sometimes obtained—and there are also great heaps of poor ore which have from time to time been stacked as too low-grade to pay with the appliance in use at the time, but which is being now turned over with the view of resorting it to crush the richest of the stone. The company have the limits of their rich ground now pretty well defined, and, although they have a number of years' work to look forward to, the circumscribed area where the rich ore is found will gradually get worked out. It is only by having an extensive reduction-plant that the mine pays so handsomely for working; for, although there are rich blocks of ore in the mine, there is a great deal of poor ore to deal with in having to work the whole in a face.

A large quantity of the drift is washed in boxes at the mine, and by this means concentrated to a certain degree before filling into the trucks to take to the reduction-works. The muddy water from these boxes flows into the creek, where there are settling-ponds, and the slime from these ponds is treated on ringtail tables some distance lower down the creek than the dressing-works. Annexed are sketches (Figs. 101 to 107) showing some of the ore-dressing machinery.

The plant at this company's reduction and dressing works consists of 1 stone-breaker, 75 heads of stamps, 30 classifying pyramidal boxes, 30 jiggers, 17 Kayser buddles, 23 single-convex revolving-tables, and 12 double-convex revolving-tables, exclusive of settling-boxes. The whole of the works are lighted up with the electric light, having lamps of the Swan pattern. The company have also workshops and a small foundry, where they can make small castings and do the whole of their repairs on the ground.

After the ore is dressed at the works it is forwarded by rail to Emu Bay, and thence shipped to Launceston, where the company have smelting-works under the superintendence of G. J. Latia. According to the company's half-yearly statement, the quantity of clean ore produced at the works for the six months ending the 30th June, 1888, was 1,266½ tons, and the total quantity of ore smelted during the same period was 2,329½ tons, which yielded 1,605½ tons of tin of an average assay-quality of 99·86 per cent. The expenditure on the wages, and salaries, and working-expenses during the half-year was £29,756, while the value of the tin produced amounted to £128,460, thus leaving a clear profit of £98,704, of which £78,000 was paid in dividends. There are 12,000 shares in this company of £5 each, and £29,600 of the capital paid up; and the total amount of dividends paid has exceeded £900,000. The total quantity of dressed ore produced at the mine since the formation of the company has been 29,880 tons.

Reduction and Dressing Works.

The ore is conveyed from the mine to the ore-reduction and dressing works by a light railway, having a locomotive engine to haul the wagons, which hold about a ton of ore in each. When the wagons arrive at the dressing-works they are emptied into a large hopper, and from this hopper the ore is conveyed to the rock-breaker, and thence to the stamping-battery, where it is crushed until it passes through a woven grating having from 144 to 169 holes to the square inch. The great object here in crushing the ore coarse is that it does not make so much slime, and solid ore-veins are better adapted for separation when they are crushed coarse.

Pyramidal Boxes.—The first thing to do when the ore arrives at the battery is to classify it—the richer the ore the coarser it is crushed—and the next thing after crushing is a proper classification of the ore, regardless of the size to which it may be reduced. This is described by Mr. Kayser, the manager of the company, as follows: "The classification can be accomplished for the coarser kind of ore by revolving classifying-trammels, or a series of sieves one above the other, worked by a shaking motion and by percussion, and for finer sands by self-acting classifying-boxes—'spitzlutten.'"

Mr. Kayser prefers rolls for crushing very rich ore, but for fine crushing revolving stamps of about 6cwt. each if the ore-stuff be moderately soft, but for hard ore the stamps should be heavier. In crushing tin-ore the same principle has to be observed as that in crushing auriferous and argentiferous ore—namely, there should never be more than about 2in. in depth of material in the mortar, or else the stamps will not do efficient work. In crushing tin-ore the object is to prevent slimes as much as possible, and to accomplish this the manager uses only as much water in the stamping-mortar as will cause sufficient splash to send the sand through the grating, and afterwards supplements it with such an additional quantity of clean water outside the mortar as may be necessary for a proper classification of the crushed ore in the pyramidal boxes or spitzlutten. These pyramidal boxes (see annexed sketch, Fig. 101) are based on the principle that, if material composed of particles of different size and density is exposed to a rising stream of water, the velocity of the stream may be so regulated that particles of a certain size and density sink to the bottom, while the lighter material is carried upwards with the stream; and by repeating this process, or passing the material from one set of pyramidal boxes to the other, with a gradually-decreasing velocity of the rising stream each time, the material can be separated into as many classes of fineness as is considered desirable, and the remainder of the sand and sediment left in the water is then conveyed to a conical settling-trough. This trough or box is constructed so that the precipitated material can be drawn off by small pipes, and carried direct to a revolving-table, one table being sufficient for every ten head of stamps.

At the Mount Bischoff plant there are two sets of pyramidal boxes for each five heads of stamps—one for the coarser material and one for the finer sand, and also one or two large boxes for collecting the slimes. The manager states that the chief thing to observe in working the pyramidal boxes is regulating the pressure of the water which forms the rising stream, and that it is advisable to draw this supply from a tank or reservoir at a good elevation, and regulate the supply by a tap close to the box. The water must be clear and free from rubbish, as any rubbish would be liable to get to the bottom and tend to choke the 1in. gas-pipe which leads the sand from the bottom of the pyramidal boxes to the jiggers, and precaution must also be observed regarding this with the water and sand entering the boxes from the battery. The top of these pyramidal boxes should be covered with a movable lid, so that nothing can drop in by accident. A loose wire screen, with the wires three-sixteenths of an inch apart, will answer the same end.

Jiggers.—The sand from the pyramidal boxes now enters the jiggers (see Fig. 102) through the 1in. gas-pipe above referred to, the jiggers being placed at a lower elevation, so that the water and sand pass into them by gravitation. The same number of jiggers are used as there are pyramidal boxes, having screens and bedding suitable for the fineness of the material that enters them. The false bottom on top of the sieve is made of crop-tin, and is slightly coarser than the holes in the sieve, so that none of it can pass through. The thickness of the false bottom depends on the quality of the material treated. If the material be rich in tin-ore, the bedding is made thin, but for poor sand a thicker bedding is required. The proportion of the first and second sieves should be as two to one, to make sure that no valuable ore is allowed to pass out. When poor stuff is treated it should be subjected to a second jigging process, as it is only in the case of good clean ore that the product of the first sieve is in a fit state to send to the smelting-works.

The mesh of the sieves should be in due proportion to the material to be treated, in order that particles of ore may pass easily through. The wire should be well woven and well stretched upon the frame, so that the sieve is firmly fixed.

The speed of the jiggers also depends on the fineness of the ore-material to be operated on, and varies as much as from 60 to 220 strokes per minute. The ore-material passing through a mesh at the battery of 144 holes to the square inch requires from 144 to 160 strokes per minute; while that passing through a mesh of 169 holes to the square inch will require from 200 to 220 strokes: the finer the ore-material the quicker the jiggers require to be worked. The stroke of the jigger is given from an eccentric, and is about ⅛in.

Concave Buddles.—At the Mount Bischoff ore-dressing works there are two buddles used, one for the slimes and one for the coarser sand. The latter is what is here termed a Kayser buddle (see Fig. 103), but the principle is similar to the well-known Munday buddle, with some minor alterations, such as having the revolving arms, which carry the scrapers, hinged near the centre, underneath the main arms, while the outer extremity is held in the exact position by means of a screw. The end of this screw is made fast to the hinged arm and passed through the main arm, having a small wheel with a screw boss on top, which raises and lowers the hinged arm to the exact height required; also, instead of brushes, used in the Munday buddle, the Kayser buddle has scrapers. These are the principal alterations made in a buddle invented by Munday many years ago, but these alterations are decidedly an improvement. The Kayser buddle is about 20ft. in diameter, concave in shape, having an opening in the centre of about 1ft. 10in. in diameter; and the bottom of the buddle has an upward inclination of 1 in 12 towards the periphery, having sides slightly splayed outwards towards the top, and 2ft. in height.

The bottom of the buddle is made of 1½in. tongued-and-grooved timber, and the sides of 1in. timber of the same description. The bottom and sides are all dressed and closely jointed, so as to be thoroughly watertight and perfectly smooth on the inside. In the opening at the centre there is a wooden framing, on which is bolted a cast-iron step to carry the end of an upright shaft. There is also a framing made outside the periphery of buddle, having a beam across the buddle overhead, to which is bolted a plummer-block to carry the upper end of the vertical shaft, and on the upper side of this beam a cross-beam is notched and bolted for carrying the plummer-block to support the horizontal shaft, on which a pinion is placed. This framing stands about 6ft. above the top of the buddle. There are two cast-iron bosses keyed on to the vertical shaft, one about 1ft. above the bottom of the buddle, and another about 4ft. 6in. above the bottom. The lower boss has eight recesses for carrying the inner ends of the main arms, and the upper one is made in the shape of a basin, having a boss in the centre into which there is screwed eight gas-pipes, the ends of which extend to the periphery of the buddle. Also, from this cast-iron basin wrought-iron stay-rods are fixed to support the outer ends of the main arms and the gas-pipes referred to (as will be seen from the annexed sketch, Fig. 104). The lower arms, eight in number, are hinged underneath the main arms at about 1ft. 4in. from the centre, and supported by a screwed rod going through the main arm at the outer end, as already referred to. The main arms are also stayed or braced horizontally with wrought-iron tie-rods near their extremities, and in two of these tie-rods there are union screws, one on the opposite side of the buddle from the other, to tighten the rods, and by this means the arms are firmly braced together.

Two scrapers are firmly fixed on the lower side of the hinged arms, at such distances that one does not revolve in the same plane as the other. These scrapers are made of thin plate-iron, about 8in. long and 3in. wide, and they are placed about 2½in. apart, and stand at an angle of 45° to the radial arms, which are eight in number. The scrapers are divided along the bottom of the buddle as follows:—

First arm, centre between the scrapers 1ft. distant from periphery of bottom of buddle.
Second arm, centre between the scrapers 1ft. 7½in.
Third arm, centre between the scrapers 2ft. 3in.
Fourth arm, centre between the scrapers 2ft. 10½in.
Fifth arm, centre between the scrapers 3ft. 6in.
Sixth arm, centre between the scrapers 4ft. 2in.
Seventh arm, centre between the scrapers 4ft. 10in.
Eighth arm, centre between the scrapers 5ft. 7in.

These scrapers stand almost level with the bottom, and, being set on an angle of 45°, they tend to force the particles of heavy material always towards the periphery, or, at least, prevent those particles from being carried to the centre of the buddle, where the water and light sands are discharged. As the heavy concentrates settle on the bottom, these movable arms with scrapers are lifted by the screws already referred to, and by this means regulated to any height that may be required.

On the top of the vertical shaft there is a bevelled spur-wheel, which is driven by a bevelled pinion on a horizontal shaft, causing the arms to revolve at the rate of from six to seven revolutions per minute.

The material to be concentrated is carried by a launder into the cast-iron basin which is keyed on to the vertical shaft, and from there the water and sand is distributed at the periphery of the buddle by eight gas-pipes, which are about 1½in. in diameter. The heavy sulphurets lodge near the periphery, while the light sands pass down the bottom of the buddle, and are discharged over the circular riffle at the centre.

When working coarsely-crushed ore about six to seven hours will fill the buddle, but with finely-crushed ore it will run about twelve hours without cleaning out. About 8 cubic feet of water is required per minute to work one of these buddles. When the buddle is full of concentrated sand the clean concentrates are shovelled out for a certain distance round the buddle from the periphery, but on getting near the centre the concentrates which travel this length are mixed to a certain extent with the lighter sands, and are shifted to the periphery and the buddle set at work again. One buddle will concentrate the ore from ten head of stamps.

Convex Table.—For working very fine ore or slimes the convex table is used, which has proved the best concentrator for fine slimes that has yet been invented. Mr. Kayser kindly supplied me with the following description of this table (see Figs. 105 and 106):—

"The table is 16ft. in diameter, and built on a vertical shaft. The arms to receive the floor are bolted to a cast-iron centre, which is keyed to the shaft, as also is the centre at the bottom end of the shaft to receive the stays after it is finished. The floor is usually of double thickness of timber, well put together. The bottom planking is 2in. in thickness, while the top one is only 1½in.; but the latter must be hard, close-grained timber that will take a good polish, and when finished no join should be discernible, so that neither slime nor water will find any impediment in passing over its surface. The first planking is fastened with nails to the arms in the ordinary way, while the top planks are fastened with wooden dowels. This table when at work makes one revolution in every two minutes and a half. The feed as well as the clean water is supplied by a collar-launder of from 3ft. to 4ft. in diameter, made of sheet-iron, and hung over the centre within 2in. of the table-floor. The proportion of feed and clean water depends on circumstances, but when working clean ore 1 to 4 is a good proportion. As soon as the table is set to work the process is continuous, and requires no further adjustments except to see that clean water and slime are supplied in proper proportions. In nearly all cases it is advisable to hang three or four aprons touching the surface of the table at the outside periphery, by which means guttering on a small scale is prevented until the place is reached where the table completes its first revolution, and where the slimes are sufficiently clean to be washed off. To effect this a perforated pipe, or three or four jets specially constructed for this purpose, are used, and, as the water is supplied under a high pressure, and strikes the table at an angle of 45°, by the force of the water, confined by an apron in front, all the slime and water leave the table together, and pass into the outside collar-launder, to a division set apart for that purpose, from whence they are conducted by pipes into settling-pits standing underneath the table. The size and length of these launders depend upon the fineness of the slime treated. The finer the slime is the longer the labyrinth will have to be to effect a proper separation before the water is allowed to pass out into the waste-canal. The remaining portion of the outside launder of the table receives and carries off the waste."

In the proper working of the table a great deal depends on the construction, particularly upon the even and smooth surface, and, as the table is made in sixteen equal sections, it is very difficult to get a tradesman who can finish the top sufficiently accurately to make the slime and water spread equally all over its surface. Besides this, if the material operated on is poor, the slime is more difficult to collect in a quite clean state in the first process; in that case, it is best to work as for poor slime, and clean the same on a separate table. In using seasoned timber for these tables, great care is required to have the timber well soaked in water beforehand, to prevent the danger of its bulging after the table is finished and under the action of water. Latterly tables with a coating of Portland cement have given great satisfaction. The bottom planking can be nailed in a rough state to the arms, with a rim nailed round the periphery to the height or depth the concrete is to be laid—say from 1in. to 1¼in.; to increase the holding-power, clout-headed tacks are driven into the bottom planking and allowed to stick out about ¾in., and by this means hold the concrete firmly to the bottom planking. The first layer of concrete—about two-thirds of the thickness required—is a mixture of fine gravel and sharp sand, about two parts of this and one of cement; the next layer is finer and of better quality—about equal proportions of sand and cement, with a top-dressing of nearly all cement, well polished. If this is allowed to set well before being used it is all one can desire. The reworking of the slime can be effected either by having the cleaning-table on the lower floor or by lifting the stuff with a small pump after it is liquefied a second time. After this the tossing-tub will finish the process.

These convex tables have an inclination from the periphery towards the centre of 1 in 12, and two tables can be built on one shaft, one underneath the other. The latest tables constructed by Mr. Kayser are made on this principle, and also the convex tables that are being constructed at the Broken Hill Proprietary Company's mine, in New South Wales, to concentrate the slime from the crushing of the silver-ore.

The same class of machinery that is used for ore-dressing at Mount Bischoff is specially applicable to the Antimony Company at Endeavour Inlet. The loss of antimony by this company by their present process of concentration is perfectly appalling. The ore and slime in the tailings would soon pay for a proper plant, and make their property of considerably more value. These tables would also be a good concentrator for gold-ores, especially if they were finely crushed. The finer the material the better the tables would work, and they would answer admirably for final concentration of tailings from berdans after the ore had undergone two operations in crushing and had been amalgamated.

Tossing-tub.—This is a tub of the following description: It is about 4ft. in diameter and 2ft. 6in. in depth, having staves of 1½in. or 2in. timber, and slightly conical in form, tapering towards the bottom. Through the axis of the tub a hollow cast-iron cone passes, reaching a few inches above the top of the tub, being fastened by a flange to the bottom. A vertical shaft passes through this cone and rests on a journal underneath, and carries a bent yoke somewhat in shape of a horse-shoe, to which are riveted flat iron stirrers standing out horizontally when the yoke is in its place. On the outside of the tub two small sledge-hammers are placed, which work mechanically by pins in the bevelled gearing on the shaft.

When ready for tossing the tub is filled to nearly half its height with water, the stirrers are set in motion, making forty-eight revolutions per minute, and the ore is shovelled in near its periphery. When nearly full the yoke to which the stirrers is attached is lifted out, and the sand and ore are allowed to settle, while the hammers are set in motion, making about ninety-six strokes per minute on the outside of the tub, near the top of the staves, in order to facilitate the rapid settling of the sulphurets and sands. When the sands have settled the water is drawn off by an iron siphon, the skimmings are removed to a depth of about 2in. and thrown out as waste, and the remaining upper half of the sands is taken out and again tossed, while the lower half is almost clear concentrates. An appliance of this description is also used for final cleaning of gold-bearing sulphurets if they are to be treated by the chlorination process, which is usually performed in buddle-headings.

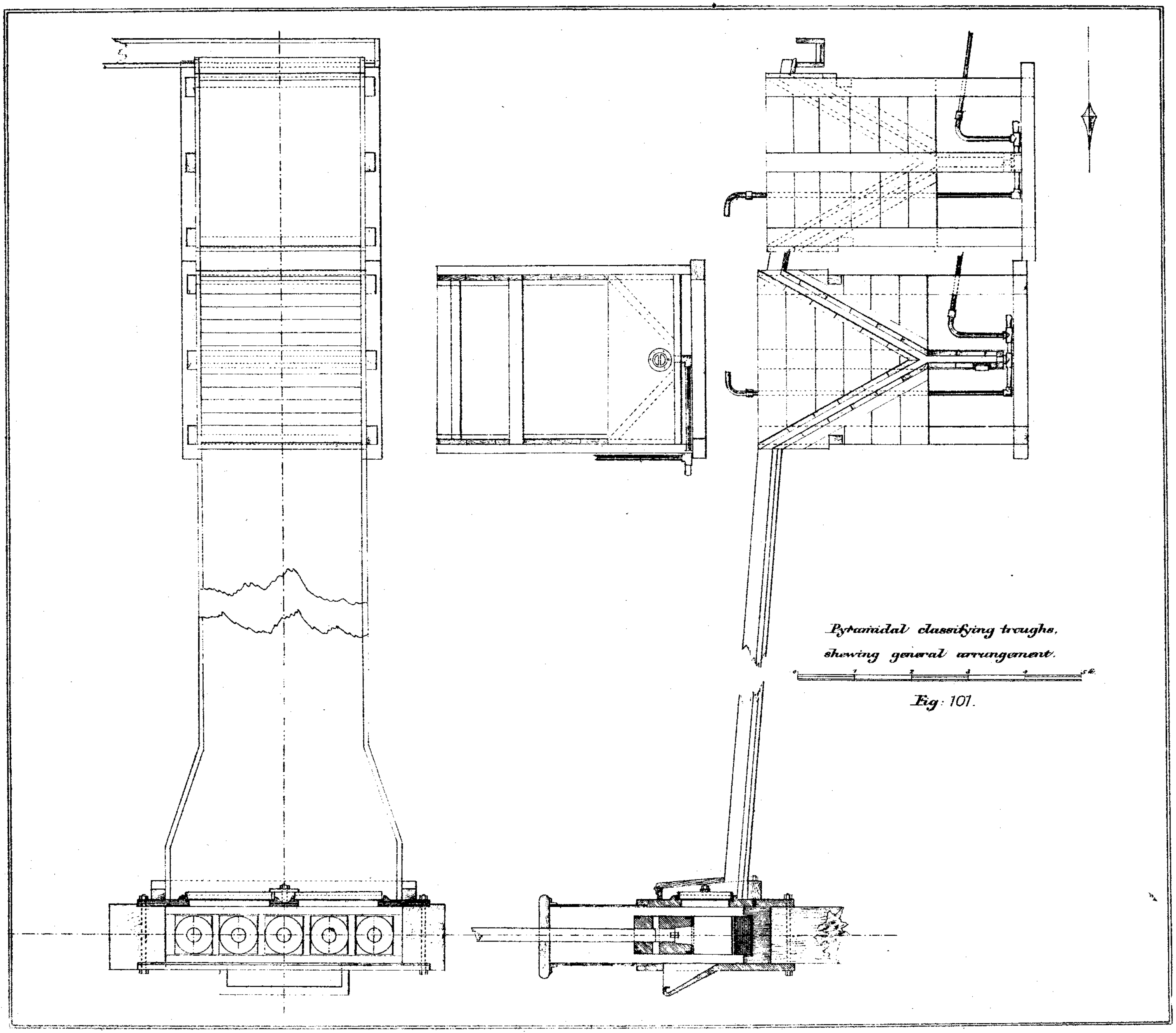
BROKEN HILL MINE AND WORKS.

This place has altered considerably since my former visit four years ago. Instead of being a comparatively barren desert having only patches of stunted trees and low scrub, a large township has sprung up, with some very fine buildings, and containing about eight thousand inhabitants. It is a township, however, that presents to a stranger a miserable aspect. The streets, as well as the whole of the surrounding country, were at the time of my last visit ankle-deep in fine red dust, which penetrates every pore. This, together with the scarcity of fresh water, makes it a very uninviting place for any one to reside in. The water at the time of my visit was brought a long distance by train and sold at the railway-station at the rate of 10s. for every hundred gallons for domestic purposes. This, together with the thermometer during the summer months standing at a high temperature, which brings out perspiration pretty freely when walking about, makes one feel gritty from head to foot, having his nostrils, ears, mouth, and throat full of the interminable dust, from which one cannot escape; and to have a good bath is entirely out of the question at the price paid for water.

The great change that has come over this place is entirely due to the rich discoveries of silver-ore at Broken Hill and the surrounding neighbourhood. The place where silver-ore was first discovered here was in the Umberumberka Mine, about eighteen miles west of Broken Hill and two miles west of Silverton, the latter place being the first township formed in the district. The lode in the Umberumberka Mine is from 2ft. to 10ft. in thickness, consisting principally of galena traversed by veins and lenticular bunches of brown iron-ore or gossan, carbonate of lead, and baryta. At the time of my former visit the depth this mine was worked to was about 130ft., or to water-level; but since then the workings have been extended to a depth of over 400ft., and the same character of ore is obtained. At about 200ft. below water-level large crystalline masses of antimonial ore, and in some instances specimens of galena coated with native silver, have been obtained.

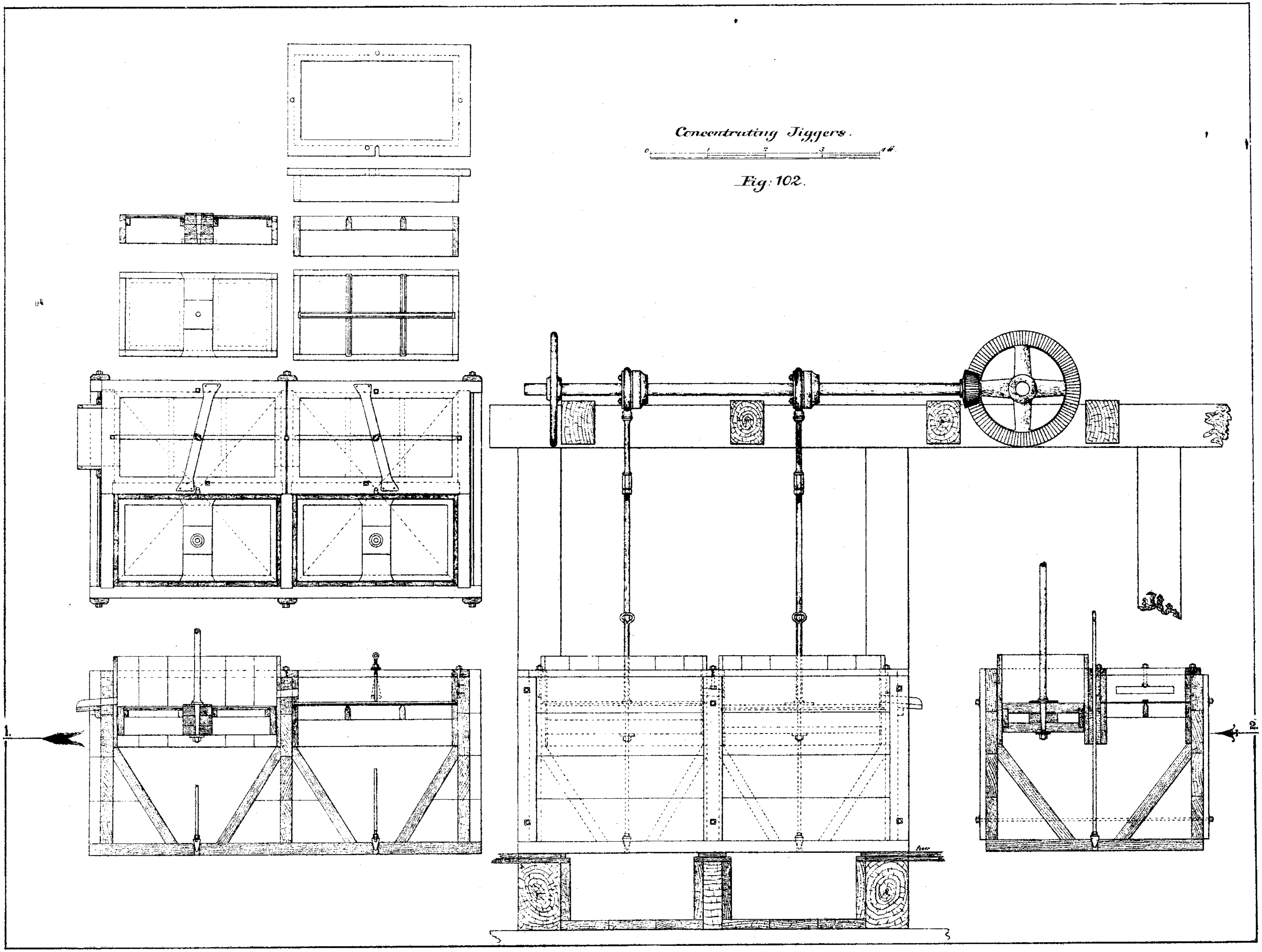
The principal workings, however, at present in the district are at Broken Hill, in the Broken Hill Proprietary Company's mines. These may be termed a quarry of silver-ore, for the lode varies from 10ft. to 160ft. in thickness. It was first discovered cropping out on the crest of a low ridge about 150ft. above the level of the plain in places here and there for over a mile in length, in huge craggy black masses changing in character in different places, but consisting generally of ferruginous quartzite, quartz, gneissen, felspar, gossan, and oxide of manganese, having veins and patches here and there of crystallized carbonate of lead and copper-ore — azurite, malachite, bornite, and chrysocolla. On sinking on the lode—which is a true fissure-vein — rich chloride-of-silver ore was found.

At the time of my visit the workings were confined to the surface and the 208ft. level. On passing down through the stopes one is perfectly dazzled with the quantity of minerals that this large ferruginous lode contains. Cavities are seen lined with beautiful rich blackish-brown stalactites of iron-ore, large masses of sparkling crystallized carbonate of lead, intermingled with carbonate of zinc and copper; while the pale yellowish-green patches of rich chloro-bromide silver-ore, mixed



Pyramidal classifying troughs, shewing general arrangement.

Fig: 101.



Concentrating Jiggers.

Fig: 102.

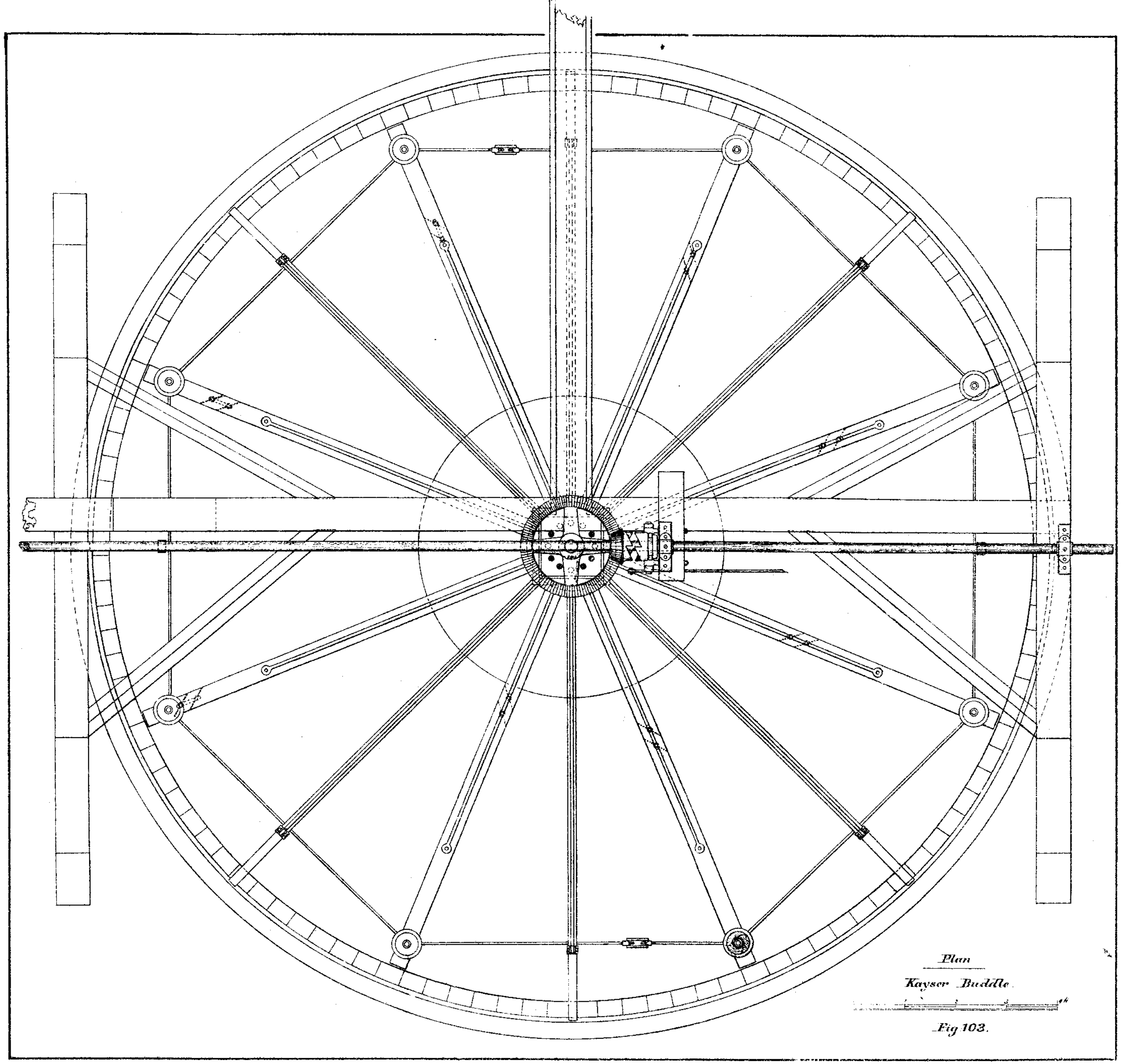
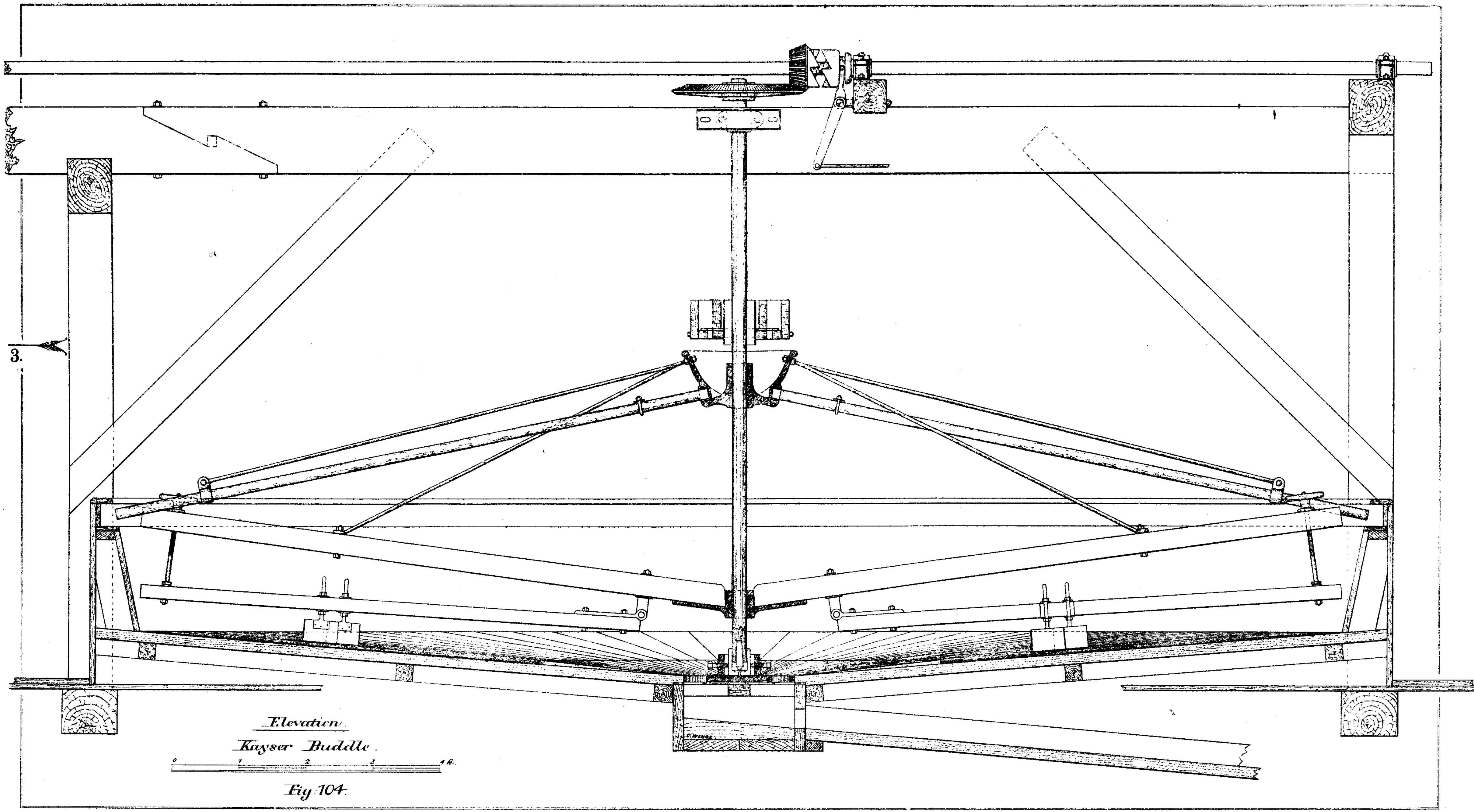


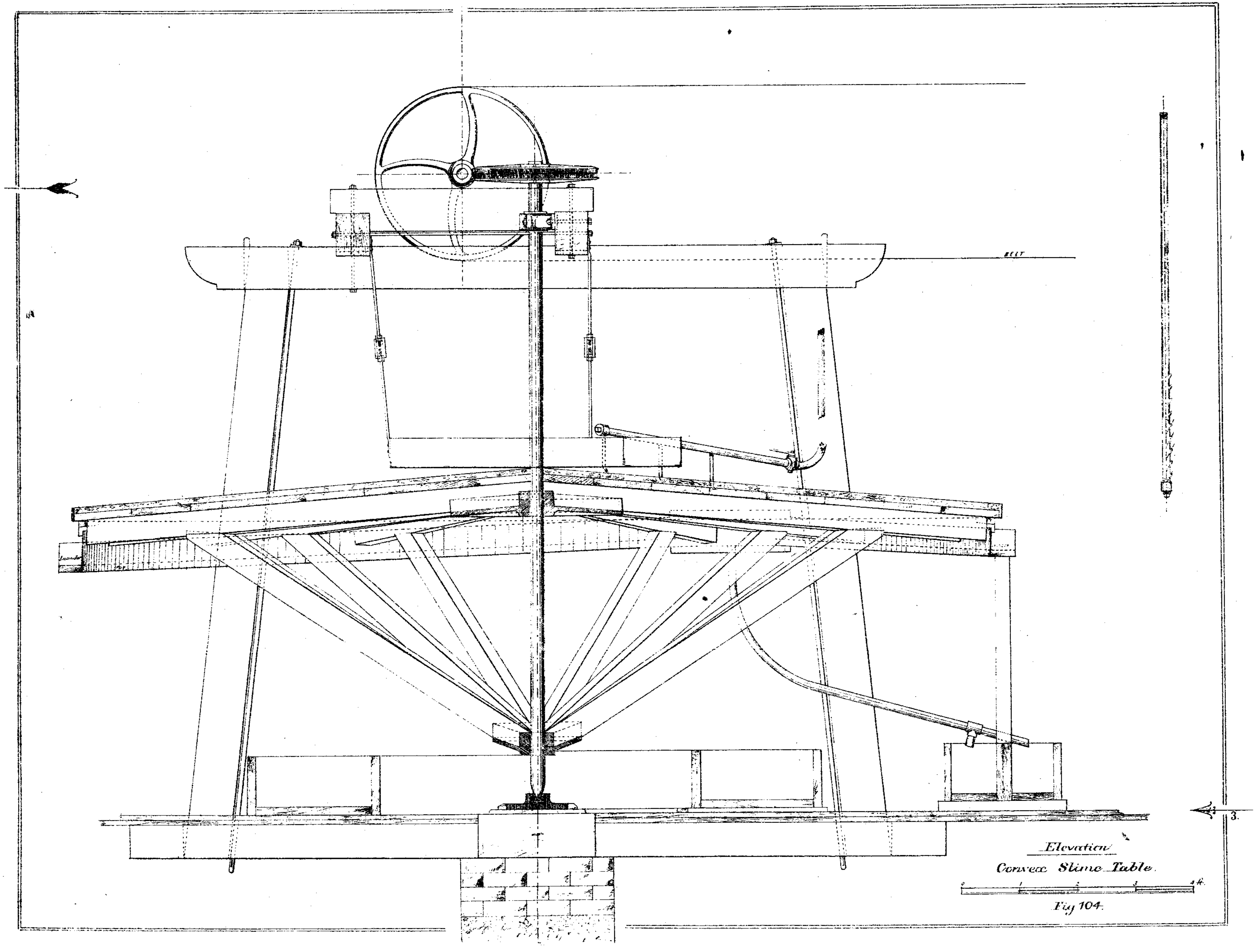
Fig 103.



Elevation.

Kayser Buddle.

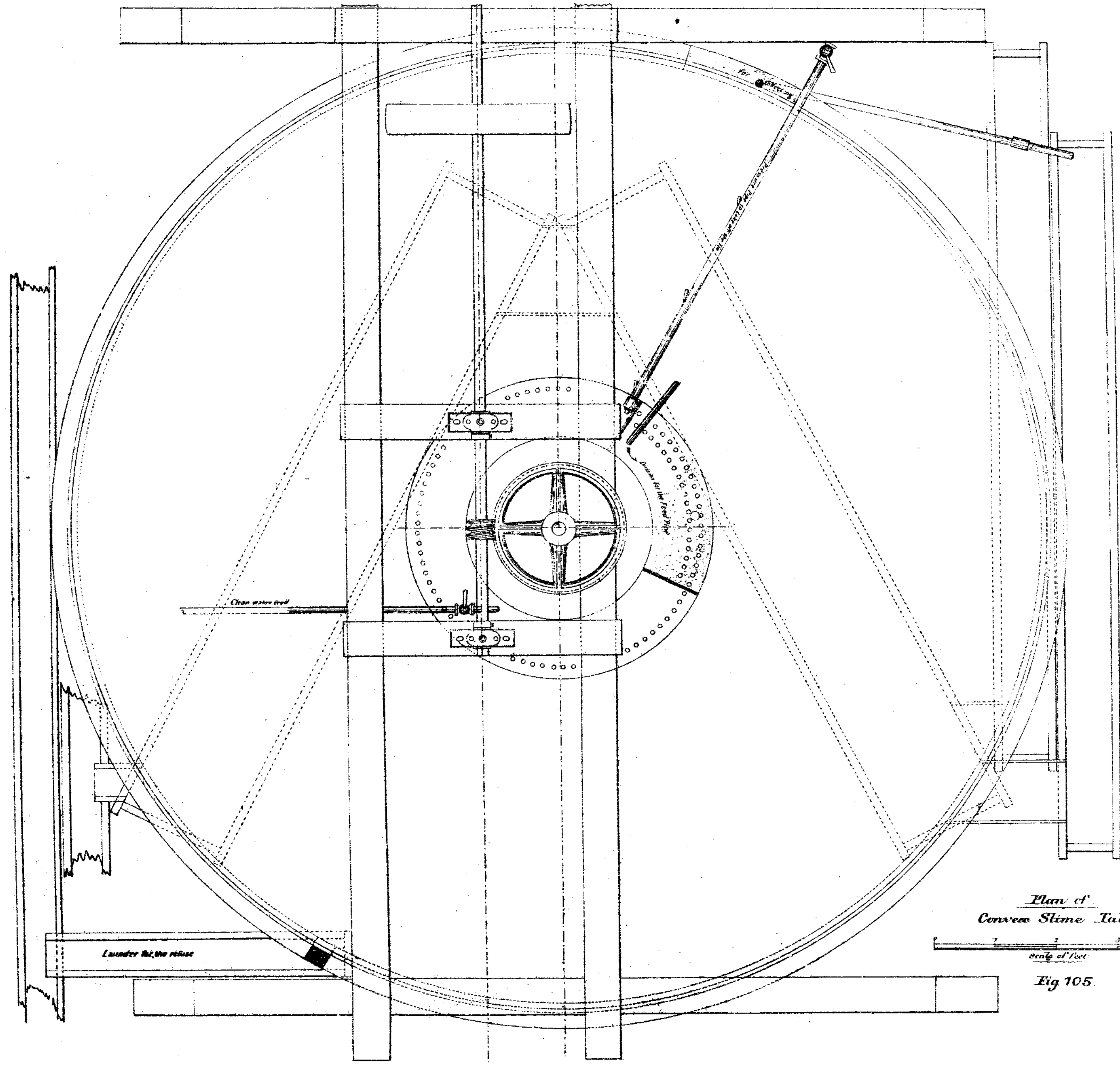
Fig. 104.



Elevation
Convex Slime Table.

0 1 2 3 4 ft.

Fig 104.



Plan of
Convex Slime Table

Fig 105

Conical settling vats.

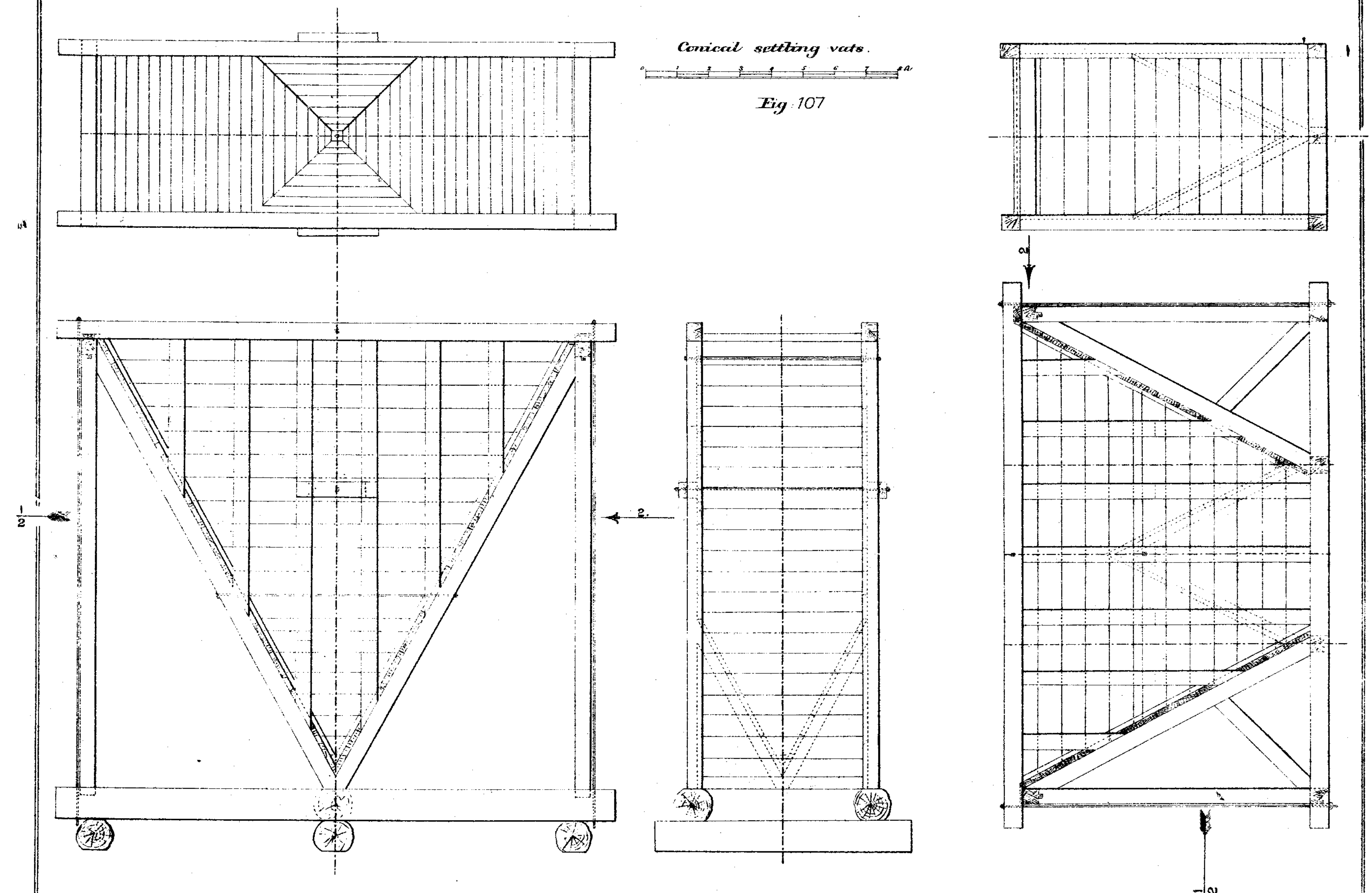


Fig. 107

among the whitish-yellow kaolin and siliceous manganic-iron ore and garnets, gives the galleries and stopes the appearance of a museum containing all manner of mineralogical specimens.

The largest percentage of copper-ore is on the north end of the Proprietary Company's mine, which comprises Blocks 10, 11, 12, and 13, the latter block being at the north end, adjoining Block 14. Here the ore changes greatly, and contains a large percentage of copper. The lode also gets narrow as it passes into the adjoining company's ground, but after passing through Block 14 it widens out again to about 30ft. in places in Blocks 15 and 16, which are held by the British Proprietary Company, and the lode also changes its character, being then principally galena. From this the lode has been again struck in the Junction shaft, about 40 chains further to the northward, and which is said to contain ore of a payable character for working. There are also shafts sunk along the line further to the northward, but the time at my disposal did not admit of visiting them.

The managers of several mines at Broken Hill showed me the greatest courtesy, and took me through the principal mines, which were extremely interesting, especially the workings of the Broken Hill Proprietary Mine. It is opened out on two levels, one at 150ft. and the other at 208ft. The older workings, which were carried on previous to the present manager, Mr. Patton, taking charge, look like a series of rabbit-warrens on a large scale—that is, the ore was taken out on the bord-and-stall principle, somewhat the same as working a coal-mine, which only allowed about one-half of the ore to be taken out of the lode. Mr. Patton has now established a new principle of timbering the lode, whereby the whole of the lode is taken out.

In taking out the lode square sets of sawn timber are used, all fitted into one another, the legs and cap-pieces being all of one length, and so fitted that they dovetail into one another. The legs are tenoned at each end, every tenon going half-way through the cap-piece, and the cap-pieces are so notched that when in their places they dovetail into one another, leaving a mortise at the joining of the ends of four caps to receive the tenon of the legs. These sets of timber are 7ft. high in the clear, and made of 10in. and 12in. square timber. When one is walking through these underground galleries it resembles going through a very strong warehouse, having upright pillars close together to support the floor above. In proceeding down from the upper to the lower level there is no descending of vertical ladders the same as in an ordinary timbered mine, but there are instead wide flat step-ladders, set on a flat inclination, so as to form, as it were, a staircase from one stope to another.

This system of timbering admits of any width of a lode being taken out, and also allows a large number of workmen to be employed in a short space of time. The main roadway goes along near the centre of the lode, and from this on each side the lode is worked. The manager informed me that it was a considerable time before he could get the workmen accustomed to put in this system of timbering, as it was looked on by old miners as an innovation on the established principle they had been brought up to. Indeed, so strong a prejudice exists among some of the old Cornish miners that they do not hesitate to state that it is not safe for men to work in the mine. When coming up from Adelaide to Broken Hill, one of the men who had been previously employed in this mine—an old Cornish miner—told me the manager did not understand the principle of timbering a mine, and that he would not work there on any consideration. This fully bears out what I have many times previously stated, that miners as a rule are the most conservative class of workmen that can be found. It is only by sheer necessity that they can be got to adopt any principle of either working a mine or treating ores beyond that which they have been brought up to. It can therefore be well understood, the trouble and anxiety that Mr. Patton has gone through in initiating this new principle of timbering, which is not only safer for workmen than the ordinary system of timbering, as the whole of it is strongly connected together, but it is the only system whereby the whole of a lode of this dimension can be taken out. The same system was adopted in working the Comstock Lode in Nevada.

It may be of some interest to know that the greater portion of the timber used in this mine is rimu—red-pine—sent from Southland, which the manager informed me cost, landed at Port Pirie, 8s. 6d. per 100ft., and it cost 11s. 6d. more per 100ft. to convey it to the mine by rail, thus costing at the mine £1 per 100ft. A set of timber is required for every 9 cubic yards of the lode that is taken out, and, taking the cost of timber at £1 per 100ft., and the cost of making the sets, the timbering of this mine costs about 10s. for every cubic yard of the lode that is taken out.

It must not be supposed that the whole of this lode is composed of rich ore—indeed in some places it is extremely poor, varying generally from 100oz. to 15oz. of silver per ton. As the ore is broken out it is placed in bins and assayed, and then mixed in certain proportions so as to produce something like a fair average of the whole. This is also another reason for this mixing: the rich lode-stuff requires a certain quantity of siliceous iron-ore as a flux to smelt it. There is sufficient flux of this description in the lode: the only flux that has to be brought on the ground is limestone, and this is obtained at a cheap rate.

There are seven shafts on the Proprietary mines, one of which is sunk down to a depth of 278ft. and another 379ft. From what could be learned respecting the ore below water-level, it contains a good deal of zinc-sulphide ore and galena. To use the words of the mine-manager, Mr. Harper, "The outcrop is manganic-iron ore, capping over carbonate-of-lead ore in Block 13, and kaolin-ore in Block 12; the same iron outcrop passing into Block 11, although not showing so prominently." In the latter block very little work has yet been done. Mr. Harper also went on to state that this manganic-iron ore in places was a good fluxing-material, and in other places the ore is more of a siliceous character, carrying silver-chlorides in payable amounts. The kaolin-ore in Block 12 is very rich in silver-chlorides, and also the carbonate-of-lead ore in Block 13 is rich in silver; but under water-level they change to lead- and zinc-sulphides, carrying silver in payable quantities for working. There is, however, no work yet done of any extent below water-level to test the value of the lode. It will be interesting to know the changes that take place in the lode below water-level, as the same changes may be met with in the lodes in New Zealand. If a large percentage of zinc-sulphides be found, it will make the ore very refractory to treat in extracting the silver.

The object in giving a detailed description of the ore in this mine is to show that it is entirely different from that found in New Zealand, where the ore contains 95 per cent. of silica, and consequently cannot be treated in the same manner as it is treated at Broken Hill, where the smelting-process has been carried on successfully. In order to adopt the smelting-process the ore must contain a large percentage of either galena or copper, combined with siliceous iron-ore to be used as a flux; but there are none of the argentiferous lodes yet discovered in New Zealand that contain so small a percentage of silica as the Broken Hill lode.

The reduction-plant at Broken Hill consists of three 80-ton and five 30-ton smelting-furnaces. The former is rectangular in shape, 5ft. by 8ft. 4in., with water-jacket similar to the La Monte furnace, and they have twelve tuyeers in each furnace. There is a longitudinal flue leading from these furnaces below the level of the charging-floor. This allows the top of the furnace to be open on a level with the charging-floor, and it is easily fed. The five 30-ton furnaces are of the La Monte pattern, and similar to those erected at the Thames and Karangahake. English coke is used as a fuel, which costs about £4 per ton. The superintendent of the smelting-works thinks it superior to the coke from Brunner, inasmuch that it breaks more in the form of cubes than the Greymouth coke, and is therefore better suited to mix among the ore in the furnace. He also used English fire-brick for lining the furnaces above the water-jacket, and when asked if he had tried the fire-bricks from the Brunner Mine he stated that he considered the fire-clay from the Brunner Mine superior to the English, but, to make bricks capable of standing in the furnace without cracking as the Brunner fire-bricks did, the clay should be burned or calcined previous to being ground up to make the bricks.

The material to be smelted is brought from the several bins along with flux, and mixed, in proportions according to the character of the ore, on the charging-floor before it goes into the furnace. Then it is thrown into the furnace along with coke, the object being to mix the coke among the ore as much as possible. The blowing-machines in connection with the furnaces are Baker's blowers, those for the 80-ton furnaces being No. 7½, which are driven by a pair of compound steam-engines, having cylinders of 16in. and 30in. in diameter respectively. The half-yearly statement for six months ending the 31st May, 1888, shows that 41,340 gross (equal to 39,789 net) tons of ore produced 6,773 tons of lead and 1,633,737oz. of silver, or an average of 17·3 per cent. of lead and 43oz. of silver per ton of ore: of this quantity 25,125 tons gross, equal to 23,943 tons net, were treated in the 80-ton furnaces, and 16,215 tons gross, equal to 15,846 tons net, were treated in the 30-ton furnaces. The large furnaces were completed in February, and were kept going continuously for a hundred days, with the exception of one of the furnaces, which was shut down for five days to effect some necessary repairs. The average quantity of ore smelted by these three furnaces was 239 tons net in twenty-four hours, or an average of 79⅔ tons for each furnace per day. The 30-ton furnaces during ninety-five days' run averaged 33 tons of ore net for each furnace per day.

It is found that the large furnaces are more economical to work, and that a larger percentage of refractory ore can be smelted in them than in the smaller furnaces, as will be seen from the following table:—

	Small Furnaces. Per cent.	Large Furnaces. Per cent.
Lead-ore ...	48	45
Iron-ore ...	16	12
Siliceous iron-ore ...	24	22
Kaolin-ore ...	12	21
Percentage of coke used ...	21	19
Percentage of limestone ...	20	18

This shows that they were able to increase the amount of kaolin-ore in the large furnaces by 60 per cent. over the amount they were able to reduce in the small furnaces, with a less percentage of coke and flux.

The following statement shows the comparative cost of smelting per ton of ore in the large and small furnaces:—

	Small Furnaces. £	s.	d.	Large Furnaces. £	s.	d.
Cost of labour directly about furnaces...	0	5	4	0	4	2
Cost of handling ore between mine-bins and charging-floor	0	1	4	0	1	4
Engineers, firemen, &c. ...	0	1	0	0	0	10
Sundries—blacksmiths, &c. ...	0	0	6	0	0	5
Superintendence, assay office ...	0	0	5	0	0	4
Total wages ...	0	8	7	0	7	1
Cost of coke per net ton of ore ...	1	2	0	0	19	10
Cost of limestone ...	0	4	10	0	4	3
Cost of firewood ...	0	1	0	0	1	0
Total cost of labour, fuel, flux, and repairs per net ton of ore	1	16	5	1	12	2

This shows that the cost of smelting was 4s. 3d. less with the large furnaces than with the small ones.

The water for supplying the water-jackets of the furnaces is pumped up from the flat when a shaft is sunk, and also from the mine. The quantity from the latter amounts to about 50,000 gallons per day. The whole of this water is brackish, and not fit for domestic use.

The company have hitherto shipped all the silver-lead bullion to England for refinement; but they are at present erecting refining-works at Port Pirie, where marketable lead and silver will be produced. Taking the amount of silver-lead bullion smelted during the six months ending the 31st May, 1888—namely, 6,824 tons—it means that they have turned out on an average, including stoppages for repairs, 262½ tons of silver-lead bullion per week; but the manager informed me that now that the large furnaces are in full operation they can produce about 400 tons of silver-lead bullion per week. The total amount of bullion produced from this mine to the 31st May, 1888, was 18,253 tons 14cwt.: of this amount, 18,112 tons 18cwt. was lead, and 4,788,836oz. silver. This amount of silver does not correspond with that marked on the Broken Hill trophy in the Melbourne Exhibition, which represents 5,508,836oz.; but the difference between the actual amount produced up to the 31st May and this amount was estimated up to the 1st August, the date of opening the Exhibition. The price of lead in the London market during the six months, according to the half-yearly statement referred to, varied from £15 10s. per ton in December, 1887, to £12 7s. in May, 1888, while the price of silver varied from 3s. 8⅝d. to 3s. 6d. per oz. The statement shows that both lead and silver are gradually declining in price. The total value of bullion produced for the six months referred to was £373,035, and the total cost of wages, management, and working-expenses for this period was £170,122; leaving a net profit on the working of the mine of £203,913, of which amount £49,964 has been spent in plant and machinery and other improvements in connection with the mine, and £152,000 has been paid in dividends; still leaving a balance to carry forward to the next half-year. The total spent on plant and machinery, including improvements, since the formation of the company has been £136,960; and the total amount of dividends and bonus paid is £952,000, which represents a cash payment of £59 10s. per £20 share, while the value of shares which the company have had allotted to them amounts to £1,744,000, making a total of £2,696,000.

It may also be of some interest to those engaged in silver-mining to state that the cost of getting and treating the ore is £4 6s. 2d. per ton. This includes total cost of raising, smelting, freights, realisation, management, and every charge thereon. Or, in other words, it requires ore of a furnace-return value of 10 per cent. of lead and 18oz. silver to pay the total working-costs of the ore per ton, taking the value of lead at £12 per ton, and of silver at 3s. 6d. per oz. in the London market; or, looking at it from another point of view, it will take 25oz. silver, furnace-return, at these market-values, to pay all costs of every nature other than permanent improvements.

The total number of men employed by this company is 796 in connection with the mine, 219 about the smelters, 172 engaged in general construction and repairs, 6 in the assay-office, 7 in the office and stores, and 31 general hands about the surface; making a grand total of 1,231 men.

It will be seen from the description of the ore in this mine that it is not particularly rich; indeed, there is a large percentage of the lode that will not average more than 20oz. of silver to the ton, and plenty of it considerably below this value. Therefore, to make low-grade pay for working, another system has to be resorted to—one which is especially applicable to the argentiferous ore found in New Zealand—namely, concentration. The object of concentrating the ore is to remove the excess of silica, and enable the residue to be treated economically by smelting. To effect this object the Broken Hill Proprietary are erecting a large concentrating-plant capable of treating 300 tons per day. This plant consists of a large number of Callom jigs for separating the coarse material, and convex buddles for treating the slimes.

The material from the mine is first put through a steam-stamp, which has an oblong shoe about 16in. by 8in., the die in the bottom of the mortar being 16in. in diameter. The upper end of the stamp-shank forms the piston-rod, and at every stroke of the stamp it revolves. The screen through which the crushed material is discharged has a ¼in. mesh, and its dimensions are about 4ft. by 3ft. The bottom of the screen stands about 3in. above the level of the die in the bottom of the mortar. The stuff goes from this through a German disintegrator, and through a number of Callom jigs, and the slimes pass on to convex buddles. By this means they calculate to reduce the ore 33 per cent., or, 300 tons of ore will be reduced to 100 tons of concentrates. The jiggers are similar in construction to those used for dressing the ore at the Mount Bischoff Mine in Tasmania, which have already been described, with the exception that, instead of being worked by an eccentric, they are worked by a tappet, which seems to me no improvement, as it certainly will increase the wear-and-tear considerably. The steam-stamp and jiggers in course of erection at Broken Hill were manufactured by Fraser and Chalmers, of Chicago, and the whole plant is a duplicate of the ore-dressing machinery used at Lake Superior and the Anaconda copper-mines, in America, where it is said to have proved exceedingly economical and efficient. This plant being only in course of erection, a perfect description of it cannot be given or anything said definitely as regards its efficiency; but one thing is certain, that proper concentrating machinery that will separate the silica from the heavy metalliferous portions of the ore will enable it to be smelted at a moderate cost.

ZINC DESILVERISATION PROCESS.

Dry Creek Smelting-works, Adelaide.

A great deal has been said from time to time as to the proper treatment of auriferous and argentiferous ores, and the Dry Creek Smelting-works have been represented as a place where parcels of ore from New Zealand may be sent with the view of ascertaining the accurate value of the ore. As far as the smelting-works are concerned, they are suitable to deal with a certain description of ore, but they certainly are not suitable to treat the general character of auriferous and argentiferous ores that are found in New Zealand. The ores treated there are principally those containing a large percentage of galena, and it is well known that this character of ore is suitable for treatment by the smelting process; but where the ores contain 95 per cent. of silica, being the same as the generality of the ores in New Zealand, those ores cannot be treated at the Dry Creek works or any other smelting-works with success. The quantity of flux required to smelt this class of ores is such that the cost of treatment would far exceed the value of the ore unless it were extremely rich, and even if rich ore would pay for treatment by this method, it is not the process that should be adopted. A large percentage of the silica must be got rid of by some means, such as concentration, before smelting can profitably be resorted to. However, these works are specially adapted for the treatment of galena ores containing silver or gold, or, if a sufficient quantity of copper were in the ore, it could be made a base to collect the silver and gold in lieu of galena; and they also treat auriferous ores after concentration.

The plant of the Dry Creek Smelting Company consists of rock-breakers, two 50-ton water-jacket smelting-furnaces, and lead-refining furnaces where the silver is extracted by the zinc disilverisation process, having kettles and refining-furnaces to separate the precious metals from the zinc. The whole process is conducted on the same principle as that used at the Germania Works, situated at Flack's Station, on the Utah Southern Railroad, six miles from Salt Lake City; the St. Louis Smelting and Refining Company, at Cheltenham, near St. Louis, Missouri; and the Pennsylvania Lead Company, at Mansfield Valley, near Pittsburgh, Pennsylvania.

The ore is brought to the works and there mixed with the necessary fluxes before charging the furnaces, on the same principle as at the Broken Hill Smelting-works, which does not require to be specially described again in referring to this part of the treatment of the ore at Dry Creek works. It is well, however, to observe that in treating auriferous and argentiferous ores by the smelting process it is most economical for the company and also better for the owner of such ores if they are sold by assay-value, allowing a certain percentage for waste and also a certain amount to cover the cost of treatment. The owners will receive in most instances a higher price for their ore, and the company can treat the same far more economically, inasmuch as different classes of ore act as fluxes to the others, instead of having to purchase flux necessary for the particular character of ore to be operated on. The Dry Creek Smelting Company purchase ores on this basis, and the same is done at the smelting-works at Freiberg, in Germany.

After the ore has been smelted the silver-lead bullion is taken to the refining-works, where the first process that it goes through is what is termed the softening process. The furnaces for this process are built at a high elevation, so that the lead can run into the kettle by gravitation, and thence run into the next furnace or kettle, as the case may be, until the silver and precious metals are entirely separated from the lead. The softened lead is then cast in bars fit for the market. In treating auriferous ores they are crushed and concentrated, and the concentrates roasted in matte in a reverberatory furnace previous to taking them to the smelting-furnace.

Softening Process.

The furnace where this process is carried on is in the shape of a reverberatory, having a wrought-iron riveted tank which forms the hearth and part of the sides. This tank is surrounded by a water-jacket, and is different in this respect from the furnace used at the Germania Works, but the general principle is the same. Therefore a general description of these works in America, where this principle is greatly in use, will give any one a fair idea of the process adopted.

The following extracts are taken from Professor Egleston's new work recently published on the metallurgy of silver and gold:—

"At the Germania Works, near the Salt Lake City, they treat silver-lead and also ores which they purchase in the open market. They have one shaft-furnace, and its capacity is 40 tons of argentiferous lead and 3 tons of ore per day. The value of the product of the works in copper and lead counted together, silver, and gold in 1874 was £281,250, about the proportion of 5, 6, and 2. The coke used comes from Connellsville, and costs £6 5s. per ton."

"At the works of the St. Louis Smelting and Refining Company, near St. Louis, there are two shaft-furnaces for the treatment of ore and the residues from the refining process. The lead which they treat contains about one-half of 1 per cent. of arsenic and antimony, so that it sometimes has to be polled. They treat from 500 to 600 tons of ore per month in addition to the bullion-refining, their daily capacity being 90 tons of lead and 15,000oz. of silver."

As the most of the works in the United States treat ore, the prices paid for ore in the best smelting-works in the West are given below.

The following is a schedule of prices given by the Colorado Smelting Company, of South Pueblo, in 1886:—

Lead-ores.

Containing	Silver, per Cent.	Gold, per Ounce.	Charge for Treatment, per Ton. £ s. d.
From 10 to 20 per cent.	95	£3 15s. to £3 19s.	1 8 6
" 21 to 30 "	95		1 4 6
" 31 to 40 "	95		1 0 10
" 41 to 50 "	95		0 16 8
" 51 to 60 "	95		0 12 6
" 61 upwards "	95		0 12 6

Dry Ores.

Containing	Silver, per Cent.	Gold, per Ounce.	Cost of Treatment, per Ton.
			£ s. d.
Up to 100oz.	92	£3 15s. to £3 19s.	2 14 0
101oz. to 200oz.	93		2 10 0
201oz. to 300oz.	94		2 8 0
301oz. to 400oz.	95		2 6 0
401oz. and upwards	95		2 4 0

All ores containing less than 10 per cent. of lead are termed dry ores.

The prices paid by the smelting and refinery works at Socorro, New Mexico, in August, 1885, were as follows:—

Silver-and-Gold Ore.

—	Per Cent.	Charge for Treatment, per Ton.
		£ s. d. £ s. d.
Silver below 100oz. (siliceous ore)	90	2 10 0 to 3 15 0
Silver 101oz. to 150oz. (ore containing iron)	91	2 1 8 to 2 10 0
Silver 151oz. to 200oz.	92	2 1 8 to 2 10 0
Silver 201oz. to 250oz.	93	2 1 8 to 2 10 0
Silver over 250oz.	94	2 1 8 to 2 10 0

After deducting one-tenth of the gold the balance is paid for at the rate of £3 15s. per ounce. If the ore contain over 5 per cent. of zinc an additional charge is made of 2s. 1d. for each 1 per cent. over this quantity in the cost of treatment.

The following will show how the ore is purchased: Say,—

Lot 1—	Lb.
Ore (bulk)	17,555
Moisture (1 per cent.)	175
Net weight	17,380

	£ s. d.	
Gold	3 15 0	per ounce.
Silver	0 4 0	per ounce.
Lead	0 1 5	per unit.

	£ s. d.
Assay—	
31 per cent. lead per ton, at 1s. 5d.	2 5 0
95oz. silver (90 per cent.), at 4s.	17 12 0
0·20oz. gold, at £3 15s.	0 15 0
	20 12 0
Less cost of treatment	2 10 0
	£18 2 0
17,380lb., equal to 8·98 tons of 2,000lb. each, at £18 2s. per ton	£162 10 9

In every case the amounts of silicon, iron, magnesia, lime, sulphur, &c., are considered, and affect the prices, so that no general statement of the prices paid for ores can be given.

The ore and argentiferous lead arriving at the works are sampled and assayed. When the ore is purchased at the mine it is sampled by the agent of the company and assayed at the works. When the assay of the agent's sample does not agree with that of the mine-owners they send a sample. The argentiferous lead is assayed on a sample being taken by boring into the top and bottom of both ends, and sometimes the middle, of the pig. When the lot sent is large, only one pig in a specified number is assayed; when the lot is small or the pigs very rich each pig is assayed. The pieces are melted in a previously-heated crucible, so as to have them rapidly melted. At a red heat the metal is stirred for five minutes, and then poured, with the dross, into a sample-mould. The sample ought to weigh 8lb., and represent 10 short tons of metal. Four samples, or a little over an assay-ton each, are taken from the middle of the four sides of the assay-bar, cutting it through. From these four, samples of an exact assay-ton each are taken and cupelled. If the gold is in small quantity four assay-tons are taken.

The process of desilverisation, as conducted in the works at Cheltenham, Salt Lake, and Mansfield Valley, consists of (1) softening the lead, (2) incorporation of zinc and separation of zinc scum, (3) refining the desilverised lead, (4) treatment of the zinc scum.

The object of desilverisation, as performed in these works, is to concentrate all the silver into a very small quantity of an alloy of zinc and lead, so rich that the lead resulting from its distillation will contain from 8 to 12 per cent. of silver, and to leave behind in the kettles lead which will not contain over 5 grammes of silver to 100 kilogrammes, or not more than 0·5 to 0·75 per cent. of zinc, and be pure enough to make white-lead, and hence command the highest market-price. The details of the arrangements of each of the works are different, but their general arrangement is about the same. The oldest of these works were arranged without reference to level, but the more recently-constructed ones are built in such a way that the softening-furnaces, where the metal is received to be melted and refined, are on the highest level. The desilverising-kettles are on the levels below, so that the discharge can be made directly from the softening-furnaces into them. The liquation-kettles are in a series below the desilverising ones, and the furnaces used for the liquation are on the same level. The refining-furnaces should be on such a level that the desilverised lead can be run from the kettles into them, and thence to cast by some simple arrangement into pigs. The cupel-furnaces are usually on the same level as the distillation-furnaces.

1. *Softening the Lead.*—As the argentiferous lead comes from all parts of the country, and contains a number of impurities in variable proportions, it must be refined or softened before it can be desilverised. The furnace used for this purpose is called the softening-furnace in most of the works. At the Germania Works it is called the A furnace. It is a large reverberatory, with a cast- or tank-iron basin, into which the hearth is built. The object of this iron basin is to have a furnace so cool that if the lead goes down into the hearth it will chill, or if the furnace is very hot it will be caught. The larger the furnace the better. Made of cast-iron, its size is limited; made of tank-iron, there does not appear to be any reason why it should not be of double the size, except the uncertainty of being able to purchase the supply of lead to work continuously. With an uncertain supply, it is better to multiply furnaces, as a small amount can be better and more economically treated in a small than in a large furnace. There is a point, however, beyond which it will not be profitable to increase the size, and this will be the quantity that can be held by the kettle. The limit in the kettles evidently will be that at which a man can no longer work the kettle conveniently.

The fireplace at Cheltenham is 2ft. 3in. wide and 5ft. 6in. long. The grate is 12in. below the bridge; the bridge is 2ft. 2in. below the roof, 1ft. 6in. above the hearth, and 2ft. 10in. wide. The hearth is made of a cast-iron basin, which is 15ft. 5in. long, 9ft. 6in. wide in the middle and 5ft. 3in. at each end, 2ft. 4in. deep, and 1¾in. thick. It weighs 8 tons, and is calculated to hold 25 tons of lead. At Cheltenham the tank forming the bottom of the furnace is cast in one piece. At the Germania Works it is cast in three pieces, and bolted together. This latter method is the cheapest; but if any of the bolts become loosened there will be a loss of lead, to avoid which the works at Cheltenham had the pan made in one casting. At the Pennsylvania Lead Works the pan is made of tank-iron, about ½in. thick, which is riveted. The bottom of the pan rests upon rails, which are supported on brick walls on either side of the furnace. It is now proposed to water-jacket all of these furnaces, which will both reduce the amount of repairs to be made to them, and shorten the time spent upon them. The doors of this furnace are counterpoised with pigs of lead, so that they can be very easily removed. They are bevelled, and fit into a slot, so that when they are closed and luted they are hermetically sealed. The hearth proper is built on the iron-pan bottom. It is made of fire-brick, laid in the form of an inverted arch, placed on a bed of coke next the pan, which is covered with a large brasque. If the pan is made of cast-iron the roof must not rest on it, as the pressure might increase its tendency to crack; if of wrought-iron, there is no such danger. The side-walls resting on it bear against projections on the rim on the side of the pan. These precautions are necessary in all iron-pan hearths, to prevent the rising of the hearth from the lead penetrating below it and breaking it up. Notwithstanding all the precautions taken against it, this accident, which causes great inconvenience and loss, happens so often that at the Germania Works holes are now bored in the angles in the bottom and sides of the pan, so that the lead cannot collect. The flowing lead warns the men before serious accident has happened that it is time to make repairs. These furnaces should all be placed at the highest point of the works, so that the lead and other products may descend by gravity from one furnace to the other. The hearth is made on an inverted arch, by stamping brasque on it and cutting it out to the shape the hearth is to have. On this fire-bricks, placed on end, are put so that the fire-bricks have exactly the same shape as that of the brasque below. The side walls are all built with either a course of red bricks or a layer of brasque behind them. The hearth inclines towards the tap-hole, which may be either at the flue end or on the side of the furnace. The tap-hole is made of cast-iron. If the pan is of cast-iron it is screwed on to it, if of wrought-iron it is bolted on. It is lined with brasque or arranged as described for the Mansfield refining-furnace. The lead is charged through the charging-door in the side of the furnace, on a spaddle with a long handle. A roller is placed in the door to relieve the workman from the friction. The lead is thus distributed about the furnace so as to melt readily. The charge is always melted at a low heat.

The usual charge at the Germania and Cheltenham works is from 22 tons to 24 tons, depending on the purity of the lead. In the works of the Pennsylvania Lead Company, at Mansfield Valley, they sometimes charge as much as 25 tons to 26 tons, the charge depending on the quantity of crasses that the lead makes. It is always made at Cheltenham so as to produce about 20 tons at the end of the operation, or a quantity sufficient to completely fill one kettle. When the furnace is hot the whole charge melts in about two hours. It remains in the furnace from six to eighteen or even twenty-four hours, depending on the work in the kettles, which must be kept full. During this time it is kept at a low heat, and air is allowed to have free access to the surface of the metal.

The operation of softening consists in keeping the lead melted at a very low temperature, the object of which is to separate the copper by liquation, as it is much less fusible than lead. The scums containing the copper are drawn with a tool made of birch-wood, so as not to contaminate the lead, as would be the case if an iron tool were used. It is always necessary to endeavour to remove all the copper, whether gold is present or not. The gases in the furnace are oxidizing, and crasses, containing the oxides of the foreign metals, rise to the surface. At the end of three hours the temperature is raised to a dull-red heat. At this temperature the volatile metals rise to the surface, are then oxidized and mixed with the lead-oxide, and are removed with them. The bath is kept for twelve to fifteen hours, if necessary, at the same temperature, and frequently rabbled to bring the impurities to the surface. If the lead contains from 3 to 4 per cent. of impurities, the crasses are only drawn as they form, but if more impure a steam-jet blast is discharged directly into the bath, which, by constantly bringing fresh portions of the lead to the surface, promotes the oxidation, and the crasses are removed several times; but if the lead is moderately pure the crasses are drawn but once, which will generally be at the end of six to seven hours. An air-blast, as it acts mostly on the surface, does not answer so well as the steam, which has, however, the disadvantage that the constant wave-action of lead-oxide against the surface of the brick tends to wear it away. It should be protected with a water back where steam is used. The first crasses will amount to from 1·5 to 2·5 per cent. of the charge, and are taken off at the end of from five to seven hours. Before drawing them they are mixed with coal on top of the melted charge, to reduce any oxide of lead, and are then drawn; and if they form again they are removed. When they no longer form, the furnace is cooled gradually, but is kept about the melting-point of lead. The crasses are drawn from the working-door, and are collected in a bin, where they are allowed to accumulate until there is enough to work.

When litharge commences to form, the crasses are no longer drawn, but are left in the furnace after the lead has been tapped. In refining the next charge they give up their oxygen to more easily-oxidized metals, and thus help to separate them from the lead. Quicklime is usually added as soon as they commence to form, to keep the litharge from cutting. The time required for softening the charge varies from eighteen to thirty-six hours, depending on the amount of impurities contained in the lead; usually it is less than twenty-four hours.

Sometimes all the impurities have been removed at the end of twelve hours or less; but the charge in the furnace must stand until the desilverising-kettles are ready. This is done by simply shutting the damper, and only adding enough fuel to the fireplace to keep the charge melted; but, as all the compounds of arsenic and antimony are very fusible, the softening must be kept up as long as these form. With a charge of 26 tons at the Pennsylvania Lead Works from 24½ tons to 25½ tons of softened lead remain in the furnace.

It often happens that the charge is ready for tapping, but the desilverising-pots are in use; so that the lead is kept in the furnace at the melting-point until the pots are free. It is cheaper, even if the lead is extremely pure, to keep it melted in the furnace during the time necessary, rather than to cast and remelt it.

At Cheltenham, the top hole opens into a deep but narrow trough lined with brasque, from which the lead is siphoned off with a Steitz siphon. The brasque is made of four-fifths clay and one-fifth coke-dust; it is made as dry as can be stamped, and then carefully shaped and cut down to make the arch leading into the furnace. When the kettles are ready the furnace is tapped. The tapping-spout is very large, and during the time of casting exposes a large surface to oxidation, thus increasing the losses in lead. If the furnace was sufficiently high above the pot, the lead could be tapped by a gutter directly into the kettle. The contract is always made to have the kettles cast bottom down.

At the Germania Works the tapping is very inconveniently done through an iron pipe 40ft. long and 5in. in diameter, with holes cut into it at intervals to facilitate the removal of dross, which might clog the pipes. It is necessary to heat the whole length of this to prevent the lead from chilling. This is done with coals suspended under it in pieces of sheet-iron; but there must be a shield between the fire and the pipe to keep the latter from cracking.

As the softening-furnace is always above the kettle, a much simpler plan is to run the lead into the kettles by gravity, using an iron trough for the purpose. At the Pennsylvania Lead Works this is accomplished with a trough made of angle-iron, so as to form a gutter. The other end is placed over the pot into which the metal is to be discharged. When the furnace is ready to be tapped, a charcoal fire supported on a sheet-iron frame, a bracket for which is fastened to the side of the furnace, is made around the stopcock attached to the side of the iron tank of the furnace-bottom, until the heat is raised above the melting-point of lead. The handle is then turned, and the contents of the furnace discharged into the kettles. When the furnace is empty the angle-iron is taken away, and the space left free until the next tapping. At these works there are three of these softening-furnaces, each one having three desilverising kettles. At the Germania Works there are two, with five kettles each; at Cheltenham, one, with three kettles.

The crasses from the softening-furnaces are first liquated to remove any excess of lead they may contain. At the Germania Works this was formerly done in a reverberatory furnace, of peculiar construction. The hearth was 3ft. deep; 18in. above it a set of grate-bars was placed: the skimmings were placed on these, and the crasses remained there, while the lead flowed through. This lead is very hard. It is either sold, or refined with the other lead. The first crasses drawn contain the most of the copper. They are always kept separate from the other. They are put through the blast-furnace at the end of a campaign, with galena or pyrites, in order to concentrate the copper in a 40-per-cent. matte, which is sold. Some hard lead is produced, which is treated with the lead of the other crasses. Sometimes the crasses, without liquation, are melted in the blast-furnace, producing very hard lead. This is treated with soft lead, poor in silver, in the softening-furnace, and made into marketable hard lead, richer in silver, however, than the ordinary softened lead.

At the Germania Works a copper matte is produced which contains 20 per cent. of copper, 20oz. to 25oz. of silver, and a slag containing 10oz. of silver. The matte is concentrated to 40 per cent. of copper, and is sold.

The assays of three of these concentrated samples contained—

	No. 1.	No. 2.	No. 3.
Silver	113·54oz. ...	88oz. ...	94·56oz.
Gold	1·18oz. ...	1·02oz. ...	1·02oz.

From the dust-chambers connected with this furnace only a small amount of material is collected, and this very near the furnace. It contains only from 3oz. to 4oz. of silver. The other crasses are treated in a reverberatory furnace. The material being at first only partially reduced, the first lead which flows carries most of the silver, and is put to one side. The charge is then completely reduced. The product is a very hard lead, which is allowed to accumulate until there is enough to make a charge in the softening-furnace.

If the ores contain a very large amount of antimony there will be two or three sets of crasses after those containing copper have been removed, which will be mostly very impure litharge. The lead produced from them is a compound of arsenic and antimony, which is not refined, but is sold as hard metal. The loss in lead in softening is about 2½ per cent.

2. *Incorporation of Zinc and Separation of Zinc Scums.*—To effect a desilverisation there are at Cheltenham three kettles set in a triangle, at Mansfield Valley a series of three kettles set in a row, and at the Germania Works a series of five set as shown in Fig. 108; the first two holding 20 tons each, the next two 7 tons, and the last 4 tons. These kettles are set in masonry, with a fireplace underneath them. The furnace is tapped into the two upper ones alternately. The upper kettles at Mansfield hold 23 tons. The upper kettles at Cheltenham weigh 4,700lb. each, and cost between £80 and £100 each. They are 6ft. 6in. in diameter, and 3ft. deep. The depth of these kettles is the same in all the works; the diameter varies with the capacity. They were first made to hold 10 tons, then 20 tons, then 30 tons, and now some of the works make them to hold as high as 40 tons, as it was soon found that it cost but little more to treat 40 tons than it did to treat 25 tons in twenty-four hours. The limit will evidently be fixed, first by the supply of silver-lead, secondly by the number and capacity of the softening-furnaces. The time these kettles last varies greatly. They are usually bought by weight, and not on the specification of what they will do. In some of the European works they are bought by contract to last during the melting of a specified number of tons. The contractors much prefer to make the contract in this way, but it is not usual in the United States.

At the Germania Works the discharge-spout is cast on the bottom of the kettle, and is constantly breaking. At Mansfield the middle one has a spout at the bottom, which communicates with the third and smallest. At Cheltenham the kettles are discharged by the Steitz siphon, which is a much simpler method, and much less expensive than having pipes cast on the bottom, as they always increase both the cost of the kettles and the liability of loss of lead.

The kettles are filled with melted lead from the softening-furnaces. When they are full they are heated up to the melting-point of zinc, which takes about one hour. It is important that the heat should be high enough to melt the zinc readily. The kettle is so large that there is but little danger of overheating. When the temperature is at the right point the zinc is added. At the Germania Works and at Mansfield the zinc is thrown in, or laid on the top of the lead, and incorporated as it melts. At Cheltenham it is placed in an iron cage, which is let down to the bottom of the pot. The amount of zinc to be added will generally be about 1lb. for every 5½oz. of silver. This will usually amount to between 250lb. and 550lb. to each kettle. In general, with ores varying from 100oz. to 300oz. of silver, 1·4 to 3 per cent. of zinc is added. It is not all added at once, but sometimes in two and sometimes in three additions, the proportions being determined by assay in each case. These additions should be so regulated as to make the richest possible alloy at first, in order to shorten the process as much as practicable, to diminish the liability to oxidation when it is liquated, and to produce a lead containing not more than 0·1oz. to 0·2oz. of silver.

At Mansfield the lead contains from 50oz. to 400oz. of silver. To this, from $1\frac{1}{10}$oz. to 2 per cent. of zinc is added in four additions. The zinc is thrown in on the top of the melted lead, and then is stirred into it by a tool 5in. by 10in., with a long handle. After the first addition it is stirred for half an hour. The scum is then allowed to rise and cool until there is a ring of 3¼in. around the outside. It is then skimmed with a perforated skimmer until the lead is bright. The other additions are made in the same way.

At the Germania Works, for a charge containing 60oz. of silver and ¼oz. of gold, 1·85 per cent. of zinc was added. For a charge containing 140oz. of silver and 3·8oz. of gold, 2·3 per cent. of zinc was used. Of this, 0·5 per cent. was added in the first addition, 0·4 per cent. in the second, and 0·1 per cent. in the third. For a charge containing 350oz. 2·6 per cent. of zinc was used.

The following table, prepared by Mr. A. V. Weisse, of the Germania Works, gives the amount of zinc used in two charges:—

Example.	Total Weight of Softened Lead.	Silver contained, in Grammes to the 1,000 Kilos.	Gold contained, in Grammes to the 1,000 Kilos.	Zinc used.
	Lb.			Per Cent.
No. 1	402·442	4,300	125	2·3
No. 2	402·224	4,256·7	127·45	2·6

To be sure of lead at 5 grammes from lead containing 1,000oz. to 1,400oz. of silver at least 1½ per cent. of zinc must be added. Pure zinc is no longer used for all these additions. The second, third, and fourth scums of a previous operation, which are not very rich in silver, are used for the first and sometimes for the second addition, thus greatly reducing the amount of zinc required for the operation. When the lead is very poor in silver the first addition is used several times, in order to make it as rich as possible. The object of dividing the additions is to arrive as quickly as may be at the highest percentage of silver, and to get an alloy so rich that there will be little liability to loss in the subsequent liquation; thus shortening and cheapening the process. The amount to be added in the first charge will depend on the amount of copper in the lead. If it contains but a small amount of copper and some gold, 100lb. is added at Cheltenham. If there is much copper more zinc must be added to bring out the copper, as most of the copper comes off with the first crasses. If gold is present in large proportion the quantity of zinc must be increased, since all the gold comes off with the first scums. In making the assay to ascertain whether the gold has been removed, small quantities should not be taken, for with them no traces of gold will be found. It has been ascertained by experience that it is best always to take from 8 to 10 assay-tons, in order to get a weighable amount. It is a question whether, as the gold has a stronger affinity for zinc than it has for silver, it would not be better to make a doré bullion alone by taking out the gold and silver by two or three additions of zinc, rather than make doré bullion and fine silver by repeating the additions of zinc. If no gold is present, two-thirds of the charge of zinc necessary for the whole operation may be added in the first charge. It is then stirred from one-half to three-quarters of an hour with a flat spatula, which is 17in. in diameter, attached to a piece of gas-tubing 6ft. long. The temperature during this time is kept above the melting-point of zinc. The tool is made to work from the sides toward the centre with a downward motion at the same time. In order that the workman shall not have to bear the whole weight of the stirrer, it is suspended on a hook attached to a chain hung in the ceiling. The stirring is very difficult work, and a great many tools have been devised to do it mechanically, with no great success. Dry steam from a $\frac{3}{4}$in. pipe placed 12in. to 15in. below the top of the melted lead in the centre of the kettle does this stirring very thoroughly. It does not require much attention, and by its use for from half to three-quarters of an hour much labour is saved. The lead is also made much poorer in silver than can be done by hand-work. Any metal that adheres to the sides of the kettle must be detached with a chisel-shaped tool and thrown to the centre of the kettle. When the zinc is thoroughly incorporated the fire is drawn and the kettle allowed to cool until the zinc-alloy, which contains the silver, rises and floats on the top of the melted lead. The time taken depends on the heat of the metal and on the season of the year: in summer it is four hours; in winter only two.

It has been proposed to introduce water running through pipes bent to the shape of the kettle, for the purpose of hastening the cooling of the bath. These are movable, so as to prevent the metal from cooling on them. The movement of the pipes reduces the temperature about equally throughout the bath. Usually the upper part cools faster than the lower. By this method it was expected to materially reduce the quantity of zinc required, as well as the time of the operation; but it has not worked very satisfactorily. Generally, however, the cooling is done by either drawing or banking the fire, or by throwing water on the surface of the metal and taking off the consolidated crust as quickly as it forms.

The skimmings are taken off in perforated ladles suspended on chains, and put into one of the smaller kettles. These first skimmings are carefully separated from the rest if the lead contains either much gold or much copper, or both. At Cheltenham the skimmings from the first addition of zinc are charged into a small kettle between the two large ones. At the Germania Works kettles Nos. 1 and 2 are skimmed into Nos. 3 and 4. If the skimmings come from the first addition of zinc they are partially liquated in Nos. 3 and 4 and transferred to No. 5, where the liquation is completed. All the lead in Nos. 3 and 4 is then put back into Nos. 1 and 2, ready to receive the second addition of zinc. The skimmings from the second, third, and fourth additions of zinc are not liquated, but are used over again. The amount of labour required is one man to each kettle. The kettle is left until it is full, and is then fired up and partially liquated, which takes about an hour. The kettle must not be heated too hot in this liquation, for there would be danger of oxidizing the zinc, in which case the silver would go back to the lead. The lead separated in liquation is put back into the large kettle, No. 1, before the second addition of zinc.

At Mansfield all the skimmings except the first, which contains copper and may contain gold, are ladled into the middle kettle, which is kept heated, and are liquated at once, the lead flowing into No. 3. The lead which collects there is put back into No. 1 with the next charge of lead. At Cheltenham the zinc skimmings are taken from kettle No. 1 and liquated in No. 2. While the second addition of zinc is being made the liquated lead is removed to No. 3. The charge in No. 3 is then put back into No. 1 after the second addition of zinc. The lead remaining in the kettle after the first skimming should not contain more than 20oz. The zinc for the second and third skimmings is not liquated, but used in the next operations. The skimming is made into the adjacent kettle. After making an assay of the melted lead to ascertain what is required, the next addition of zinc is made, and the skimming continued about the same time. After the second skimming there should not be more than 10oz. to 15oz. of silver remaining. An addition is made if the assay shows it to be necessary. The last two charges are placed partly on top of the melted lead and partly in the cage. It is then stirred for three-quarters of an hour, and left to cool down. The skimmings are liquated as before. The lead contains from 1oz. to $1\frac{1}{2}$oz. of silver. A new addition of zinc of about 100lb. is made, after which the lead will not contain more than $\frac{2}{10}$oz. to $\frac{3}{10}$oz. of silver.

At Cheltenham there is not more than $\frac{1}{6}$oz. of silver remaining when the lead is tapped into the refining-furnace. Frequently the last skimmings are too poor in silver to admit of treating. They are put to one side, and form either a part or the whole of the first additions of zinc in the next kettle.

It has been suggested that the assays taken from the kettle to determine the value of the metal are often taken either too soon or when the metal is too hot. In the latter case it is said that the silver rises to the top and gives a higher value to the metal than the average of the kettle.

At Mansfield poor lead is not tapped if it contains more than $\frac{1}{10}$oz. of silver to the ton, and the merchant pig assays 0·075oz to 0·15oz.

When the Germania Works were first built the Flack process was used. The liquated zinc skimmings were charged in a blast-furnace with a very basic slag and small pressure of blast. The result was rich lead and a rich slag. In the condensation-chambers a very impure oxide of zinc was collected, which was but a small part of that actually charged in the furnace. As the use of this process occasioned a loss of from £3,750 to £5,208 a year in zinc it was abandoned, and the Faber du Faur furnace was introduced in its place.

It is always best to use good zinc for the separation. An attempt was made at the Chicago Silver Smelting and Refining Works to economize in this direction by using scrap-zinc; but it was found that the lead after its use sometimes contained as high as 18oz. to the ton, and the attempt had to be abandoned.

The following statement of several charges at the Germania Works is made by the superintendent, Mr. A. V. Weisse:—

	No. 1.	No. 2.
Number of pounds charged in the softening-furnace...	41,614	40,120
Number of grammes of silver	5,700	1,980
Number of grammes of gold	110	10
First addition of zinc, from second and third additions of a previous operation, in pounds	4,000	3,000
Grammes of silver in lead after first addition ...	1,360	1,600
Second addition of zinc, in pounds	600	600
Grammes of silver in lead after second addition ...	20	30
Third addition of zinc, in pounds	80	125
Grammes of silver in lead after third addition ...	Trace	6

The following table was prepared by Mr. E. F. Eurich, of the Pennsylvania Lead Company:—

Desilverisation.	No. 1. Lb. oz.	No. 2. Lb. oz.
Quantity of work-lead charged in the kettle ...	87,294	...
Taken off—schlicker (cuprous oxide)	3,497	...
Pure work-lead	83,797	62,895
Silver contained	6,305 6	6,165 9
Quantity of zinc added	1,760	1,260
Weight of skimmings after liquation	9,525	6,362
"Abstrich," from dezincation of poor lead ...	7,810	3,500
Oxides and metallic lead from the market-kettle ...	1,000	700
Lead from liquation of zinc crust	808	...
Market-lead	67,104	53,420

At Cheltenham the liquated skimmings, still soft, are thrown on iron gratings from 1in. to 1½in. apart, and pushed through, in order to reduce it to pieces of small size, which can be more conveniently introduced into the retort. In most of the works it is thrown upon an iron plate in front of the kettle, and, in order to break it up, is rapidly moved about with a rake, and, if necessary, cut up with a shovel, so that the pieces are about the size of a hickory-nut.

That it is very necessary to refine the lead before adding the zinc is shown by the following experiments made by Mr. C. Kirchhof: At the Delaware Lead Works 20 tons of lead containing 95·5 per cent. of lead, and containing antimony, arsenic, zinc, bismuth, and copper, were slowly melted in a kettle. During the melting it was carefully crassed to remove as much copper as possible. At the same time a charge of refined lead was treated. The zinc was added as usual. The table below gives the result:—

—	1. Not Refined.			2. Refined.		
	Silver per Ton, in Ounces.	Charge of Zinc.		Silver per Ton, in Ounces.	Charge of Zinc.	
	Oz.	No.	Lb.	Oz.	No.	Lb.
Before adding zinc	85·60	...	...	94·90	...	...
After 1st charge	85·50	1	250	85·60	1	150
" 2nd "	85·30	2	250	47·60	2	150
" 3rd "	83·80	3	150	16·10	3	150
" 4th "	83·50	4	100	1·70	4	150
" 5th "	83·00	5	100	0·18	5	100
" 6th "	48·20	6	100	...	...	...
" 7th "	8·20	7	100	...	...	...
" 8th "	0·80	8	70	...	...	...
" 9th "	0·15	9	30	...	...	...
Total	...	...	1,150	...	...	700

The two operations were carried out under exactly similar circumstances. The amount of zinc used with the unrefined lead was 2·87 per cent., with nearly double the amount of time and labour required for the second. The zinc used for the refined lead was 1·75 per cent. With the unrefined lead the amount of marketable lead produced was 43 per cent.; with the refined, 72 per cent. The unrefined produced a very large amount of impure-zinc scum, which increased the time, cost, and losses in distilling, cupelling, and working the products.

Refining the Desilverised Lead.—The lead in kettle No. 1 contains usually ¾ per cent. of zinc, no matter what the heat is, or how much zinc is added. At the Germania Works it contains from 0·7 to 1 per cent. of zinc and antimony together. It must be refined to separate the zinc and get it ready for the market. This operation is one of refining, but in the West it is known under the name of "calcination." This is done in a furnace with a cast- or tank-iron bottom like the softening-furnace, holding from 20 to 40 tons. At Mansfield Valley the bottom is made of tank-iron, with a pipe having a stopcock at furnace similar to that used at the Germania Works. The one used at Cheltenham is essentially the same; it is a little larger, but the dimensions vary only a few inches. The fireplace is 2ft. 3in. wide and 4ft. 5in. long; the bridge is 8in. below the roof on the fireplace- and 11in. on the hearth-side. It is 2ft. 10in. wide, 3ft. 6in. long, and 2ft. above the hearth. The hearth is 13ft. 4in. long and 7ft. 3in. wide in the middle, and 3ft. 6in. wide both at the firebridge and the flue. It is here made of one casting. At the Germania Works it is cast in three pieces, as shown in the section A–B, Fig. 109. The arch is 2ft. 9in. above the floor of the laboratory; it has three openings 4in. square in the firebridge and two on its side, for the introduction of air. The charge remains in this furnace from eighteen to twenty-four hours. The surface is constantly exposed to the air entering the furnace by the air-holes at the bridge. Towards the end of the first half of the time that the charge is to remain in the furnace the bath is skimmed. By this time all of the zinc will have been either volatilised and carried off in the gas, or have been oxidized and be contained in the scums which have been withdrawn. After this the antimony is oxidized, some of the lead oxidizing at the same time. The progress of the operation is ascertained by making small test-bars from time to time, on which the quality of the lead is determined. As soon as the antimony is all gone the lead is fit for corroding, and is cast to be sent to the white-lead works. The skimmings amount to from 1 ton to 1½ tons. They contain from 45 per cent. to 50 per cent. of lead, and most of the zinc and others remaining impurities. The charge is rabbled after the oxides have been removed, but any others which form are allowed to remain until the furnace is tapped into the polling-kettle, which is usually about twenty hours after the charge is made.

At Mansfield Valley the refining is done in twelve hours. The lead is not polled, but is cast into pigs directly from the furnace. At Cheltenham and elsewhere, where the lead is cast before all the antimony is out, it is polled. The polling-kettle is placed at the flue end of the furnace. The lead flows into a deep cast-iron channel lined with brasque, from which it is siphoned off. The top of the kettle is about 6ft. from the floor. Directly in front of the kettle, and about 2ft. below the floor-level, there is a sunken track upon which a car is run, the top of which comes up to the level of the floor. The car is about 6ft. wide, and receives the pigs and carries them to the storehouse. There is a space of 4ft. between the car and the furnace. The polling is done in eight hours. The wood is held at the bottom of the kettle by a crutch (Fig. 110). The same apparatus is used at the Germania Works, except that, instead of the crutch, the bars are straight and pointed; the holes are bored in the wood to receive them. Short sticks of green wood are used; but, to insure a plentiful escape of steam, all the wood for this purpose is soaking in a pool of water. Three or, exceptionally, four pollings are made, the number depending on the quality of lead. Each polling lasts about an hour, so that the furnace is ready to receive a new charge as soon as the ore refined in the softening-furnace is desilverised. There is a great advantage, both in the reverberatory furnace and in the polling-kettle, in using dry steam, both to oxidize the zinc and to bring fresh surfaces in contact with the air. It is for this reason that the very wet wood is used in the polling-kettles, as it is the steam which does the work, producing a very high quality of lead. The advantage of polling with wood in the reverberatory furnace has long been recognised, but it is only lately that steam, which is much more manageable, has been used. When steam is used in polling it is introduced from an iron pipe 1½in. in diameter, which is let down nearly to the bottom of the kettle. The steam has a pressure a little above what is necessary to overcome the plumbostatic pressure. The current of steam is kept up for from ten to sixty minutes, the time depending on the purity of the lead. Generally, however, in the United States, the lead contains so large an amount of antimony that the softening-furnaces must be used. The polling-kettle is only used as an auxiliary, so as not to delay the preceding operations in case the antimony is not all out when the next charge is ready to go into the softening-furnace. It is much better to poll with steam than to keep the lead longer in the softening-furnace, and thus keep back the rest of the process. The weight of the dross collected from the kettle at the Germania Works which was polled four times is given below:—

	Lb.
First polling	1,301
Second polling	881
Third polling	671
Fourth polling	290
Total	3,143

The crasses from all the pollings, usually amounting to from 1,000lb. to 2,100lb., are melted at the Germania Works in a reverberatory furnace, and make common soft lead. The crasses from the softening-furnace, however, make silver-lead, which is treated by zinc. Those from refining, which at the Germania Works is called calcination, make soft lead of ordinary quality.

The following table gives the quantity of skimmings for examples Nos. 1 and 2:—

From refining-furnace	31,700lb.
From polling-kettle	20,352lb.
Quantity of work-lead taken from the polling-kettle	76·25 per cent.
Silver contained in the market-lead per 1,000 kilogrammes ...	6 grammes.

The polling-kettles at Cheltenham are emptied by the Steitz siphon. To do this it is first heated in the melted lead; it is then turned over so that the funnel end is uppermost. The stopcock is then opened, and melted lead poured into the funnel, which is inserted into a pipe for the purpose. When full, the stopcock is closed, and the siphon full of lead is then turned over into the kettle and placed in position. The joints of this siphon are made of gas-pipe couplings. At first it was supposed to be necessary to make them perfectly air-tight, but afterwards it was found that if six or eight threads of the screw were run into the coupling the joint was lead-tight and perfectly flexible. The end of the siphon where it turns down to discharge the lead is a simple gas-pipe coupling, to which a handle is attached for convenience of moving. While the lead is not being cast the vertical arm is simply turned up. When the car with the pig-moulds is ready the siphon is turned down, being held by the handle, and is moved from one pig-mould to the other in succession as they are filled with lead. The joint is long enough to allow of filling all the moulds without moving the car.

At the Pennsylvania Lead Works the same arrangements for discharging the furnaces are made as have been described in the softening-furnaces, except that the angle-iron at the end away from the furnace is supported above a swinging trough suspended to the ceiling, one end of which is circular in shape. The tap-hole of the furnace is opened; the lead flows in the angle-iron launder to the foot of the swinging trough, the spout of which can be directed at will towards any ingot-mould which is to be filled. When the casting is complete the angle-iron is taken away, and the swinging trough removed. As this involves the use of a large number of ingot-moulds in other parts of the same works, the casting is done in pig-moulds attached to bogies, of which there are four. The first is filled and rolled away to cool; the second and third are filled and rolled away in the same way. When the third is withdrawn, and by the time the fourth is ready to fill, the first is cool enough to be tipped, and is brought to its place to be filled again by the time that the fourth commences to fill. This method requires but few moulds, but needs quick and dexterous men.

4. *Treatment of Zinc Scums.*—The zinc crusts from the liquation are reduced to small pieces and distilled. If it has been carefully liquated it will not contain more than 4 or 6 per cent. of lead. The distillation is done in graphite retorts in fixed furnaces, as was formerly the case at Bloomfield and Cheltenham, or in Faber du Faur's tilting-furnace.

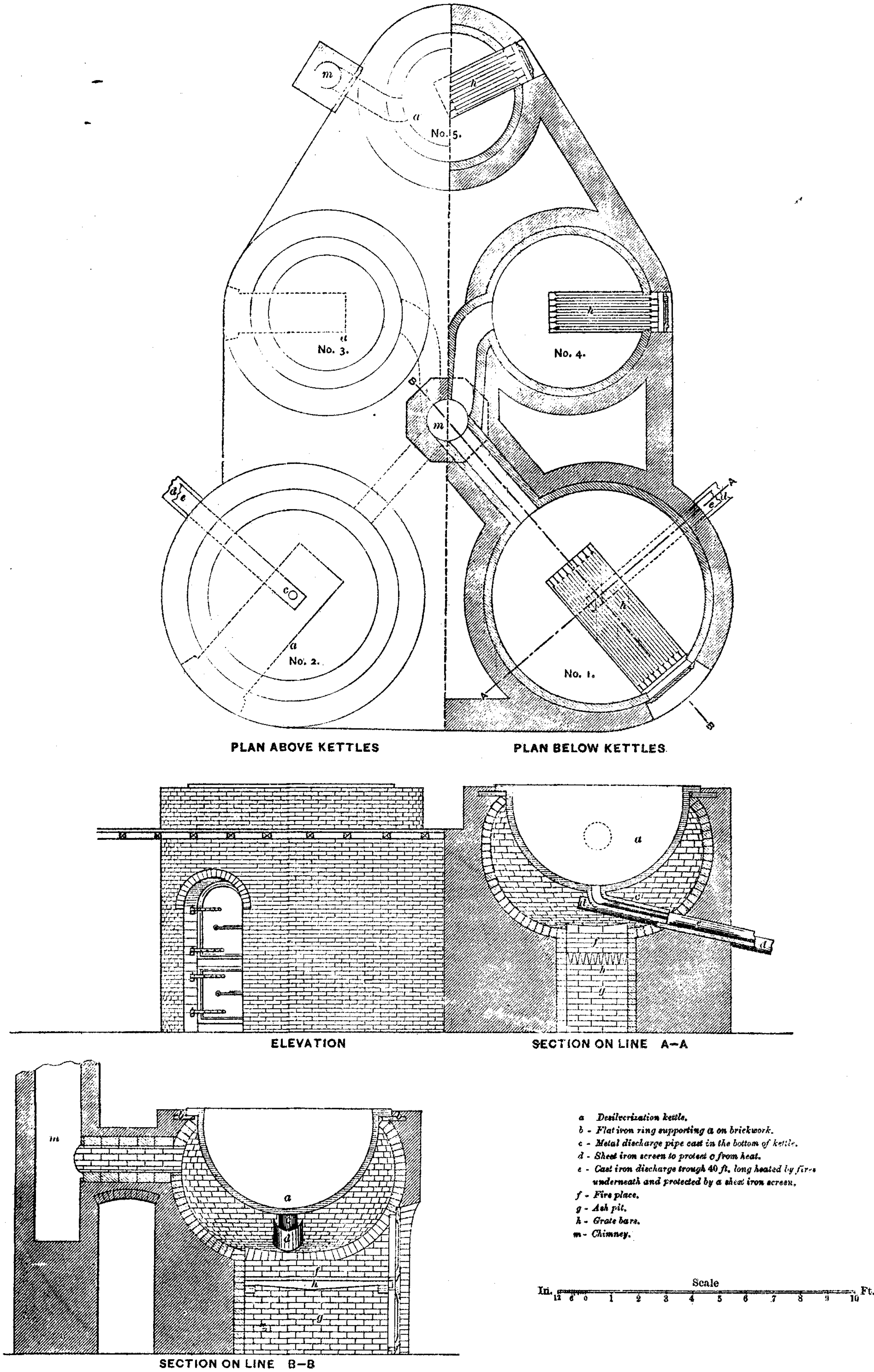
At the Germania Works the Flack process is sometimes used for commercial reasons, but it was never considered good metallurgy. It consists in charging scums in a shaft-furnace, with the drosses from refining and ores of all kinds. The result of this treatment is a rich silver-lead, but the greater part of the zinc is lost. From a metallurgical standpoint this treatment is very objectionable, and should not be imitated; but the commercial conditions in Utah are so peculiar that it has proved financially successful, owing probably to the great skill with which the process is managed; for a bad process well conducted may sometimes be made successful. In almost every other establishment in the country the zinc scums are retorted. The retorts used at Bloomfield (New Jersey), Philadelphia, Cheltenham, and the Germania Works are shown in Figs. 111, 112, and 113. They formerly varied but little in different works. They were made of as small a capacity as 200lb., but this was found to be too small. They have been recently made as high as 700lb., but this was rather large. The usual capacity is between 400lb. and 500lb. Generally they are ¾in. thick on the sides, and nearly twice as thick on the bottom. The neck is 7in. long, and the body of the retort is 2ft. The diameter of the extremity of the neck is 5½in., but where it joins the body it is 8in. The body in its widest part is 14in., but there is only 9in. at the end. These retorts are made of New Jersey clay and chamotte, with 25 per cent of graphite. They were formerly one of the largest items of cost in the conduct of the operation.

One of the first furnaces used for distillation of the zinc was invented by Mr. W. M. Brodie, and has been constructed in several works. It consists of a large chamber, in which six retorts are placed in two levels, as shown in Fig. 111. These are heated by a fireplace 2ft. 10in. long and 15in. wide, with cast-iron grate-bars, which is blown by a forced blast which enters the ashpit at C, having first been heated in two hot-air pipes, which are placed in compartments above and behind the furnace. The retorts are protected from the direct action of the fire by the arches *d*. The heat escapes by the flues above the retort-chamber, passes into the chamber above, down at the back, and out of the furnace by an underground flue. The retorts are the ordinary graphite retorts, holding from 450lb. to 500lb., so that the furnace would hold from 2,600lb. to 3,000lb. of alloy at a time. Each retort has a condenser, *b*, attached to it, and in front of it a charging-table, *f*, covered with cast-iron. It is necessary to remove the condenser, as in the other furnaces, to clean the retort. The furnace is tapped on the back side at E, from holes, ¼in. in diameter, bored through the bottom of the retort, into moulds placed on the iron ledge *g*.

If the material charged is pure the time required for an operation is twelve hours. If it is not it may require as much as twenty-four hours, depending on the quality of the material charged. One man does the work of the six retorts. The amount of fuel required is 1 ton of coal for 1 ton of alloy. The results do not differ materially from those of the other furnaces, except that the operation is longer. They were constructed in the now abandoned works at Bloomfield, New Jersey, and the works of Messrs. Tatham, in Philadelphia.

The following table of the results of the working of this furnace has been prepared by Mr. C. Kirchoff, jun., who had charge of these furnaces while they were working.

Fig 108



DESILVERIZATION KETTLES AT THE GERMANIA WORKS, FLACH'S STATION, NEAR SALT LAKE, UTAH.

Fig. 109.

LEAD REFINING FURNACE,

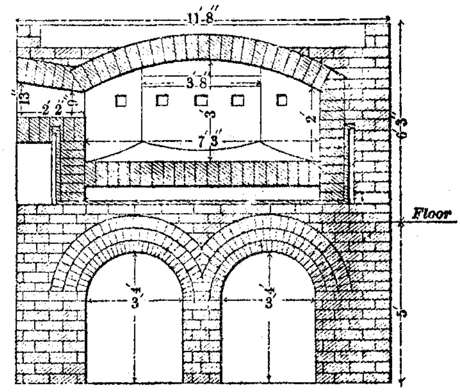
AND MARKET KETTLE.

Holding from 18 to 19 tons.

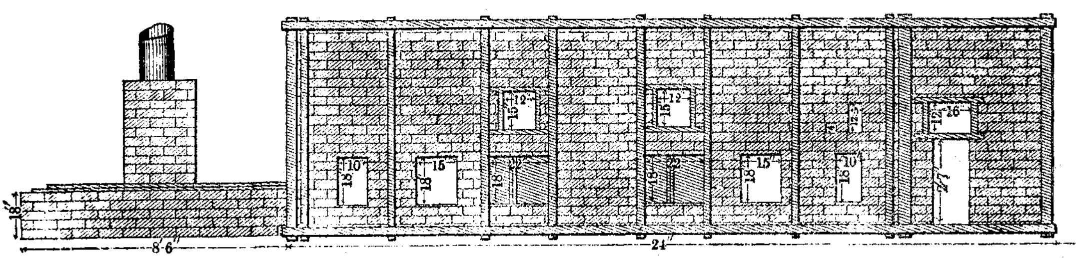
AT GERMANIA WORKS, UTAH.

SCALE

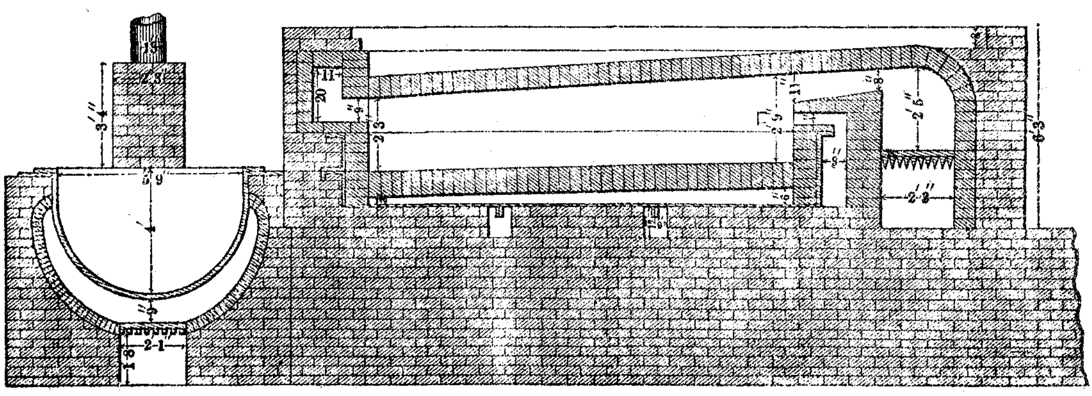
In. 12 6 0 1 2 3 4 5 6 7 8 9 10 Ft.



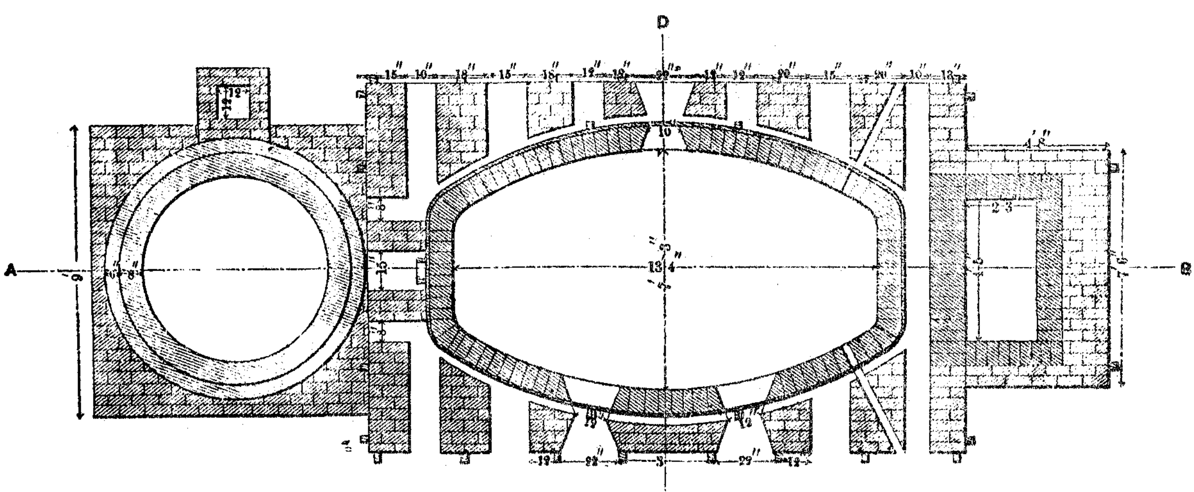
SECTION THROUGH C.--D.



ELEVATION

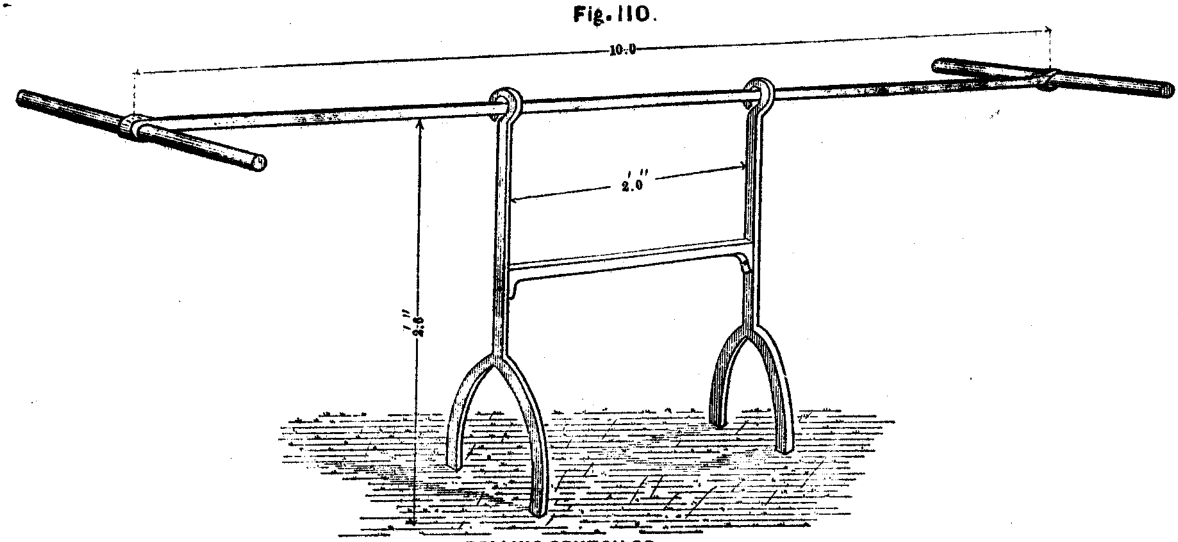


SECTION THROUGH A.--B.

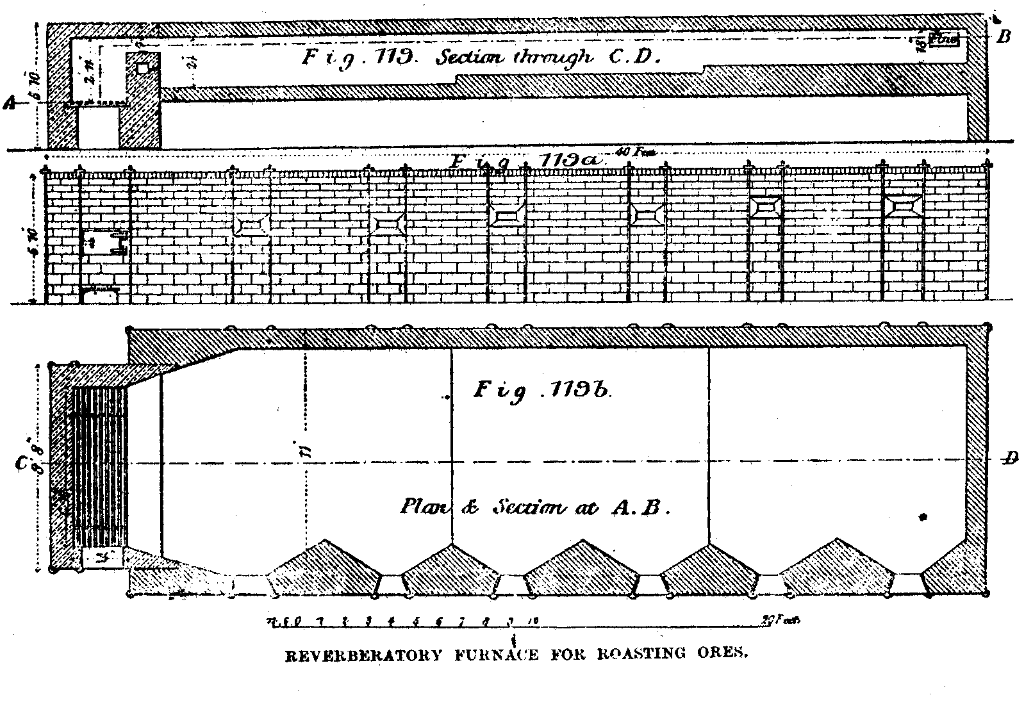


PLAN.

Fig. 110.



POLLING CRUTCH OR
WOOD-HOLDER FOR THE POLLING POTS



REVERBERATORY FURNACE FOR ROASTING ORES.

Fig. 111.
BRODIE'S DISTILLATION FURNACE.

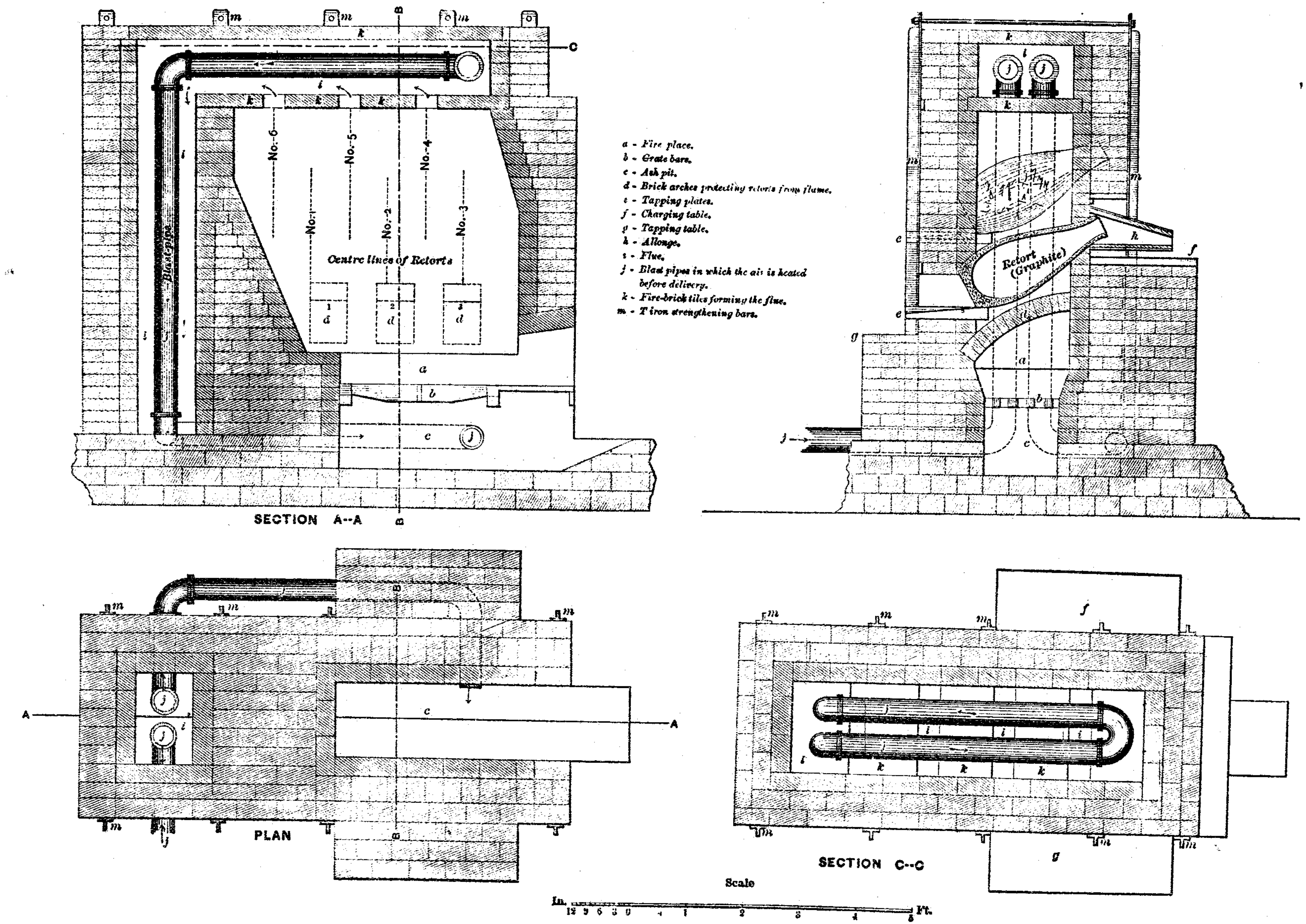


Fig. 112.

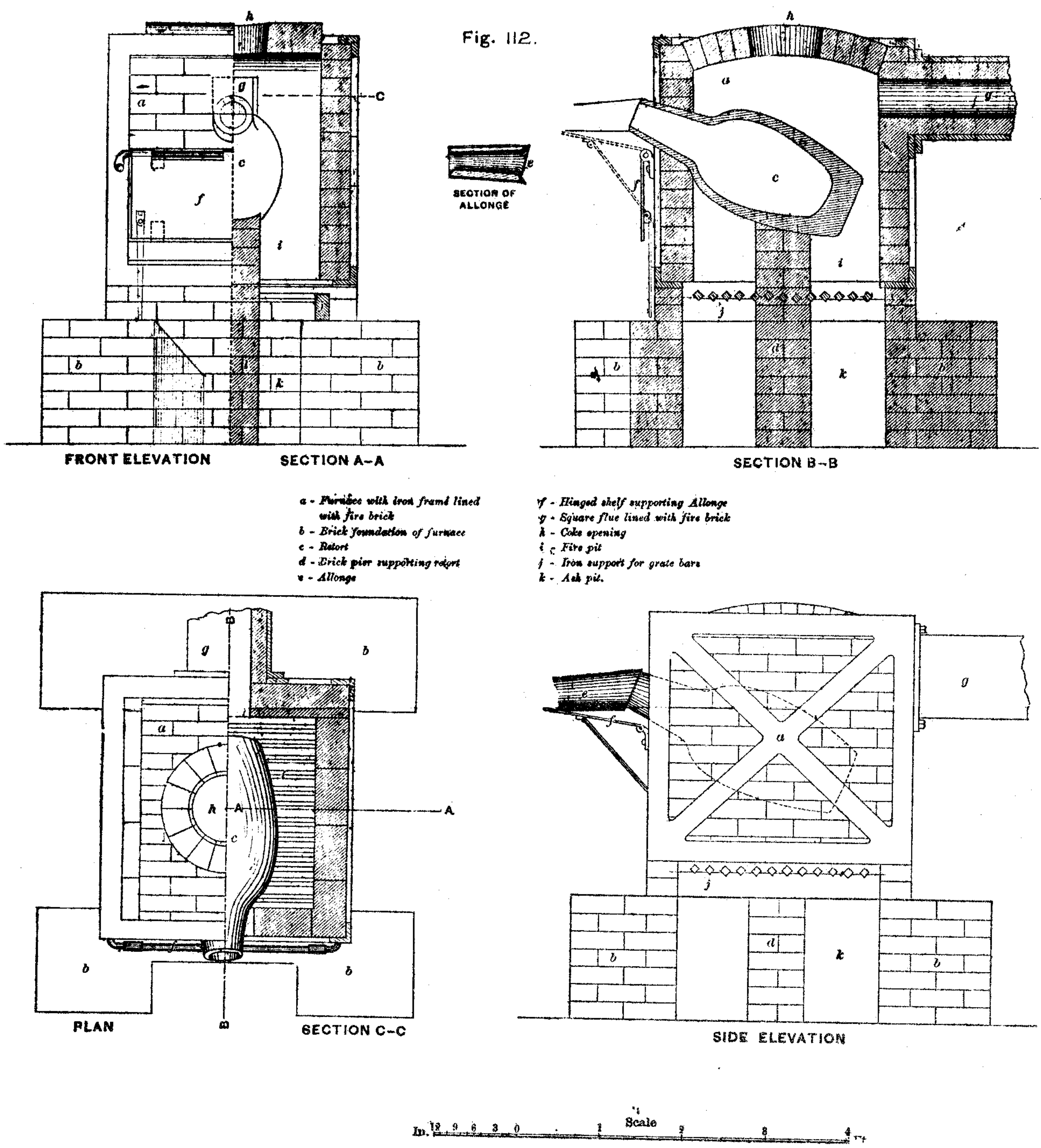
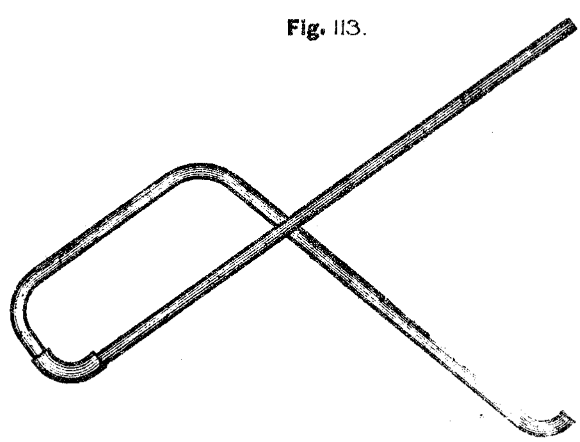


Fig. 113.



STEITZ'S SIPHON TAP FOR THE DISTILLATION FURNACE

Scale In. 12 9 6 3 0 1 2 F

Fig. 114.

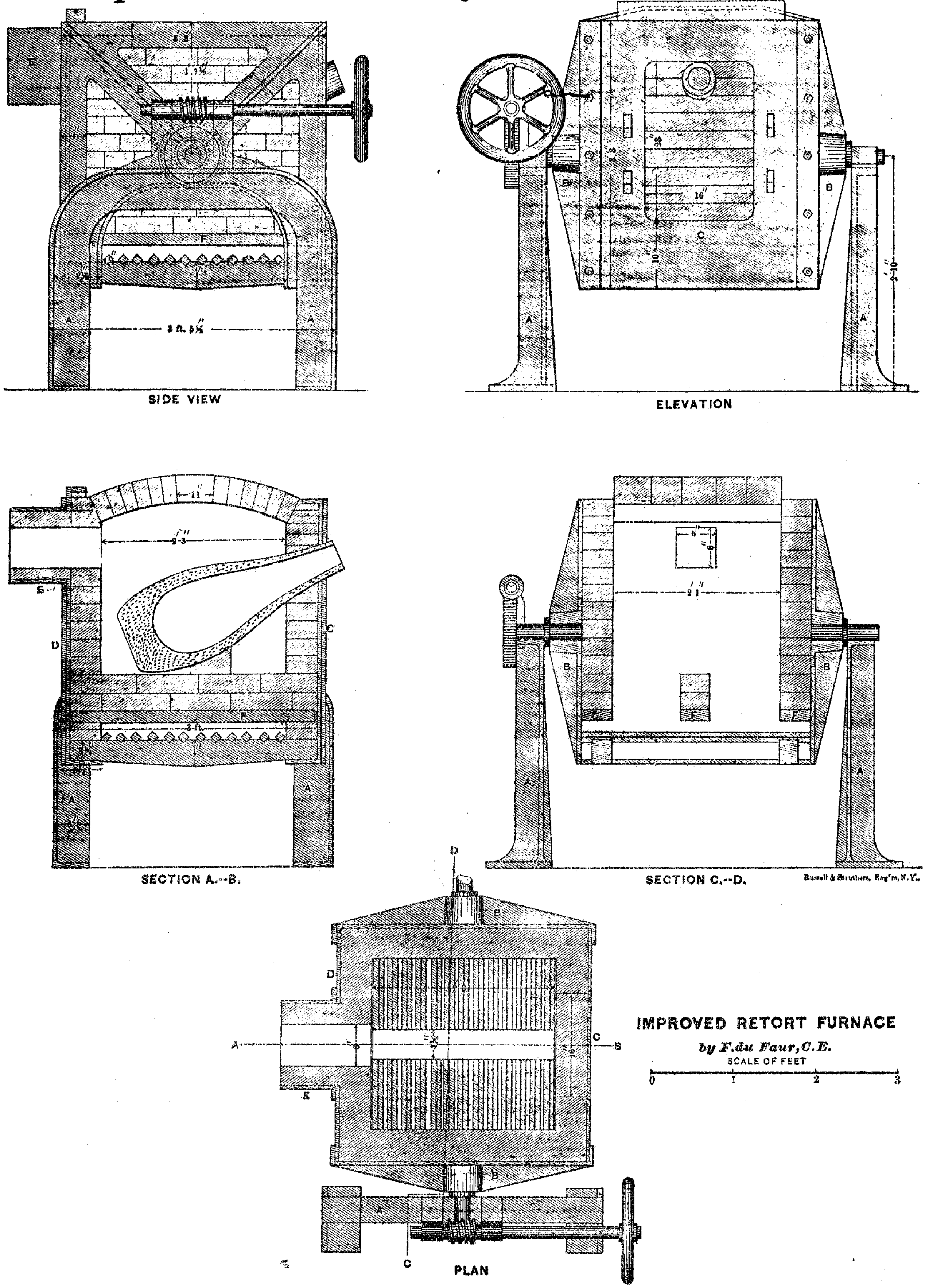
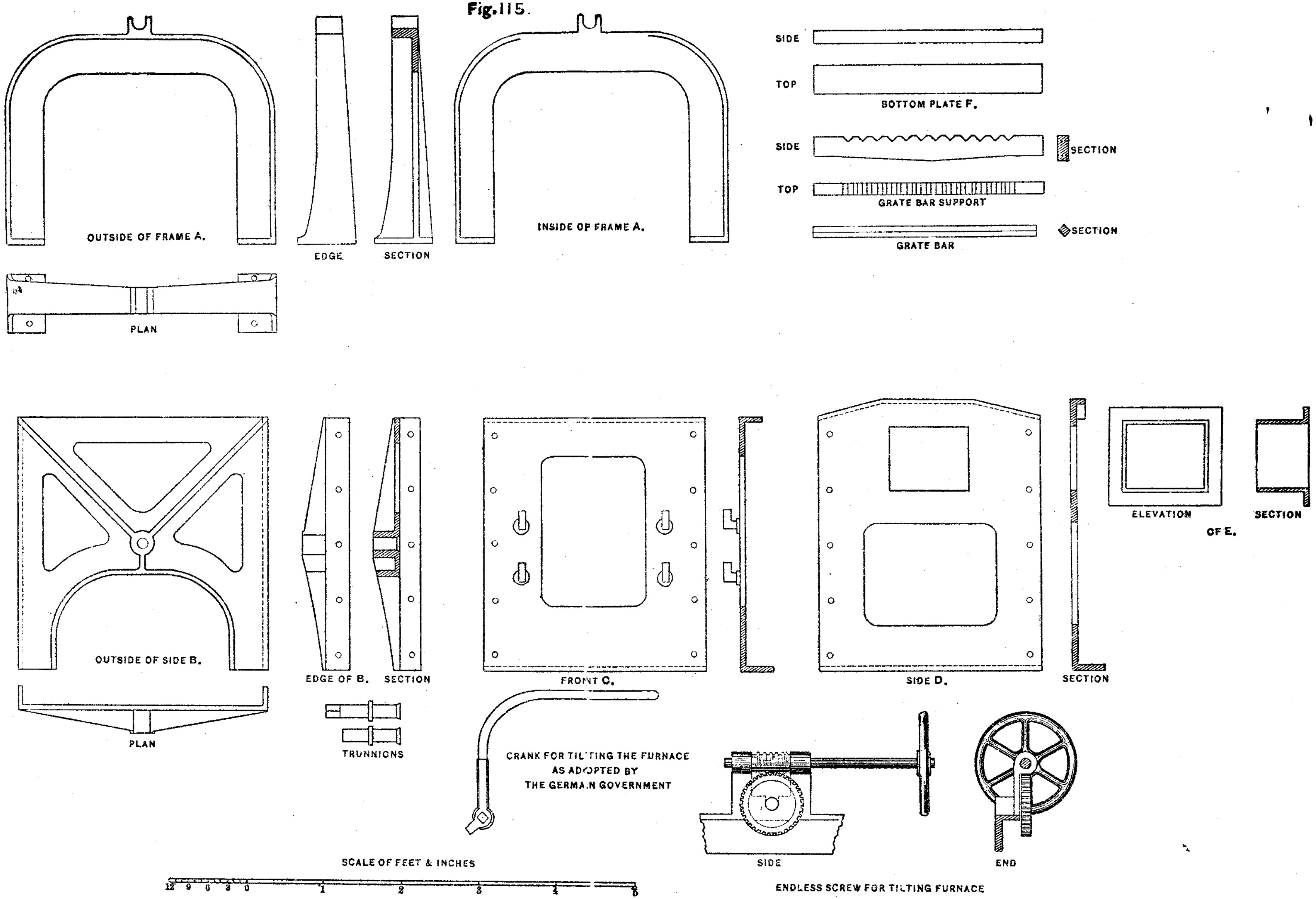

SIDE VIEW
ELEVATION
SECTION A.--B.
SECTION C.--D.
Russell & Struthers, Eng'rs, N.Y.
PLAN
IMPROVED RETORT FURNACE
by F.du Faur, C.E.
SCALE OF FEET
0 1 2 3

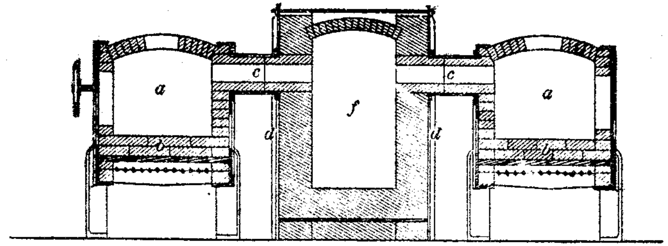
Fig. 115.



DETAILS OF A. FABER DU FAUR'S RETORT FURNACE.

RUSSELL & STRUTHERS, ENG'S, N.Y.

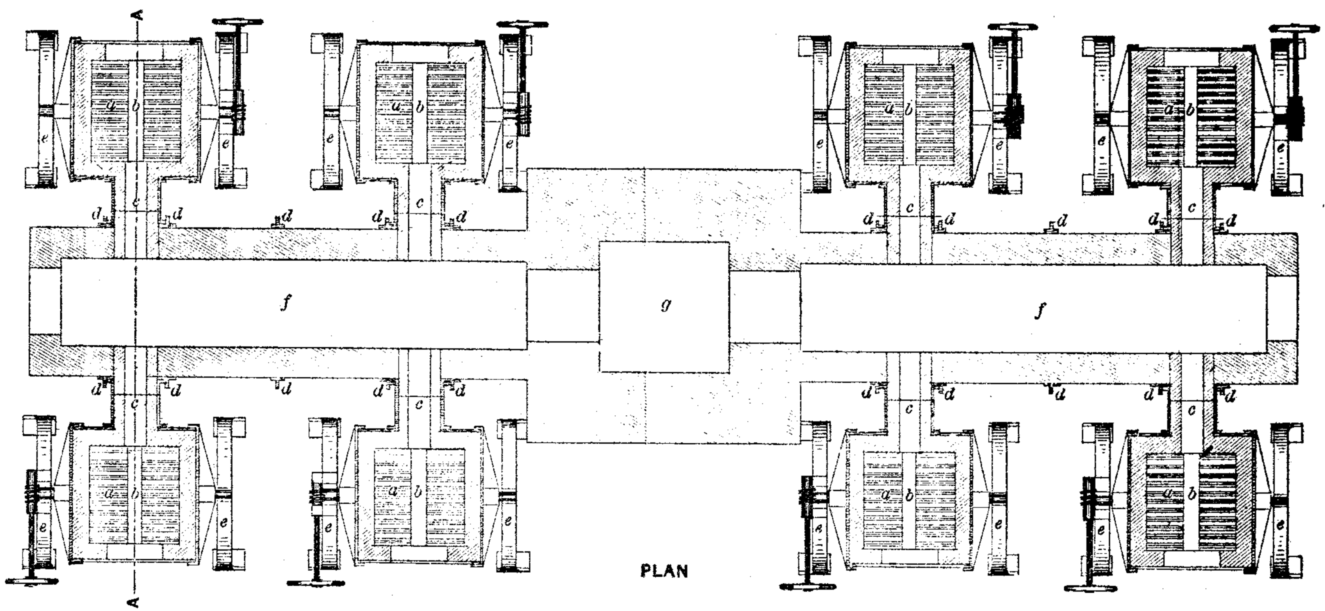
Fig 116.



SECTION A--A

a - *Faber Du Faur's tilting furnace.*
b - *Support for retort.*
c - *Flue jointed so that the furnace may swing.*
d - *Vertical T iron ties holding brick walls together.*
e - *Stands supporting furnaces*
f - *Main flue.*
g - *Chimney 60 feet high.*

Scale In. 12 6 0 1 2 3 4 5 6 7 8 9 10 Ft.



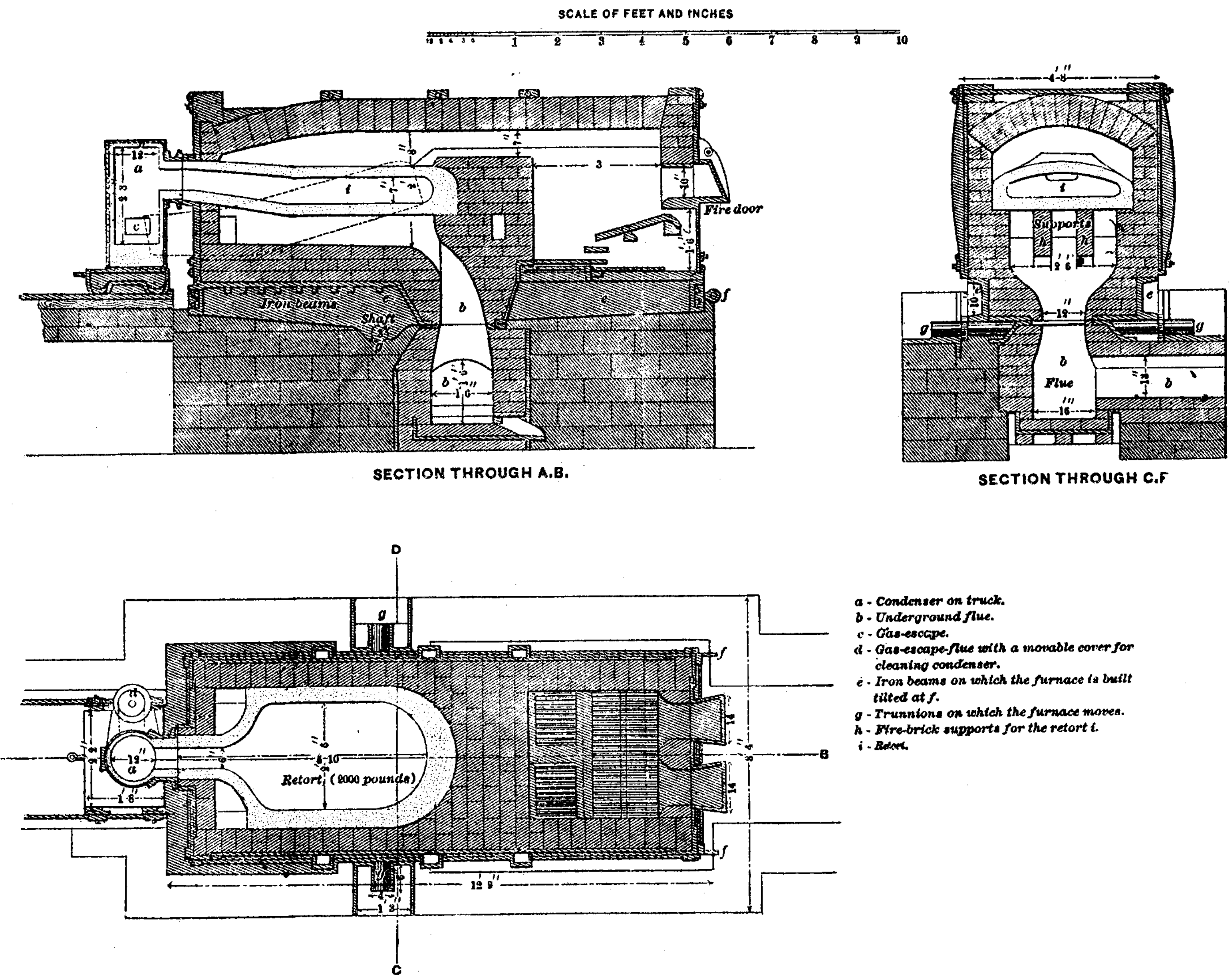
PLAN

Brick *Fire brick* *Iron in section* *Iron in elevation*

PLAN OF FABER DU FAUR'S TILTING RETORT FURNACE AT THE GERMANIA WORKS FLACH'S STATION, NEAR SALT LAKE, UTAH.

TILTING FURNACE

FOR FLAME OR GAS; BY F. DU FAUR.



a - *Condenser on truck.*
b - *Underground flue.*
c - *Gas-escape.*
d - *Gas-escape-flue with a movable cover for cleaning condenser.*
e - *Iron beams on which the furnace is built tilted at f.*
g - *Trunnions on which the furnace moves.*
h - *Fire-brick supports for the retort i.*
i - *Retort.*

Fig 117

Furnace for treatment of Zierrogel Tub Residue.

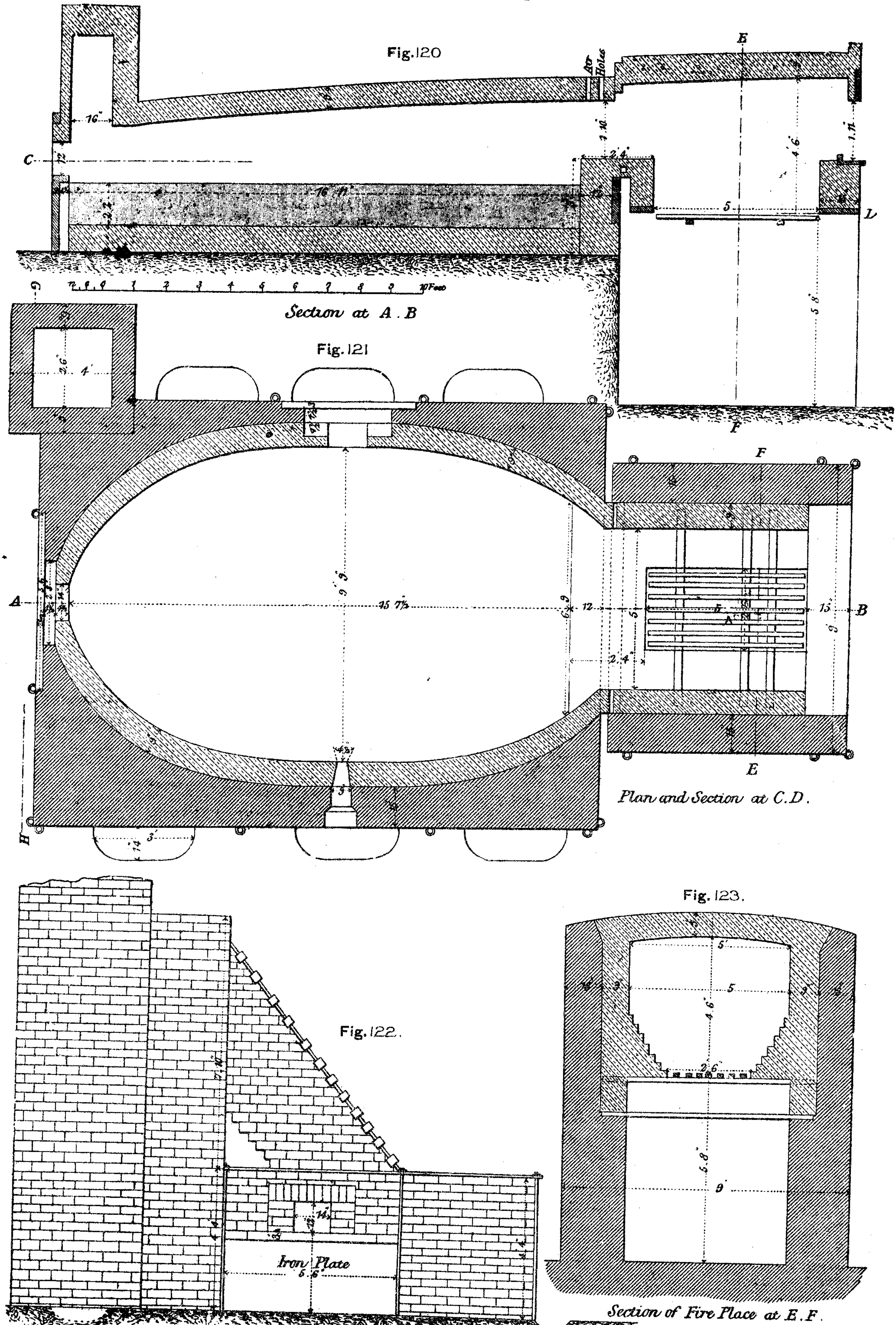


Fig. 120. Section at A . B

Fig. 121. Plan and Section at C . D.

Fig. 122. Elevation at G . H.

Fig. 123. Section of Fire Place at E . F.

Table of charges in the Brodie furnace:—

—	Number of Shifts of Twelve Hours.	Pounds of Zinc Scum liquated in Kettle.	Pounds of Bituminous Coal used for Distillation.	Number of Barrows of Charcoal.	Yield in Rich Lead.	Number of Charges.
No. 1 ...	19	9,916	22,000	6	8,681	65
No. 2 ...	17	13,656	26,000	6	10,862	66
No. 3 ...	21	19,944	14,000 with 1 ton coke	3	14,511	60
No. 4 ...	26	19,622	34,000	...	19,015	73
No. 5 ...	26	27,324	20,000	...	23,738	73
No. 6 ...	28	21,114	...	...	11,927	...
With hot air	28	17,300	...	...	14,902	83

At Cheltenham the retorts are set in the furnace (Fig. 112) with the level of the bottom below the mouth, and so inclined that the siphon can draw out nearly the whole of the silver-lead. Some of it will remain, but this is no disadvantage, as it is not lost. It is collected when the retort is broken. Its presence, however, requires that a reducing-temperature should always be kept up in the retort, otherwise litharge would form and the retort be quickly pierced. The furnace is a cube of fire-brick, 3ft. in cube, braced in every direction with wrought-iron bands 3in. wide. On the top there is a round hole, *h*, 10in. in diameter, for the introduction of the fuel; on the front is an opening for the neck of the retort, *c;* and on the back a square flue, *g*, leading to the chimney. The retort is introduced from the bottom. The furnace has twelve grate-bars 1in. square, and is supported in front on masonry, B, built with two steps, each of which is 18in. high, but vertical behind. The retort is supported on a pillar of brickwork, D, resting on the ground, through which the grate-bars pass. It is thus in the centre of the furnace, and is surrounded on all sides by fuel. It costs from £2 9s. to £3 6s., and lasts from fifteen to thirty turns. When it breaks it is not because it is worn out, but because the workmen break it in trying to force off the cinders attached to it. Five of these furnaces were arranged in a house by themselves about a hexagonal chimney, and connected with it by the flue G, 3ft. long. The sixth side of the chimney was occupied by a melting-furnace. Only three of the furnaces are run at a time, the others being kept in reserve in case of accident or necessary repairs.

The fuel used was at first coke, which was given up because the clinkers attached themselves to the retorts; in trying to remove them the men constantly broke the retorts by poking them, while the cinder was soft, with iron tools, through the opening for the introduction of fuel. Petroleum was then used with great success, but the furnaces were finally abandoned at these works for Faber du Faur's furnace.

The charge of 380lb. of zinc skimmings is introduced with a spoon immediately after the preceding operation is finished. Two small scoopfuls of small charcoal are added at the same time. The heat is so high that most of the charge melts at once. A prolong, *e*, Fig. 112, 2ft. long, 4in. in diameter at the small and 9in. at the large end, is then put on and luted. It is partially filled with charcoal. The prolong is covered on the outside with sheet-iron to protect it against accident. It is supported on a cast-iron shelf, *f*, which can be raised or lowered at will by detaching a bar underneath it. This is necessary to prevent the weight of the prolong breaking the retort while the furnace is working. When the charge is drawn it must be let down so as not to interfere with the siphon (Fig. 113).

The zinc commences to distil in about three-quarters of an hour. Metallic zinc collects in the condenser; some blue powder and oxide of zinc also form there. The object of the charcoal is to prevent the formation of oxide as much as possible. The zinc is allowed to accumulate, and is drawn from time to time with a spoon into a mould placed in front of the prolong. About 60 per cent. of the zinc is recovered as metallic zinc, and is cast into slabs to be used over again; 20 per cent. is recovered as blue powder mixed with oxide; the rest of the zinc is lost. The amount of silver contained in what is recovered is not appreciable. When the zinc is nearly distilled a small piece of wood is put into the retort to make a reducing-atmosphere, to prevent the formation of litharge, which would pierce the sides, and to form a current of gas from the inside to the outside of the retort. The charge of rich silver-lead remaining after the zinc is distilled is drawn with the iron siphon. It must be heated before it is introduced, and is handled with heavy mittens. The lead is cast into pigs ready for cupellation.

Before the invention of the Steitz siphon, the neck of the retort, which was necessarily built into the masonry of the furnace, had to be disengaged while it was at a white heat, before the rich silver-lead could be discharged from the furnace. The percentage of breakage was thus greatly increased, so that, between the necessity of getting rid of the clinkers on the outside of the retort, and the necessity of disengaging the neck every time it was discharged, the number of retorts broken was very large. The siphon proved to be a complete remedy, but was difficult to use—much more so than the polling-pot siphons. The objection to using these furnaces was not only the breakage of the retorts, but the large quantity of fuel they consumed. The Brodie furnace, with two tiers of retorts, consumed less than the Cheltenham furnace, but the retorts were more difficult to manage. The use of petroleum seemed to be a real progress, and the use of gas was proposed, when the invention of the tilting-furnace overcame all difficulties; and it is now almost universally used for this purpose.

The general shape of Faber du Faur's furnace is essentially the same as that at Cheltenham, but it is suspended on pivots, so that it is capable of rotation by means of a worm attached to a hand-wheel, as in the American type of the furnace (Fig. 114), or by means of a lever, as in the German type used in Newark and in Prussia (Fig. 115). The furnace is 3ft. 3in. by 2ft. 11in. in section, by 3ft. high on the outside; and 2ft. 1in. by 2ft. 3in., and 2ft. 9in. from the grate-bars to the centre of the arch, on the inside. There is an opening 11in. in diameter on the top for the introduction of the flue, and on the back a flue 6ft. 6in. leading to the chimney. There are twelve grate-bars, 1in. square and 2ft. 9in. long, set on edge. The retort is built into the furnace in the same way as at Cheltenham.

Fig. 116 gives the plan of the furnace at Salt Lake, showing the disposition of the eight furnaces *a* with regard to the main chimney *g* and a section across the flue *f*. At Mansfield Valley the chimney is at the end of the line of furnaces. The weight of the iron for a furnace is nearly as follows:—

	Lb.
Cast-iron box	1,260
Grate-bar bearers	306
Two standards	530
Cast-iron	2,096
Wrought-iron bars	181

The wrought-iron costs from £31 4s. to £34 7s. The furnace is fired until the retort gradually arrives at a dull-red heat, when a charge of 250lb. to 400lb. of the alloy, broken up while still soft in order to get it of a suitable size for the charge, and mixed with 5lb. to 6lb. of small charcoal, is introduced with a scoop. It is brought to the retorts at Mansfield in a box on wheels about 3ft. by 3ft., and a little lower than the mouth of the retort. As soon as the retort is charged, the temperature is gradually raised to a white heat, and when the zinc-vapour begins to appear, the condenser, made in the same way as that at Cheltenham, is put on. At Mansfield they use for a condenser a retort No. 100, with the bottom broken out, and a hole punched in the side to discharge the zinc. A piece of common stove-pipe is attached to the mouth to carry off the gases.

The retorts usually last fifteen to twenty charges. The time they will last depends largely on the quantity and quality of the ash in the fuel used, and the care exercised by the men when poking the fire. At the Germania Works, with an English coke of 4-per-cent. ash, and careful handling, they have been made to last twenty to twenty-five charges; and with very careful watching they have been made to last forty-five. As soon as the zinc commences to collect, a wagon containing the moulds for the zinc and the support for the condensers is rolled up against the front of the furnace, which has been entirely free since the charge was introduced. The zinc distils, and is collected in the condenser and held there by the oxides and blue powder which collect in front, and are used by the workmen to form a dam to hold the zinc back. When sufficient has collected it is drawn into the moulds. The total amount collected as metal varies from 45 per cent. to 60 per cent., and is used over again. The blue powder and the oxides amount to from 20 to 30 per cent.: these are sold to the zinc-works. Some of the zinc is lost by volatilisation, and from 0·7 to 1 per cent. retained in the lead. The blue powder of the condensers and the dross which forms in the retort are difficult materials to treat. They contain some silver. As the blue powder is mostly metallic zinc, it can be mixed to some extent with zinc in the first additions of zinc used on the charge or added to the blast-furnace charge. It was formerly sold to the zinc-furnaces, but this is not now usually done. The dross, slags, old retorts, cupel-bottoms, litharge, and most of the products containing silver are worked into the shaft-furnace charges, or are sometimes treated in a special reverberatory furnace.

As soon as the zinc escaping appears in small quantities the lead contains but little zinc; but, as it is desirable to remove, as far as possible, the last traces of it, the heat is kept up, the condenser is removed, and small pieces of wood are put into the retort to assist the discharge of the fumes. When no more escape the furnace is tipped down, and the contents of the retort discharged into a lined receiver, and there left until cool enough to be cast into pigs. They generally contain from 2,000oz. to 3,000oz. of silver, and not more than from 0·5 to 0·8 per cent. of zinc. The retort is now carefully scraped with an iron scraper, to remove any slag or other material adhering to the sides. The amount removed in this way is not large; but it is necessary to keep the retort clean, for if the material was allowed to accumulate it might be difficult to remove it, and there would be a risk of breaking the retort in doing so. The material so collected, amounting usually to a few pounds, is reduced with the cupellation litharges. The unburned charcoal is put back into the retort. When the retort is cleaned it is turned up partially, and fine charcoal-dust or a piece of wood thrown in, to make a reducing-atmosphere, and prevent the formation of litharge from the oxidation of the very small quantity of lead attached to the sides of the retort. This precaution is very necessary, for if the litharge was allowed to form it would soon destroy the retort. The furnace is now turned up, and is ready for a fresh charge. The time required for a charge of 350lb. is from eight to twelve hours.

The workmen are obliged to be careful in all these furnaces that in introducing the coke they do not push too hard on the retort, which is quite soft. The fire must be kept at a constant temperature of white heat throughout the operation, which lasts from eight to ten hours, according to the percentage of zinc in the alloy, but when the lead contains antimony it lasts a much longer time.

The only precaution required during the operation is to keep the temperature high enough to prevent the formation of a crust on the surface of the charge. To prevent this, and to know what is going on in the interior of the retort without removing the condenser, it is probed from time to time to break the crust, for if it should form an explosion would be likely to take place. The men can always tell the condition of the heat by looking into the coke-charging hole.

It is very necessary that the current of gas should always be out of the retort. The retort should last twenty operations on an average, and it is generally broken before it is worn out; but when much antimony is present in the lead they last a much shorter time, so that it is always desirable to soften the metal before treating it with zinc. In some of the works, owing to careless management in not carefully cleaning the inside and outside of the retorts, they last for only nine to ten operations. When a new retort is necessary, the furnace must be allowed to cool down, the grate-bars are taken out, and the retort introduced from the bottom. It would seem that there should be no question with regard to the adoption of the tilting-furnace. The most expensive articles used in it, however, are the retorts. In a fixed furnace, like that at Cheltenham, they may, with a little care, be made to last twenty-five or even thirty charges. In the tilting-furnace, however, they do not last much over fifteen. The cause of their decay is on the inside from the formation of lead-oxide, and on the outside of clinker, which is not so easily removed in the tilting as in the stationary furnace. In some works in Europe retorts weighing over a ton, lined on the inside with graphite, have been used in fixed and in regenerative furnaces with success. In the United States they have not, to my knowledge, been used. In some works the outside of the retort has been washed with clay, which does not allow the cinder to adhere to the sides of the retort, or makes it separate easily; in others it is said that they have been coated with a mixture of fine quartz and feldspar, which effects the same purpose.

The flues leading to the chimney at Mansfield are made with flaring sides at the bottom for 18in. in height. The sides of the upper part are vertical, and are rounded at the top. Every 7ft. at the bottom a partition is put in, one-third of the whole height of the flue. In the brick flues, which are 5ft. high, the partitions are put in every 18in., and further apart. In both the iron and brick flues the most dust is caught near the furnace. The dust settles by gravity in these catches, and, as there can be no velocity there, owing to the partitions, it remains there. Short flues of this construction have been found to be much more effective than large condensing-chambers. The experience of some of the best works in Europe is that surface, not volume, is the indispensable requisite to the proper collection both of the particles carried off mechanically as well as those which are volatilised.

The amount of zinc in the skimmings is very variable. If they contain 35 per cent. of zinc, 20 per cent. will be recovered as metallic zinc, and 10 per cent. as oxide, which is afterwards reduced, and 5 per cent. will be lost. This last is either in the lead or volatilised in the different operations. If the skimmings contain only 10 per cent., 3 per cent. will be recovered as metallic zinc, 5 per cent. will be recovered as oxide, and 2 per cent. will be lost. No lead or silver is found in the distilled zinc.

The tilting-furnace is a very great improvement on all those in which the retort is fixed, so far as the work is concerned, as it reduces the labour to a minimum, and at the same time allows perfect manipulation of the furnace; but it does not always allow of using the fewest retorts.

The following table, prepared by Mr. E. F. Eurich, gives the account of two charges in Faber du Faur's furnace at Mansfield:—

Distillation of the Zinc rich in Silver.

	No. 1.	No. 2.
Weight of alloy per charge, with ¾lb. of fine charcoal	353lb.	353lb.
Number of charges	27	20
Number of distillations in twenty-four hours in each retort	2	2
Total amount of liquated zinc crust charged ...	9,525lb.	6,362lb.
Charcoal	108lb.	80lb.
Result—		
Rich lead	7,609lb.	5,221lb.
Metallic scraps	390lb.	Not weighed.
Charcoal with little metal	Not weighed.	"
Metallic zinc	770lb.	"
Blue powder and oxide	Not weighed.	"
Coke used, in bushels of 40lb.	410·4	276
Quantity of coke per pound of zinc crust	1·7	1·73

M. Faber du Faur has proposed another furnace, shown in Fig. 117, constructed on the tilting principle, designed to receive a charge of 1 ton at a time. The retort *i* is made of fire-clay, lined on the inside with graphitz. It is 6ft. 6in. long on the outside, 5ft. 10in. long on the inside, and 7in. high. It is placed on a cast-iron frame *e*, protected by fire-brick, and connects with a condenser *a*, 12in. in diameter and 2ft. 3in. high on the inside, which is placed on wheels, so as to be moved when the retort is to be tilted. The retort is moved mechanically from the fireplace end at *f*. The furnace may be constructed for solid fuel as in drawing; but it was invented exclusively for the use of hot air and gas. The object in the construction of the retort was to have the largest possible surface for distillation, with the shallowest depth of metal, which will not exceed 2½in. to 3in. It was proposed to make the retort in two parts if necessary. This furnace has never yet been built on account of the commercial depression: contracts for its construction were once prepared, but not completed. It seems to have the advantage of being able to treat a large quantity expeditiously, and thus economize in labour and material.

The silver-lead, containing from 2,000oz. to 3,000oz. of silver, is cupelled in an English cupel-furnace. At the Germania Works there are two of these furnaces, at Cheltenham only one. They are blown with a steam-jet in both places. They are usually at work one week, during which time they treat thirty-five bars of 65lb. each per day. The silver is then tapped and the test changed, or the other furnace used. The silver bullion produced weighs about 9,000oz., and is usually 990 to 995 fine, and contains four- to five-thousandths of gold, the proportions of both metals varying with the bullion or ore purchased. The litharges produced are reduced in a reverberatory furnace. At Cheltenham the cupellation is done on a Steitz water-back cupel made of a hollow casting through which water flows. This was formerly made of iron, cast in one piece; but it was found not to answer, as the front, which contained the litharge-channel, wore out rapidly. This part was then made separate, so that it could be quickly replaced when the furnace was working. Recently copper water-backs, with replaceable iron litharge fronts, have been used. The lead is concentrated on the water-back up to 60 per cent. silver; it is then finished on an ordinary English hearth. A very great advantage has been found in thus dividing the operation, as up to about 60 per cent. of silver the concentration can be done by an ordinary workman. The finishing only will thus have to be done by an experienced man. It is also considered that, as the responsibility is more individualised, this method of conducting the operation in two parts greatly diminishes the temptation on the part of the men to steal the silver, and the risk on the owner of losing any of it.

At Mansfield Valley the cupel was formerly made of an artificial marl, made by mixing clay and pulverised limestone together, as is frequently done in Germany. This was stamped into a mould, and the shape of the test cut out of it. Now, however, both here and at Aurora, Illinois, the cupel is made of the best hydraulic Portland cement, moistened enough to ball in the hand, and stamped in an iron mould. The test is 3ft. by 4ft. on the inside. The iron frame which supports it is flanged on the bottom at right angles to the rim, which is 7½in. high, while the flange is 5½in. wide. The test is made either on an iron mould, which gives the shape to the inside, or is cut out of the material after the frame has been stamped full. At first they were always cut out; now they are generally stamped over the mould. When made, the cupels are left to temper for four weeks, to insure a good test. They could be used after a week, but it is better not to do so. The test is supported in the furnace on an iron plate, and is held up to its place by four large screws. The charge of a rich alloy is 1,400lb. The cupel is used for a week, and cupels from ten tons to twelve tons up to 996 fine, and that directly from the lead. The lead is added in the cupel till just before it is too rich, then cleaned off, and the silver is refined, and is run into the brick-moulds directly from the cupel. A little copper is added to prevent the spitting of the silver. The copper absorbs the oxygen and prevents the spitting. When any copper is present in the lead, even when gold is present, it rarely ever spits. When the silver is ready to cast into bricks the test is loosened, and a curved bar is placed on a support made for the purpose underneath it. The whole test is then raised by a Weston pulley, and the silver, tipped at once into the moulds for the bricks, is 994 to 996 fine. The cupel thus allows of casting without refining in a separate furnace. It is the invention of Mr. Eurich, formerly the manager of the Pennsylvania Lead Works, and is one of the many ingenious additions to metallurgical progress which he has made.

Taking into view the fact that lead which contains an appreciable amount of silver, along with impurities which usually accompany the precious metals when they are found in lead, cannot be used in most manufacturing processes, what amount of silver it will be economically worth while to separate is rather a question of the relation between the price of the lead containing these impurities and that of marketable lead than of the quantity of silver contained in it. The value of such material may, however, be approximately estimated by the charges usually made for refining it. For ordinary silver-lead, containing no gold, and of not too high a grade, the ordinary refining charges would be about 95 per cent. of the lead, and all but 3oz. of the silver would be returned. Lead with a lower silver-contents, if free from other impurities, could be treated at a much lower price; but such material is rarely, if ever, found.

DESCRIPTION OF PROCESS USED IN EXTRACTING METALS FROM ORE AT THE BOSTON AND COLORADO WORKS, IN AMERICA.

Considerable interest is now taken in the Australasian Colonies with regard to the treatment of refractory ores, and the systems adopted in America and Europe in dealing with them. So far, the treatment of this class of auriferous and argentiferous ores in the colonies has not proved a success; but this is not to be wondered at, as the old systems are still clung to like a drowning man grasping at a straw, and nothing short of sheer necessity will induce ordinary mining-men to turn their attention to new ideas. The large amount of loss in the treatment of auriferous and argentiferous ores in the Australasian Colonies is really appalling. With the ordinary stamps and quicksilver-tables not more than about 50 per cent. of the marketable products, and in many instances nothing like this percentage, is obtained. The reason of this is, the gold is not always in a free state, but is associated with other metals—namely, sulphur, arsenic silver, antimony, lead, copper, iron, zinc, bismuth, &c. The greater portion of these metals act injuriously on the quicksilver and destroy its affinity for the precious metals.

It is well known that wherever gold and silver are associated with arsenic and sulphur, which is in reality the simplest combination, a large percentage of the precious metals is carried away. Sulphur has a great affinity for these metals, and, being a very light substance, carries away a percentage of them along with it, floating on the top of the water, and a portion of it never settles among the tailings at all, but is carried onwards with the stream; therefore, in treating the simplest combination of auriferous and argentiferous ores something more than merely crushing it and allowing the crushed material to be carried over quicksilver-tables has to be done before a reasonable percentage of the gold and silver is obtained.

Where there are pyritiferous ores with no other combination than gold, sulphur, arsenic, and iron, the pyrites can be collected by concentration; and no better concentrators have as yet been employed than the Triumph and the Frue vanners, which separate the pyrites from the sand very effectually. These concentrates have to be roasted before the gold can be separated; but in roasting the pyrites the sulphur and arsenic can be condensed and collected in condensing-chambers and in the flues, and after the ore is roasted and clean of the most of the sulphur and arsenic it can then be treated either by amalagmation or chlorination. When there are other ores of a more rebellious character, and smelting is resorted to, pyritiferous ores can be used for flux.

The great combination of metals in some of the auriferous and argentiferous lodes in the Australasian Colonies demands far more attention to be paid to the separation of the metals, so that the whole of the by-products can be utilised and rendered a marketable commodity. It is only by this that mining can be carried on successfully. As far as it has been carried on yet in these colonies it is nothing but the extreme richness of the lodes that has made them payable for working. Millions of pounds' worth of metals have been wasted with no attempt to extract them from the ores on anything like an intelligent principle.

This question has received a great deal more attention in the United States. Large plants have been erected for dealing with every class of ore, and some of these have been able to treat the refractory ores very successfully. The ores containing various combinations of metals, along with silver and gold, are separated at the Boston and Colorado Works, near Denver, and from 90 to 95 per cent. of the precious metals extracted. The principal means adopted at these works are roasting, smelting, and leaching. For the treatment of tellurium ores, smelting is the only process by which they can be dealt with satisfactorily. Smelting, however, is but one stage of the process. After the metals have been collected in copper matte the separation of them is a more intricate process, and requires considerable metallurgical skill. The separation of metals is accomplished by what are known as the Ziervogel and the Augustine processes; and, as these systems, together with the zinc desilverisation process, are applicable to the treatment of many metalliferous lodes in the colonies, it is the intention to give a description of these processes, with plans of the works, showing the details of all the principal operations.

The following description is an extract from a work lately written by Thomas Egleston, LL.D., Professor of Mineralogy and Metallurgy, School of Mines, Columbia College, on the metallurgy of gold and silver, in which he describes the different processes adopted in the United States:—

Separation of Gold and Silver from Copper. Boston and Colorado Smelting-works.

The Boston and Colorado Smelting-works were formerly situated in the town of Black Hawk, Gilpin County, Colorado, on the Clear Creek narrow-gauge railway, fifty-five miles from Denver, in the Rocky Mountains, at an altitude of 7,800ft. It was one of the first works erected for the metallurgical treatment of gold-ores, and the only one established in Colorado on a large scale which has been uniformly successful. The works were planned and built by Professor Hill, formerly Professor of Chemistry in Brown University, Providence, Rhode Island, and is still managed by him, assisted by Mr. Richard Pearce, formerly professor in the school of mines at Truro, England. The works are very advantageously situated with regard to the ore-producing regions, having Boulder County on the north, which produces, besides ordinary gold-ores, a series of tellurium minerals, such as altaite, sylvanite, and hessite, which are very rich in gold and silver. They are associated with copper- and iron-pyrites, blende, galena, and the oxides and carbonate of iron.

Gilpin County itself produces for the most part pyrites both of iron and copper, rich in gold, with a small quantity of galena and blende which is rich in silver. In some of the mines native gold is found. Clear Creek County, to the south, furnishes mostly galena and blende very rich in silver. The works also receive mattes rich in gold and silver made at Alma Park County, and tellurium ores rich in gold from the southern part of the territory.

The works are thus located in the very centre of the gold- and silver-producing regions of Colorado, and are also favourably situated with regard to transportation. They treated in 1874 30 tons of ore and tailings every twenty-four hours, and produced 700,000oz. silver, 12,000oz. to 15,000oz. gold, and 225 tons copper. With matte from Alma their production in 1875 was 110,000oz. silver, 25,000oz. gold, and 250 tons copper. In the year 1880 they produced £118,333 of gold, £461,667 of silver, £61,875 of copper; or a total production of £641,875.

Both gold- and silver-ores are treated, and gold, silver, and copper produced. The lead is not separated from the ores, nor paid for if it exists, and is entirely lost in the residues. In the year 1878 the works were removed to Argo, near Denver, where they are even more favourably situated than formerly.

Gold-ores.

The gold-ores are divided into three classes. The first class consists of auriferous copper-pyrites, containing from 2 to 10 per cent. copper, 2oz. to 10oz. gold, and 2oz. to 10oz. silver. These ores average 4 per cent. copper, 3½oz. gold, and 6oz. silver. The second class are tailings from gold-mills, consisting of pyrites with about 1½ per cent. of copper, 1¼oz. gold, and 4oz. silver. The third class are tellurium ores which have a very siliceous gangue, and contain 100oz. to 200oz. gold, and 6oz. to 10oz. silver. These ores come mostly from Boulder County, and are often worth £2,083 to £3,125 to the ton.

Silver-ores.

The silver-ores of the first class consist of surface-ores mostly free from sulphur, containing 70 per cent. silica. They contain 100oz. silver, and 5 to 6 per cent. lead, and no gold. The second class are sulphurets, rich in blende, and poor in galena and pyrites. They contain 150oz. silver, 15 per cent. zinc and lead, and no gold.

The cost of material at the works is:—

Wood	£1 0s. 10d. per cord.
Fire-bricks	£18 15s. per thousand.
Common bricks	£2 18s. per thousand.
Iron castings	4d. per pound.
Wrought-iron	4d. per pound.

The cost of delivering the silver in New York is $1\frac{1}{4}$ per cent. of its assay-value, taken at the valuation at the works. The rate for gold is $\frac{6}{10}$ per cent. The general plan of these works is given in Fig. 117. The diagram indicates the various processes, showing what becomes of each of the products in the different stages. The treatment is comprised in eight distinct operations, most of which are more or less subdivided. These operations are—

1. Sampling the ore.
2. Roasting the ore.
   a. Large ore roasted in heaps.
   b. Small ore roasted in a reverberatory furnace.
3. Fusion for matte.
4. Ziervogel's process.
   a. Crushing and roasting the matte for sulphate of silver.
   b. Precipitation of silver.
   c. Washing and fusing the cement-silver.
   d. Precipitating the copper.
   e. Refining the cement-copper.
5. Treatment of the Ziervogel tub-residues.
   a. Fusion for white-metal.
   b. Roasting white-metal.
   c. Treatment of purple-metal.
6. Treatment of the residues of the Ziervogel process by the Augustine process.
7. Treatment of the bottoms.
8. Treatment of the copper-alloy.

1. Sampling the Ore.

The ore is purchased in large and small sample-lots, varying from 50lb. to 6 tons or 7 tons. It is sampled by first taking one-tenth of the lot and putting it through a Dodge crusher and a pair of Cornish rolls, and then sampling as usual.

The following prices are paid by the Boston and Colorado Smelting Company for ore delivered in car-loads at their works at Argo, Colorado, in the year 1885.

Gold- and silver-ores carrying no copper, less than 10 per cent. of lead or zinc, and less than 5 per cent. of arsenic:—

Ore assaying	Percentage of Assay-value of Gold and Silver.	Less per Ton. £	s.	d.
Less than £20 per ton of 2,000lb.	90	3	2	6
From £20 to £40	92	3	2	6
From £40 to £60	94	3	2	6
Over £60	95	3	2	6
Ores carrying copper	90	3	2	0

Ores containing less than 10 per cent. of lead or zinc, and less than 5 per cent. of arsenic, containing gold and silver:—

	Per Unit of Copper. s.	d.
Over £1 13s. 4d. a ton in gold and silver, for 1 per cent. of copper	4	2
From £1 0s. 10d. to £1 13s. 4d.	3	9
Below £1 0s. 10d.	3	4

No charge is made for crushing, sampling, and assaying the ore if in car-loads; if there is less than a car-load a small charge is made for handling. They purchase no ores with over 10 per cent. of lead. Ore assaying less than this is purchased according to the scale given above, no allowance being made for lead. Special prices are given for gold- or silver-ore carrying iron- or copper-pyrites and less than 10 per cent. of gangue, and also for matte and other furnace-products. Deductions are made from the scale for ore having a large percentage of zinc or arsenic.

All the large pieces of gold-ore are roasted in large heaps and are then passed through a crusher and rolls, and afterwards passed through a screen with four-to-the-inch mesh. The tellurium ores are only crushed and passed through a screen with ten-to-the-inch mesh, and are then ready for smelting. The surface silver-ores are crushed and passed through a four-to-the-inch screen, and then go into the furnace. The ores rich in sulphur are called heavy ores, and are crushed and calcined in a reverberatory furnace.

2. Roasting the Ores.

(*a.*) *Roasting the Ore in Heaps.*—The auriferous pyrites are broken to 2in. square in a crusher, and roasted in heaps of about 50 tons each. The piles are made in the usual way, with a wooden chimney about 7ft. high in the centre, wood being used as fuel. The amount consumed is two cords for 50 tons. The wood is burnt out in about twelve hours, at which time the sulphur commences to burn. The pile is lighted at night, because the moisture in the fuel makes sulphuretted hydrogen, which would annoy the men in the day-time. The fire, except in cases of accidents, burns until the roasting is complete. The sampler takes charge of the piles. He has little to do except to throw fine ore on the cover when he sees there is too much flame. He has two or three assistants, and with them he does all the weighing and sampling, and takes care of the piles.

When the pile is finished—that is, the roasting is completed—the outside crust of unburned pyrites is taken off and put on to the next pile. The roasted ore is crushed and goes through a sieve, No. 4 mesh, and is then ready for the smelter. One man does the whole of the crushing.

The roasting is completed in about six weeks from the time the fire is lit, and the amount of sulphur remaining in the ore is about 4 per cent. As the ore contains a considerable amount of arsenic, the pile is frequently covered on the outside with crystals of arsenious acid, which are often white, but generally coloured with a slight trace of sulphur or sulphide of arsenic. They are generally formed where there has been a hole in the cover of the pile, and their usual form is that of an octahedron with hollow faces.

(*b.*) *Roasting the Ore in Reverberatory Furnace.*—The ore submitted to this process is said to be calcined. The tailings and finely-divided copper-ores are roasted in a reverberatory furnace, called a calciner, until they contain not more than ½ to 4 per cent. of sulphur. There are six of these calciners in the works. They are marked K in the ground-plan, Fig. 118, and shown in detail in Figs. 119, 119*a*, and 119*b*. Only three of them are in use at one time: two of these work into the same flue. Each furnace has three step-hearths 10ft. long. The total length of the furnace is 40ft. on the outside, including the fireplace, and their width is 11ft., having six working-doors, two doors to each hearth. The hearths are about 4½in. in height, the one above the other, and are equally divided in the length of the furnace. Each one is rectangular, with the usual waste of space at the doors filled in. The two at the end have their corners rounded.

On comparing the relative dimensions of these furnaces it will be seen that the surface of the hearths is 304 square feet, and the surface of the grate 16 square feet. If the fireplace be taken as unity, the relation between the surfaces will be as 1 to 19. The fireplace is arranged for long sticks of wood, and has a door at the side. It is 5ft. long and 2ft. 8in. wide. The bridge has an air-hole in it 4½in. square, and communicates with the interior of the hearth by four openings. The width of the bridge is 28in.; the height of the roof above the hearth at the bridge is 28in., and 18in. at the flue end.

The furnace is built of red bricks, fire-bricks being used only in the fireplace and on the first hearth. A charge of 1 ton is introduced on the hearth nearest the flue, so that there are 3 tons in the furnace at one time. The charge of ore on each hearth, spread out after being first put in, is 3in. deep, but it swells so as to be 4in. to 5in. deep on the hearth nearest the fireplace. This is particularly true of the tailings. As the charge is drawn once in eight hours, it takes twenty-four hours to complete the roasting of 1 ton of ore. The furnace is worked by one man on each shift of twelve hours each, who brings in his own wood and does all the labour; so that two men work 3 tons of ore in twenty-four hours. The ore is brought to the furnace-men, who then make the charge. One man brings all the ore for three furnaces. The men from the calciners always assist in charging the calcined ore into the matte-furnaces. The furnace burns 1½ cords of wood in twenty-four hours. The repairs to the furnace are very slight, one day in twelve months being sufficient to do all that is required.

3. *Fusion for Matte.*

The roasted ore is fused in a reverberatory furnace for matte. There are three of these furnaces, which are marked D on the ground-plan, and are given in detail in Figs. 120, 121, 122, and 123. Only two of them are in use at a time. They are constructed to use wood, so that the fireplace, which is 5ft. at the top of the bridge, is only 2ft. 6in. at the grate; it is 5ft. long and 4ft. 6in. from the grate to the roof. The opening in the fireplace for charging the fuel is at the end of the furnace, and not at the side as it usually is. The fireplace-door is of cast-iron; it slides in a groove, and is counterpoised with a weight. The bridge is 2ft. 6in. wide, the fireplace side 2ft. 3in. and the laboratory-side 1ft. 10in. from the roof. Just above the bridge there are a series of openings, 3in. by 1in., for the admission of air in the roof, which follow on the roof the contour of the laboratory in two rows, the outside having eight and the interior eleven holes each. The laboratory is 15ft. 7½in. long by 9ft. 9in. wide. The working-door is at the end; the two openings at the side are closed for this operation. In comparing the relative dimensions of the furnace, we find that the surface of the fireplace at the height of the bridge is 25 square feet, and that of the grate is 12½ square feet. The laboratory has 143·18 square feet; so that, the fireplace being taken as unity, the relation is as 1 to 5·7. Each of these furnaces has its own chimney, which is 50ft. high. The arrangement of the holes in the roof is a very ingenious one, for, as the fireplace is very deep, and is constantly filled with long sticks of wood to a depth of over 3ft., the wood distils and forms gas, which is burned by the air entering through these holes. Before this method was introduced by Professor Pearce there was not sufficient air to produce a perfect combustion. The immediate effect of its introduction was the saving of fuel and more equal distribution of heat. Formerly the flue connected with the chimney was constantly burning out, and required frequent repairs.

An opening has recently been made at the foot of the chimney for the introduction of cold air, both because the cold air is mixed with the products of combustion on leaving the furnace, and also that the combustion is better regulated. By this means the repairs to the furnace are greatly diminished. The hearth of the furnace is slightly inclined towards the working-door, and also to one side. It is made of two layers of brick, upon which fine quartz-sand is placed, which is mixed with a small quantity of wood-ashes, and then agglomerated.

When the hearth is made the temperature is lowered and the charge is introduced. The charge is made up of a crushed heap of roasted gold-ores as follows: Roasted tailings, 2,000lb.; oxidized silver-ores, 1,500lb.; roasted silver-ores, 1,500lb.; raw pyrites, 300lb.; fluor-spar, 350lb.; rich scorias, 500lb.

After the charge is drawn the furnace is repaired, if necessary, with clay, which is beaten in with a ladle-shaped tool attached to a long handle. Such repairs are usually not made oftener than twice a week. The charge is introduced with a shovel by the side door, the ore being put in first, and then the rich slags. The charge is so arranged that 10 tons of mixed ores will produce 1 ton of matte. It does not do to make the matte richer, as there are always grains of it in the slag, and so the loss would be greater. The slag is carefully calculated so that it will not be too basic, or otherwise it would cut the fire-brick to get silica. The charge is evenly distributed over the surface of the hearth, which is almost at a cherry-red heat.

It takes six men, working in groups of three at a time, nearly a quarter of an hour to make the charge. As soon as it is made the charging-door is built up and luted or closed with sand. The furnace is then charged, and is left with the full power of the draught for five or six hours. During this time the workmen clean up the slag-bed and attend to the fire, which requires looking after every twenty minutes. At the end of this time they stir the furnace carefully five or six times to bring up everything from the bottom, which should be perfectly smooth to the tool passing over it. This produces reaction. The furnace is now left in repose for twenty minutes to effect the separation of the scoria and the matte. If lumps are found the stirring is done again, and kept up during the firing, or for about one hour, after which the slag is drawn with a rabble into moulds prepared for it. The latter operation takes about twenty minutes. When the door is open to skim the slag the latter is quite hot and fluid, and there is a constant but quiet ebullition of sulphureous and sulphuric-acid gas, the bubbles being about 1in. in diameter, and quite uniformly distributed. Professor Pearce asserts that the larger part of the gas is sulphuric-acid. At the close of the skimming, as the slag becomes cooler, the bubbles become larger and less uniform. Just before the skimming pieces of sheet-iron 3ft. by 2ft. are placed in front of the slag-bed and to one side of it, to protect the workman from the heat. The casting-bed is 10in. deep in front of the furnace to receive the plate-slag, which ordinarily contains all the grains of matte.

The casting-bed has fourteen divisions, which are connected with one another. When the slag, which covers the matte to the depth of about 3in., is being skimmed off it is easy to distinguish the matte below, which shows a dark colour and a more or less brilliant surface. As the rabble goes backward and forward the slag does not close at once over it, and the surface is exposed for a short time. When all the slag is drawn off a new charge is introduced, four charges being made in twenty-four hours.

When the matte is to be tapped all the doors of the furnace are opened so as to chill the last part of the slag a little, so that it will not flow out from the tap-hole. It is then tapped and made into plates 3ft. long, 14in. wide, and 4in. thick in the middle, the bottom being rounded. No slag flows out with it, as it is too much chilled. When all the matte has been tapped the tap-hole is closed with damp sand. The operation of tapping the matte and stirring takes about half an hour each charge, making about fourteen plates. Three men per shift of twelve hours are required to work two furnaces, and eight cords of wood are consumed in twenty-four hours.

The plate-slag contains on an average 5 per cent. of copper, but it is often poor enough to be thrown away with other slags. It is generally a silicate of protoxide of iron, but it is sometimes more basic. The poor slag contains about 7oz. of silver and a trace of gold. This is considered too poor to treat, and is thrown away. All the slag richer than this is put back into the furnace. The matte contains from 25 to 30 per cent. of copper, from 20oz. to 30oz. of gold, 600oz. to 1,000oz. of silver, and some iron, lead, zinc, and antimony.

When the hearth bottom of the matte-furnace becomes loose and rises, as it sometimes does, the whole hearth-material is taken out and crushed, and treated as ore. The floors of the furnace have to be repaired every two or three months, and the roof is made over once a year. The outside walls last for a number of years before it is necessary to rebuild the furnace.

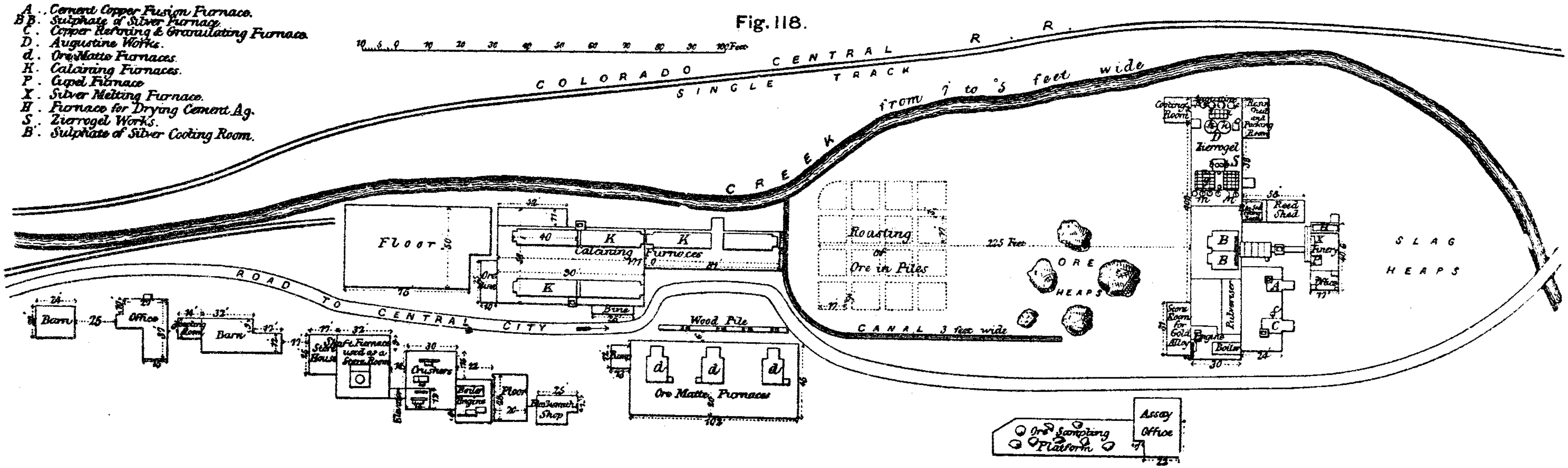
4. *Ziervogel Process.*

(*a.*) *Crushing and Roasting the Matte for Sulphate of Silver.*—The matte produced from the previous operation must be roasted, and for this purpose it is crushed fine. It is broken up by sledges, and then put through a Dodge crusher, with which one man can crush about 10 tons per day. After the crushed material is put through a No. 12 screen, it is taken to the calciners marked K on the general plan, Fig. 117, when it is roasted for twenty-four hours, a charge being drawn every eight hours, the charge being 1 ton on each hearth, so that there are always 3 tons in the furnace at one time. One furnace working constantly does the whole work of the establishment. 90 per cent. of the sulphur is removed by this operation. The roasted matte contains about 5 per cent. of sulphur, partly as sulphides and partly as sulphates. On the hearth where the matte is charged the furnace is dark. This is necessary to prevent fusion, as there must be rapid oxidation at the lowest possible temperature. When the workman is not attending to the fire he is always rabbling the charge. When the charge on the one hearth is finished, it is moved on to the other by a spadelle. On the middle hearth the heat is very dull, and from this temperature it is gradually raised until it is withdrawn from the furnace. On the last hearth the temperature is a bright cherry-red. The charge is drawn with a rabble into a "cub" beside the furnace. As there is but a small amount of sulphurous acid given off, the roasted matte remains here until it is cool enough to be wheeled in wooden barrows to be operated on by the ball pulveriser. One and a half cords of wood in twenty-four hours is all the fuel used in this operation.

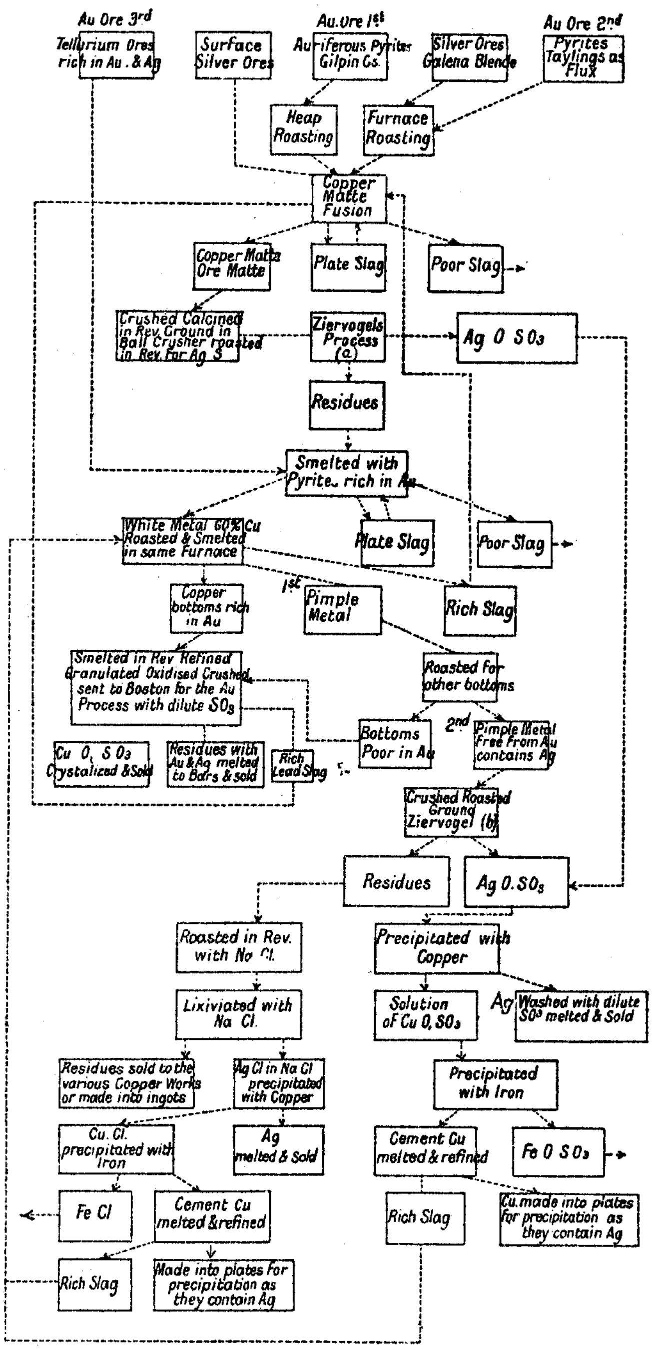
The ball pulveriser consists of a stationary horizontal sheet-iron cylinder 4ft. in length and 2ft. 8in. in diameter, inside of which another cylinder of less diameter revolves. This inside cylinder is made with a cast-iron head-piece, into which cast-iron bars are fitted so as to leave a space of $\frac{1}{64}$in. between them. These bars are kept in position by a flange and wedges, and the heads are securely bolted together. The material to be ground is introduced into the revolving cylinder through a trough in its axis. This cylinder or grinder contains one half-ton of iron balls, which when new are 3in. in diameter.

The cold calcined ore from the cubs is thrown on to the crusher-floor and shovelled into bins, from which it is carried by an endless chain to a hopper which communicates with the charging-

Fig. 118.



Plan shewing the different Works.



General Arrangement of Reduction Works.

Fig 124.

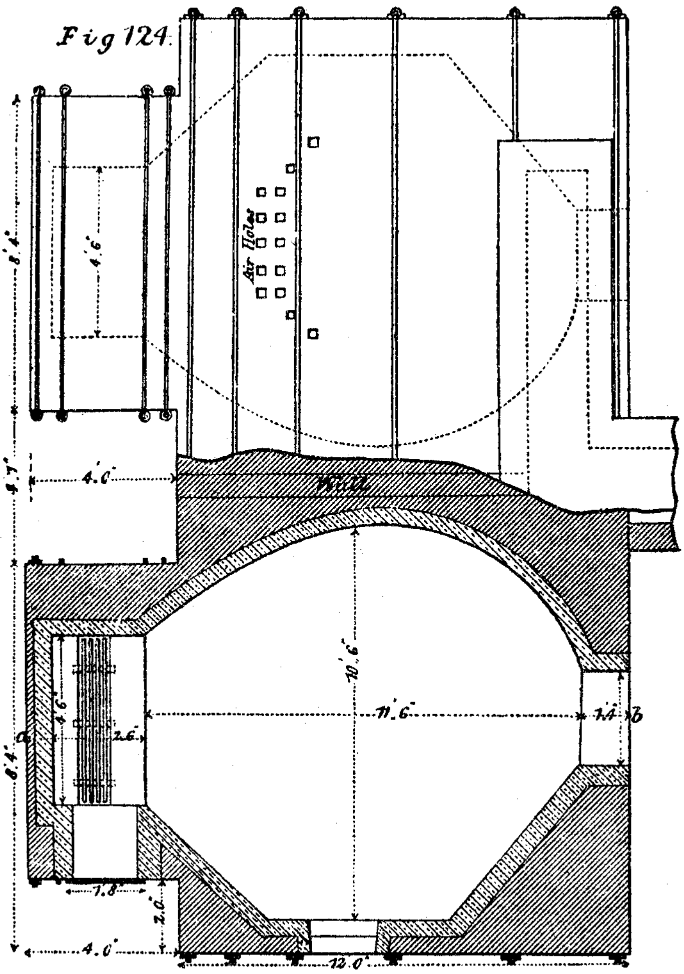
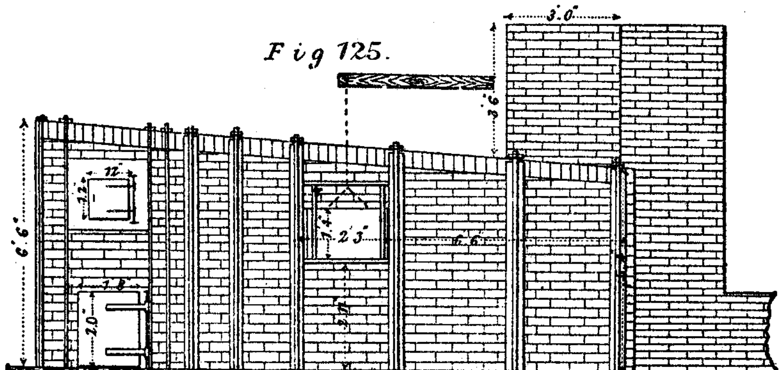
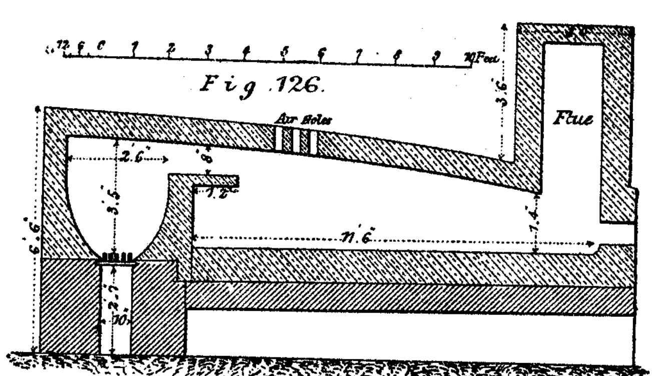


Fig 125.

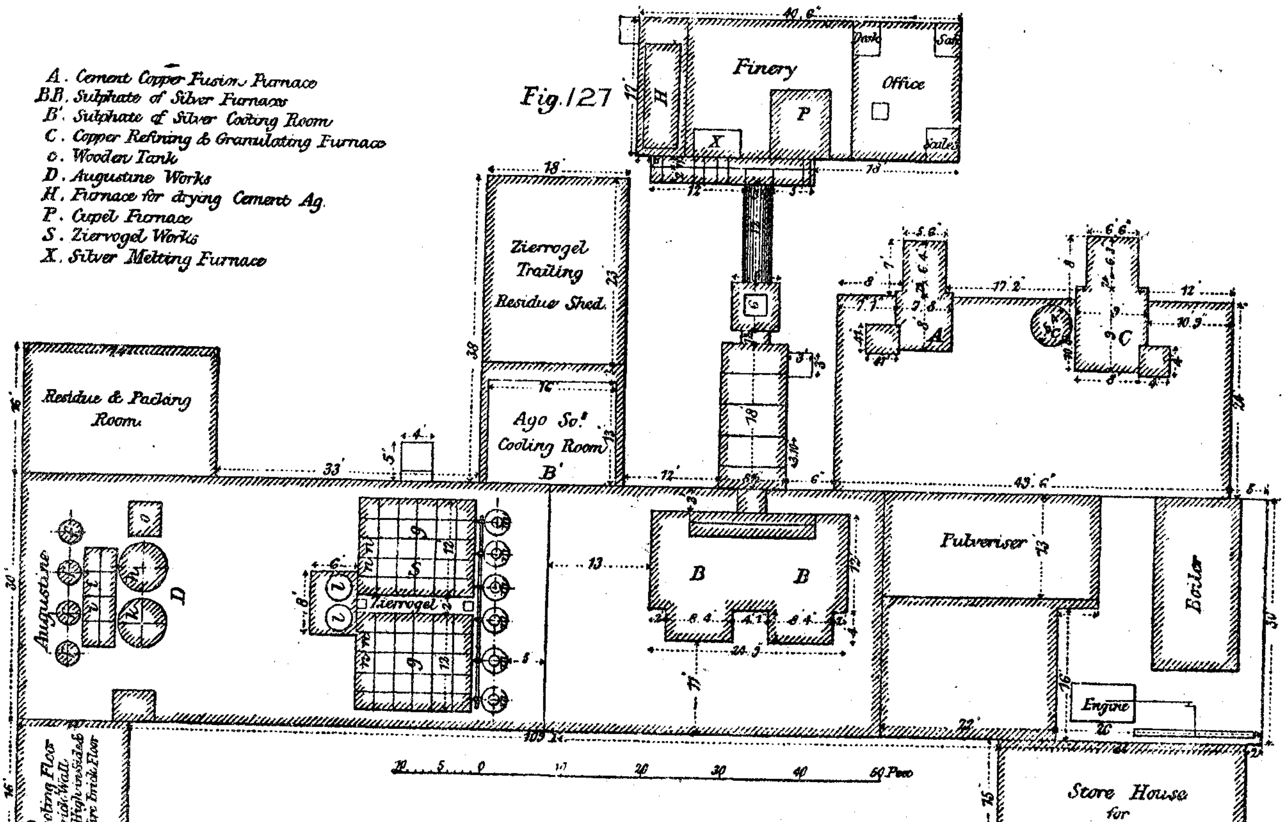


FINE CALCINING FURNACE.

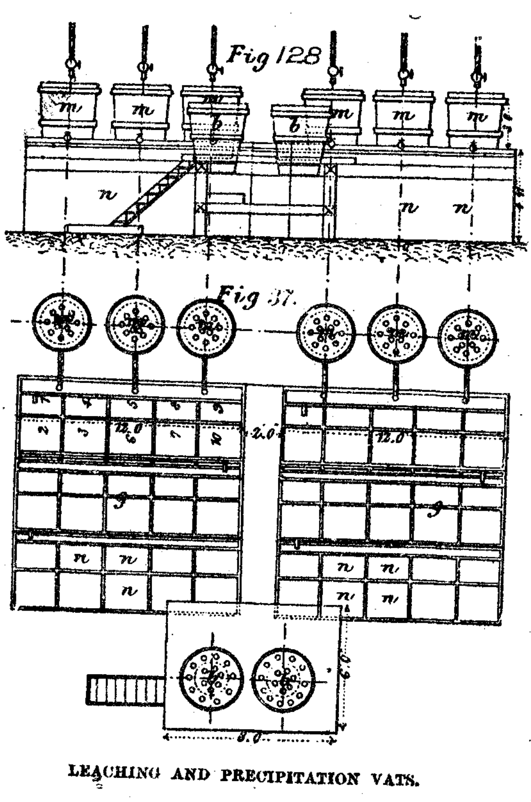
Fig 126.



FINE CALCINING FURNACE.

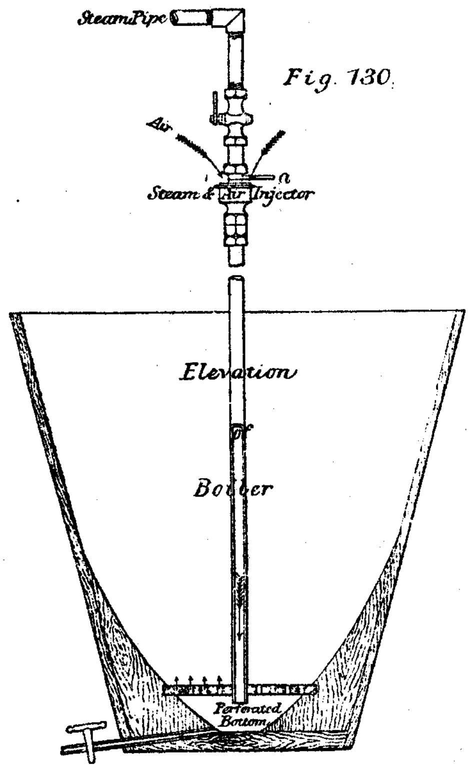


GENERAL PLAN OF REDUCTION WORKS.

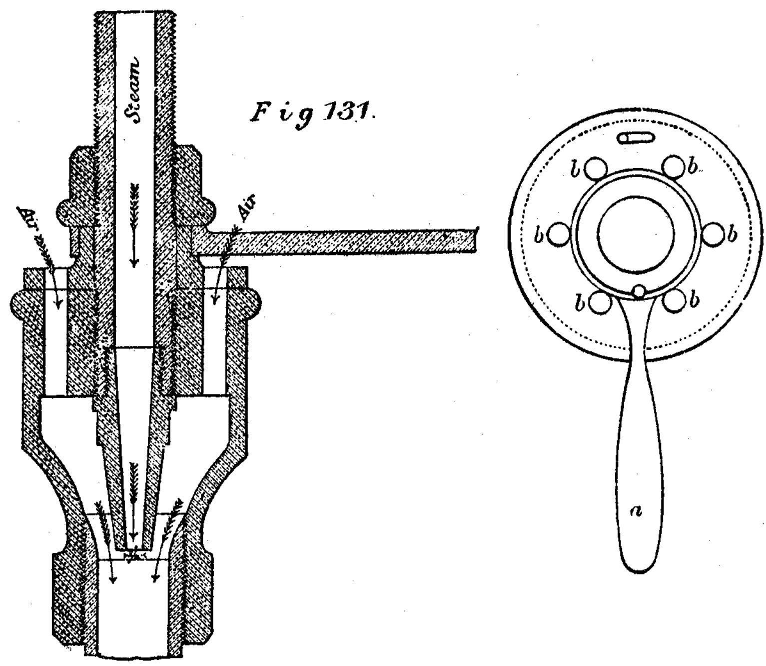


LEACHING AND PRECIPITATION VATS.

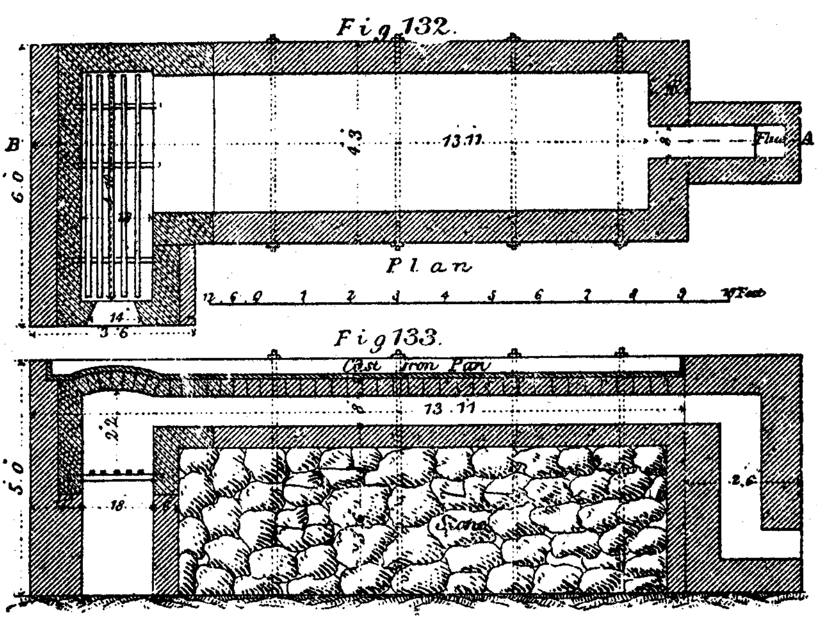
Fig. 130.



PEARCE'S WASHER.



DETAILS OF PEARCE'S WASHER.



DRYING FURNACE.

trough. The charge and balls revolve at the rate of thirty-seven revolutions per minute. The ore which is ground sufficiently fine passes through the spaces between the bars, and falls into the stationary cylinder, which is hopper-shaped at the bottom, and communicates with a trough through which an endless chain passes and carries the ore to a No.-60-mesh screen. What remains on this screen is carried back to the grinder. This crusher works between 3 and 4 tons in twenty-four hours, and has besides plenty of time for the necessary stoppages for repairs. Six tons might easily be put through in ten hours, but from 3 to 4 tons is all that is required, so that a single crusher is more than sufficient. The machine requires very little repairs: the bars wear, and when the openings get too wide new bars are put in. Not more than 500lb. of balls are worn out in course of a year. The men who do this work are obliged to wear wet sponges over their mouths in order to protect themselves from dust. One man, who also carries the wood to the calciners, brings the ore, and one man, who shovels the ore and attends to the grinders, is all that is required for this work.

Roasting Sulphate of Silver.—From the ball-grinder screens the ground matte is conveyed to a bin ready to be roasted for sulphate of silver. The furnace in which this operation is conducted is called the fine calciner. There are two of them, marked B B in the general plan (Fig. 26), and shown in Figs. 124, 125, and 126. They connect with four small dust-chambers, which are common to both furnaces, and connect with the same chimney. They are constantly in use except when silver is being melted, when only one of them is run. They have but one hearth, which is 11ft. 6in. long, 3ft. 6in. deep, and 10ft. 6in. wide. This hearth is flat. The fireplace is 4ft. 6in. long, and 2ft. 6in. wide at the bridge, the grate being only 1ft. wide. There are 11¼ square feet of surface in the fireplace, and 100 square feet in the laboratory, thus making the relation 1 to 9. The top of the bridge is 8in. from the roof, the bridge itself being 2ft. wide, but 14in. of this width is a curtain-arch, the bottom of which is 16in. above the hearth. Just beyond the curtain, in the roof of the furnace, there are a series of holes for the admission of air, of the same size as in the matte-furnace. The first line goes straight through the roof, and is composed of five holes. The second follows the contour of the furnace, and is composed of nine holes. The hearth of this furnace is made on a bed of old slag or stone, covered with sand; on these, bricks set on end, laid in cement, are placed, which form the hearth proper.

The charge is 1,600lb. of roasted matte, which is thrown in with a shovel, and made into a pile on the centre of the hearth. Just before it is introduced all the dampers are closed. The hearth of the furnace at this time is closed. The fireplace is, however, glowing, but contains only embers just sufficient to keep it hot. As soon as the charge is introduced it is levelled with a rabble, and spread out over the hearth. When spread out it is about 3in. thick. It takes about twenty minutes to do this work, during which time the dampers remain closed, and no fuel is put into the fireplace. As soon as the charge is completed the damper is slightly raised, but no fuel is charged. In about one hour the charge has a dull blackish glow. The surface looks black, but it is red when stirred. The fireplace is now charged with a small amount of fuel, and the temperature gradually raised so as to keep it at a dull-red heat. The fireplace-door is closed, and the supply of air comes from the bridge-holes, the working-door, and the grate. The work at this stage consists of forming a maximum amount of sulphate of iron and some sulphate of copper, but the silver remains unchanged.

The fumes of sulphuric acid commence to be given off from the decomposition of the persulphate of iron, and the charge increases in volume, becoming spongy. As the furnace-door is open the workman is exposed to the acid fumes, and is therefore obliged to wear a respirator. The stirring is kept up, and the heat gradually increased. From the second hour the grate is kept full until the end of the operation, the temperature being kept as uniform as possible. The ash-pit door is closed after the first hour, the air only entering through the working-door and the holes in the bridge. The flame over the curtain-arch is curly, blackish, and reducing; but, as there are more than 14in. between it and the charge below, and the working-door constantly opened, it is so mixed with air that in contact with the charge it is oxidizing. At the end of this time the heat is at its maximum and the charge becomes dry, no longer sticking to the rabble.

At this point, which is at the end of three hours, the sulphate of silver is formed, the sulphate of iron being decomposed at the end of two hours. The sulphate of copper at the time all the iron is decomposed is at its maximum, which is at the end of the third hour. When the silver is "out" a bar 2in. square and 14ft. long is used to break up any lumps, the charge being collected with it to the middle of the hearth. The pile is then, by a sliding motion of the bar on its side, cut down, bruised, stamped, and broken up, and in this way turned over twice from one side of the furnace to another. In order to facilitate this work the front of the door is provided with a roller, on which the bar rests. The whole charge by this means is ground fine, and all the lumps broken up, and a perfect oxidation secured. It is essential to have as little sulphate of copper as possible, but 1½ per cent. is left, so as to be sure that no sulphate of silver is decomposed. This operation with the bar lasts one hour, so that at the end of four hours the charge is ready to be withdrawn. At the end of the third hour assays commence to be made, and are constantly taken until the end of the operation. The first assay generally shows that the sulphate of silver is free, but it is reduced almost instantly into a metallic state by the suboxide of copper present, and spangles are formed which scintillate and sparkle, forming a most beautiful reaction.

To make an assay a sample of the hot charge is simply thrown into cold water in a small dish, so that its temperature is raised to above boiling. Whatever silver is in a state of sulphate is dissolved by the boiling water. If there is any suboxide of copper present the spangle reaction takes place.

At the end of the fourth hour the exposure of the surfaces to oxidation from the action of the bar has converted all the copper from suboxide into protoxide, and no spangles are seen in the assay; the sulphate of silver consequently remains permanent. If any sulphide of silver was present in the charge, it is attacked by sulphuric acid given off by the decomposing sulphates, and converted into sulphate. An average of from 90 to 95 per cent. of silver is thus rendered soluble, the rest being in the condition of arsenides, antimonides, or as very fine particles within sulphate of lead, and not decomposed.

The charges are constantly assayed, and the workmen, as they are skilled men, feel it for their interest to conduct the operation properly. It would not be safe to decompose the whole of the sulphates of copper, since there would be a danger of some of the sulphate of silver being decomposed and passing into the residues. The copper gives a blue colour to the solution, so that when the spangles are no longer produced, and the liquor is a very pale-blue colour, the charge is drawn. None of this work is done at night, as the operation is exceedingly delicate, and requires to be constantly watched. As soon as the charge is drawn the furnace is cooled, by opening the doors and dampers, to get ready for another charge. Only two charges a day are made in the furnace. It takes about ten minutes to discharge the furnace. The charge is drawn with a rabble into an iron barrow and wheeled to a brick cooling-floor, shown at B on the general plan, Fig. 35. Each furnace is tended by one man only. The two furnaces burn together 1¼ cords of wood in twelve hours. The repairs to these furnaces are very small, about one day in twelve months being sufficient for this purpose.

Leaching the Sulphate of Silver.—The charge from the sulphate-of-silver furnace is allowed to remain twelve hours on the cooling-floor, and is then leached in tubs. These tubs are 3ft. high, 3ft. in diameter at the top and 2ft. 6in. at the bottom, as shown in Figs. 36 to 38. They are provided with a double bottom, pierced with holes, and a cloth filter. They are charged with 1,500lb. of the matte which has been roasted for sulphate of silver. The leaching is done by a current of boiling water kept hot by steam. The tubs are constantly full, and discharge into a series of tanks below. It takes eight or nine hours to leach the charge. At first it is light, but in an hour it shrinks and the water passes less freely through it.

The residues in the tubs contain all the gold and some silver which has not been separated. They are taken out and put on one side to be treated by the Augustine process, to separate the silver, and the residues are afterwards treated for gold. These residues are in some instances re-roasted, as they contain a small percentage of silver. All the sulphate of copper is dissolved out in the first stages of the work. In about seven or eight hours assays of the liquid are made, and the hot water stopped when salt added to it shows no trace of silver. The time required varies according to the richness of the matte. Between 600lb. and 700lb. are leached in eight hours. Generally about an hour is required for every 100oz. of silver the matte contains.

(*b*.) *Precipitating the Silver.*—The hot water charged with the sulphates of silver and copper from the solution-tubs is run into a series of vats, shown in Figs. 127, 128, and 129, and on the general plan, Fig. 117, at S.

These vats are 12ft. long, 4ft. wide, and 2ft. 3in. deep. Two of these vats, one in front of the other, are placed before each series of tubs. Each of them is divided into ten compartments, which are 24in. by 20in., and 27in. deep. The liquid is discharged from the tubs into No. 1, and communicates with No. 2 at the bottom. The partition between Nos. 2 and 3 is low at the top, so that the liquid overflows into No. 4, and No. 4 communicates with No. 5 at the bottom, and so on. At No. 10 the overflow passes into the tank below, and follows the same circuitous course. Each compartment in the tanks is filled with plates of copper ¼in. thick, and 14in. by 19in. in size. Twenty of these plates, each having a precipitating-surface of nearly 400 square inches, are placed in each compartment.

In the bottom of the tanks the plates are placed upright and are slightly inclined, being separated from each other by a small strip of wood at the top. Over this the plates are laid horizontally, with strips between each to prevent actual contact. This arrangement gives about 100,000 square inches of precipitating-surface to each cistern. Both series of tanks are filled with copper in the same way. The tanks, while the precipitation is going on, are covered with wooden covers. At the end of a week they are removed, and the copper plates shaken and washed in the liquid to remove the silver sponge, which falls to the bottom and is taken out. This sponge is very light, and adheres very slightly to the copper. After the copper plates are taken out the liquid is allowed to settle.

The copper-solution is drawn into the tanks *n n*, and the silver into the tubs D, to be washed, to remove any traces of copper. It takes about two hours to get the tanks ready for another charge. More than half of the silver is deposited in the first four compartments. Here the copper plates last about four months. In the other compartments they last about twelve months. The amount of copper dissolved is equal to the quantity of sulpheric acid set free from the silver.

(*c*.) *Washing and Fusing the Cement-silver.*—The cement-silver is washed in a washer invented by Professor Pearce, and patented in England about the year 1866. It is a tub about 4ft. in diameter at the top, 2ft. at the bottom, and 4ft. in depth. This tub, with its injector, is shown in detail in Figs. 130 and 131. Two of these washers are placed on a raised platform, having a spout connecting with the sulphate-of-copper tanks. About 3,000oz. of silver are placed in the false bottom of the tub. A mixture of 1 part of sulphuric acid to 100 parts of water is then placed in the tub in sufficient quantity to cover all the silver. Steam at a pressure of 50lb. is then turned on through the injector, and the arm moved on so as to open the air-holes. The steam and air pass down through the false bottom and up through the silver and sulphuric acid. A very violent ebullition is caused in the liquid by this passage of the air and steam.

The silver is thus kept in constant agitation, and fresh surfaces are constantly exposed to the action of the acid. Besides this mechanical effect, the current of air oxidizes the metallic copper, and transforms it, together with the sub-oxide, into sulphate. The cement-silver from the tanks still contains some traces of copper as sulphate, and some metallic copper detached from the plates. This injector is also used in parting rich auriferous copper-alloy, for the separation of gold and the manufacture of sulphate of copper.

At the end of two or three hours the liquid is run off through the spout into the tanks. The silver is then washed for half an hour with clean water and steam, and then removed to be dried on top of the drying-furnace (Figs. 132 and 133). It requires from three to three and a half hours to completely purify the 3,000oz. of cement-silver. After drying the silver is melted in graphite crucibles (Figs. 134 and 135). It is cast in heated iron moulds smeared with oil (Fig. 135). The bricks are worth from £250 to £312 10s., and are from 999 to 999·5 fine.

(*d.*) *Precipitation of Copper.*—The copper-solution from the tanks G (see general plan, Fig. 118) runs into one of the tanks (Figs. 128 and 129) which are divided into compartments like the tanks G, and are also covered. These compartments are filled with scrap-iron, which is simply thrown in without any special care in piling. The spent liquor, which is sulphate of iron, when sulphuretted hydrogen on a polished steel plate shows no trace of copper, is discharged into the waste-tank, the velocity of the discharge being regulated according as the action is quick or slow. The copper precipitates on the iron, and is left to accumulate. The compartments are cleaned out about once a month. The copper is removed from the iron by simply moving it backwards and forwards in the liquid. The iron so cleaned is at once placed in an empty tank to be used on a fresh charge. All iron used is old scrap-iron, and is therefore not weighed. About 5,000lb. to 6,000lb. are used in each tank.

The cement-copper is allowed to drain and dry, and is then taken to the smelting-furnace. It contains about 90 per cent. of metallic copper when it is fresh. The small amount of impurity is owing to the fact that the tanks are closed, thus preventing the precipitation of insoluble compounds of iron. The cement-copper oxidizes very rapidly in contact with the air, so that when ready for the furnace it does not contain more than 80 per cent. of copper in a metallic state.

(*e.*) *Refining the Copper.*—The furnace in which the cement-copper is refined is shown at A in the general ground-plan (Fig. 126), and in detail in Figs. 136 to 141. The fireplace is 5ft. long and 28in. wide at the top of the grate. The grate has the same length, but is only 17in. wide. The bridge is 2ft. wide. The laboratory is 6ft. 4in. long and 4ft. 4in. wide, and has two doors, one at the end, which is the charging-door, and one at the side, which is the working-door. Just over the bridge (Figs. 136 and 141) there are two rows of six openings each for the introduction of air. These are covered with a hood to prevent the introduction of foreign substances. The furnace connects with the chimney by a flue which is 2ft. square.

The fireplace has 11·65 square feet, and the laboratory 27·79 square feet, so that the relation between them is nearly 1 to 2. This furnace runs over a month for eighteen or nineteen hours a day. A charge of 2,500lb. copper mixed with 50lb. refuse charcoal is put in at 6 p.m. The fireman keeps up the fire during the night, and the refiner takes it at 7 a.m., and then skims off the slag and exposes the surface of the bath. Considerable sulphurous acid is given off, probably from the reduction of the sulphate of iron in the cement-copper. The charge is worked for a "set," which takes three or four hours. This is done by striking the surface with the rabble, and making waves, which is termed "beating the copper." The copper produced contains from 2 to 3 per cent. oxide of copper dissolved in it; but it is not necessary to refine it completely, as it is used at once in the tanks G. The copper is taken from the furnace with a ladle, and is poured into a mould made of cast-iron, which is slightly tapering, being larger at the bottom than at the top. This mould or frame is placed on a cast-iron-plate 3in. in thickness. The ladleful of copper poured in is allowed to set—that is, a film of suboxide of copper is allowed to form. Another ladleful is poured on, and so on until the mould is full. The cast-iron frame is then removed, and the plates fall out separately, as the oxide prevents anything more than contact. Twenty-five plates are made in this way at one time.

5. Treatment of the Ziervogel Tub-residues.

(*a.*) *Fusion for White-metal.*—The residues from the tubs consist of oxides of copper and iron, with 20oz. to 30oz. gold and 40oz. silver to the ton. They amount to about 22 tons per week. They are melted in the matte-furnace (Figs. 120 to 123) with rich gold-ores of the first class, containing iron, with copper-pyrites and variable quantities of gangue and highly-siliceous tellurium-ores. All the siliceous pyritiferous ores are selected for this purpose. The ores are all crushed and put through a four-to-the-inch-mesh sieve. The charge is brought to the furnace in alternate barrows of residues and ore; but it is not mixed before charging, as it becomes mixed after it is thrown into the furnace. The charge consists of—tub-residues, 4,000lb.; raw gold-ores of the first class, 2,500lb.; gold-ores of the third class, 900lb.: total, 7,400lb.

When there are no tellurium-ores the charge of gold-ores of the first class is made to amount to 3,400lb. The treatment is exactly the same as before. The slag, containing only 2oz. of silver and a trace of gold, is very much poorer than those of the previous fusion. It has otherwise very nearly the same composition as the others, but there is no zinc, either as blende or oxide, in it.

The matte contains—copper, 60 per cent.; gold, 55oz.; silver, 130oz.; sulphur, 30 per cent.

It is called white-metal. If the matte was made richer in copper the slag would also be richer, and there would be consequently more loss. The tapping is made twice in twenty-four hours. In other respects the labour, fuel, &c., is the same as in the matte-fusion No. 4. This fusion for the tub-residues takes place once a month, and lasts a week. All the plate-slag produced during this operation is put directly back into the furnace.

(*b.*) *Roasting the White-metal.*—At the end of a week all the mattes produced are recharged in large lumps, the charge being about 4 tons. It is roasted at a dull-red heat for about ten hours, with admission of air. The reaction which takes place between the sulphide and the oxide makes a peculiar noise, which can be heard at some distance from the furnace. The operation is termed "roasting for black copper," but it is stopped at half-way. As the sulphur is drawn off some metallic copper is liberated. The slag is very thick, and not more than 200lb. to 300lb. are produced. It contains from 8 to 10 per cent. of copper, and is highly basic, often containing crystals of magnetite. At the end of the ninth hour the doors are closed and the fireplace charged. The whole furnace is brought to a high heat, so that the whole charge is in intimate fusion. Just before tapping it is rabbled for five minutes and then tapped into sand-moulds. The tapping is done as before, but moulds are made to receive the mattes, as the charge is greater. In the first three or four pigs there will be found plates or bottoms of metallic copper, containing, arsenic, antimony, and lead. These bottoms contain nearly the whole of the gold, with from 3 to 5 per cent. silver and 80 per cent. copper. The matte is purple-metal, and contains about—copper, 75 per cent.; gold, 2oz.; silver, 120oz.

From every charge about 600lb. of bottoms and 3 tons of matte are produced. The bottom-fusion takes three days: making ten days for the treatment of the residues. The labour is the same as in the matte-fusion, but more wood is used, four cords being burned in twenty-four hours. Only two operations are made in twenty-four hours.

(*c*.) *Treatment of the Purple-metal*.—The purple-metal is roasted again in the same way, treating it nearly five hours and making four charges in twenty-four hours. Other bottoms are produced, poorer in gold, but containing—gold, 60oz. to 100oz.; silver, 300oz.; copper, 75 per cent.; sulphur, 25 per cent.

The purple-metal from this fusion contains—gold, ¼oz.; silver, 110oz.; copper, 80 per cent.; sulphur, 20 per cent.; the iron being entirely removed.

This operation takes a day and a half. The bottoms are treated with the other bottoms. The purple-metal goes to the Ziervogel process B, but is kept entirely separate because it contains no gold, as does that of the process A.

6. *Treatment of the Residues of the Ziervogel Process.*

(*b*.) *By the Augustine Process*.—The residues from the Ziervogel process B, which contain 25oz. silver to the ton, are roasted with salt in one of the furnaces B (Fig. 127), for roasting for sulphate of silver in the Ziervogel process. The residues are charged moist, a charge being 1 ton. It is heated for two hours until it is hot; 20lb. of salt are then added, and well rabbled into the charge for fifteen minutes.

The charge is then drawn to prevent the loss of copper, as well as chloride of copper. Three charges are made in twelve hours. This requires one man and three-quarters of a cord of wood.

Solution.—The material is treated with a hot saturated solution of brine, a tank holding 1,000 gallons of the brine-solution being always kept in reserve. 1,600lb. of the chlorurised residues are placed in a vat, and the solution allowed to constantly flow through it by 1in. for four hours.

The liquid which runs out of the solution-tubs run into tanks (Figs. 142, 143, and 144), when the silver is precipitated with copper and the copper with iron, as in the Ziervogel process. The salt-solution contains chloride of iron, and is pumped back into the tanks and used again. Chloride of iron by constant boiling becomes perchloride, and finally sesquioxide, and is precipitated. The salt-solution lasts, with occasional renewals of water, indefinitely. The loss of salt per ton of residue treated is about 10lb. The residues from this treatment are either reduced and made into ingots, or sold as they are as residues. The precipitation is the same as in the Ziervogel process, except that chlorides are formed. The material is always kept separate.

7. *Treatment of Bottoms.*

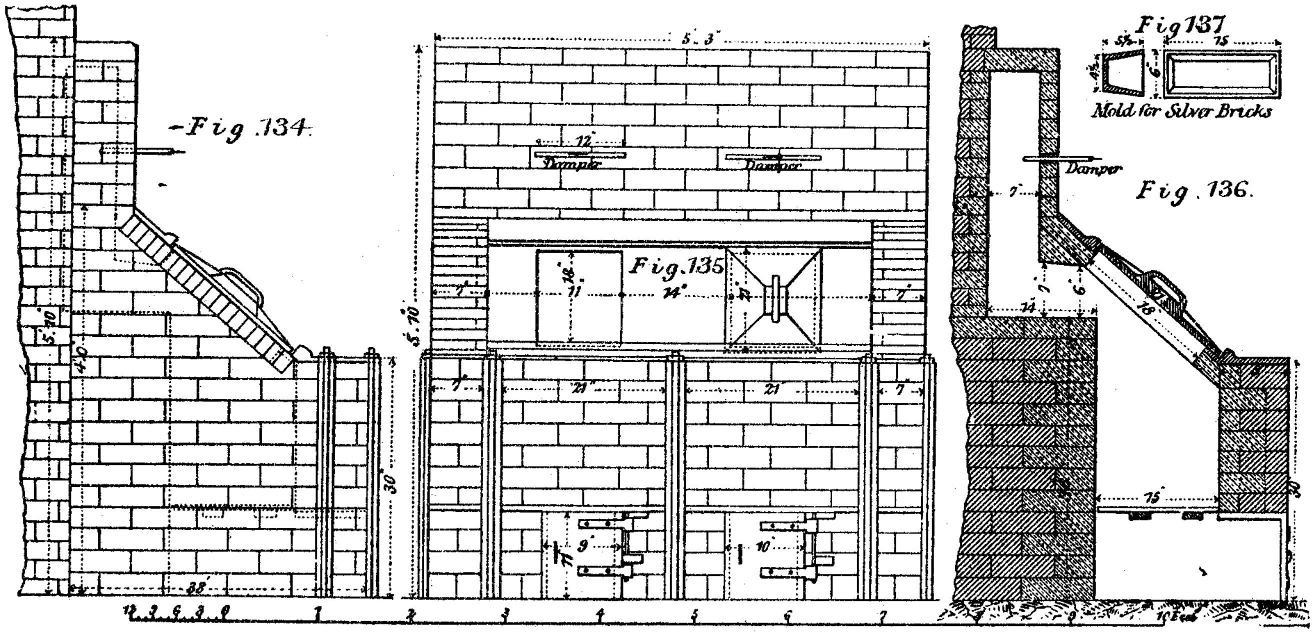
Four tons of white-metal from the Ziervogel treatment gives 600lb. bottoms. These are left to accumulate until they amount to 3,500lb., enough for a charge in a small reverberatory furnace. The furnace in which this operation is effected is shown in the general plan C, and in detail in Figs. 145 to 150. The fireplace is 6ft. long, 4ft. deep, 3ft. 6in. wide at the bridge, and 20in. at the grate. The bridge itself is 4ft. wide. The laboratory is 9ft. long, 6ft. 9in. wide, and connects with the chimney, which is 2ft. 6in. square, by a flue. The surface of the fireplace is 21 square feet, and that of the laboratory 46·27 square feet, the relation being 1 to 2·2. The furnace has a working-door at the side and a charging-door at the end. On the side opposite the working-door there is a spout which ends in a wooden tank sunk in the ground, which is 4ft. 5in. in diameter and 3ft. deep. The object of the process is to oxidize the lead and other impurities, and to prepare the metal for treatment for gold. The charge is made at 7 a.m. It is first sweated at a low temperature for two or three hours, during which time some of the lead liquates and runs out of the furnace. It is then left to oxidize for three or four hours. In about seven hours the charge is well melted. The slag, which is skimmed at the time, is composed mostly of oxides of lead and copper, containing from 10 to 15 per cent. of copper, and is sent to operation No. 3.

After the slag is withdrawn the bath is beaten with a rabble for about two hours, all the doors being open to admit an excess of air. It is again skimmed, and tapped into water. The "pitch"—that is, the condition of the copper—must be such that the whole of the sulphur is eliminated before the oxygen is absorbed. If the pitch is right the globules will be all round and hollow. This point must be seized with the greatest nicety, for if the charge remains too long in the furnace the globules will cast solid, and the charge must then be put back and worked with sulphur. The temperature of the water governs the size of the globules—they are small when it is cold, and large when it is hot—but it does not otherwise affect it. It takes about ten minutes to do this casting. The copper flowing from the spout falls on to a pole of green wood held underneath it so as to scatter the copper. Care must be taken that the slag does not flow with the copper. To prevent it the doors are opened, so that the slag is cooled until it is pasty. One charge is made at a time, and only one or two in a month. The globules contain 1,000oz. gold, 600oz. silver, and a trace of lead. Twenty tons of white-metal gives 1 ton of refined auriferous copper. Three cords of wood are used. One man attends the furnace, and one man does the firing.

8. *Treatment of the Oxidized Copper-alloy.*

The copper globules are oxidized in one of the five calciners in which the sulphate of silver is treated. One and a half tons is charged at a time. The oxidation takes thirty-six hours. The globules are put into the furnace in a heap, and spread over the hearth. The charge will be 3in.

General Arrangement of Reduction Works.



FURNACE FOR MELTING CEMENT SILVER.

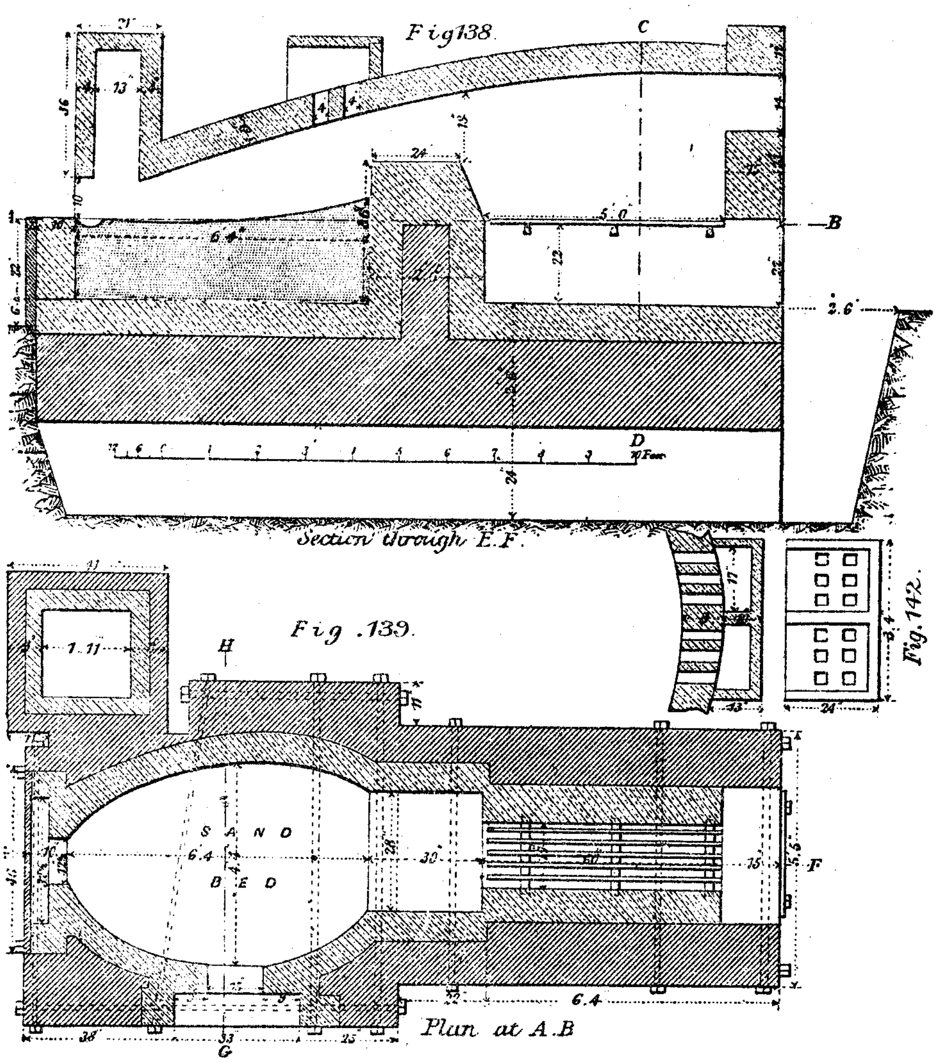


Fig. 140.

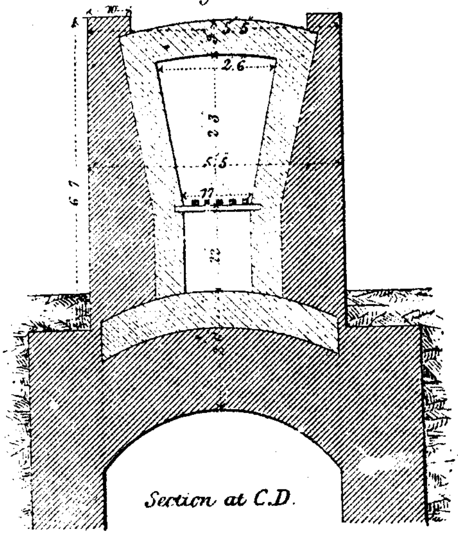


Fig 141.

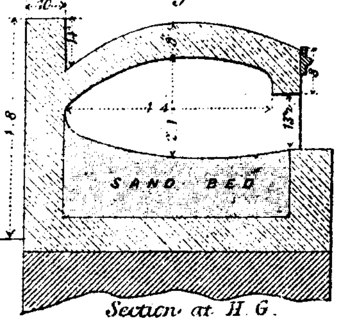


Fig 141a.

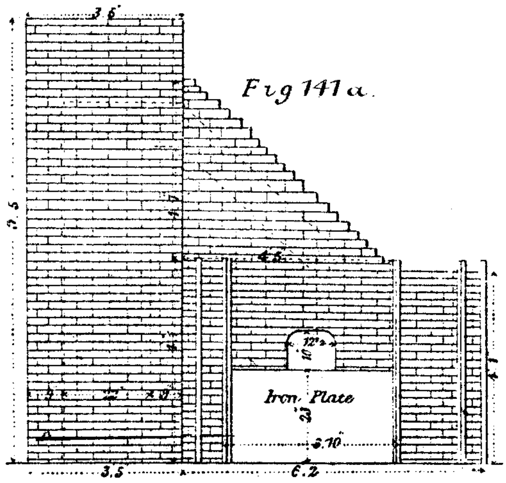


Fig 144.

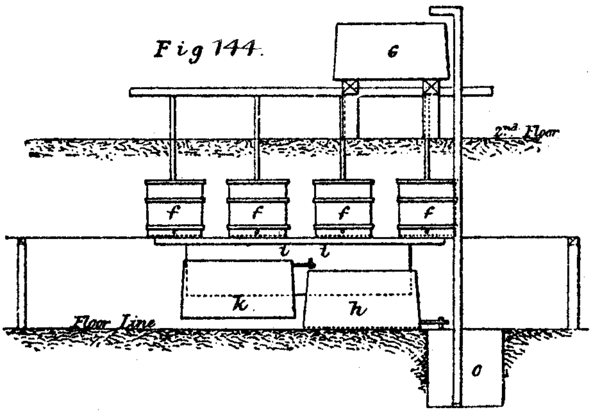


Fig 143.

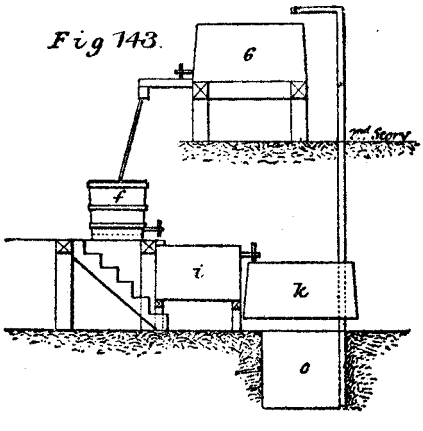
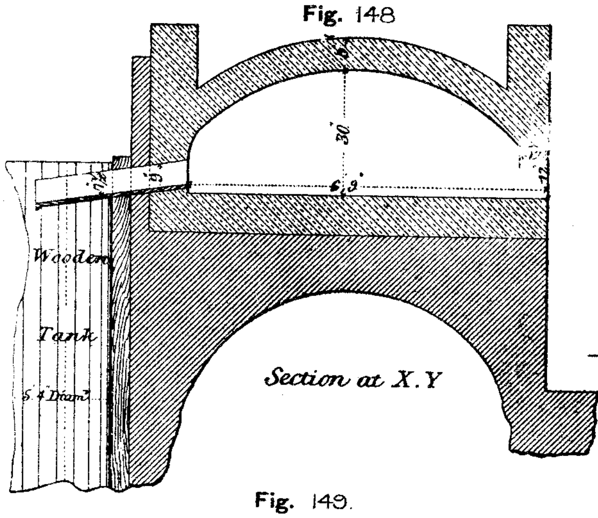
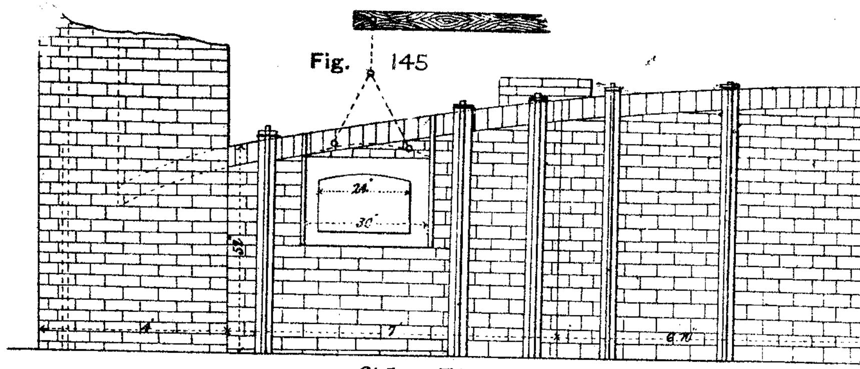


Fig. 148



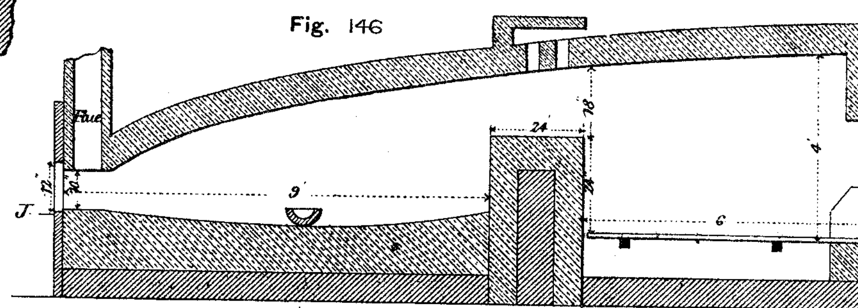
Section at X.Y

Fig. 145



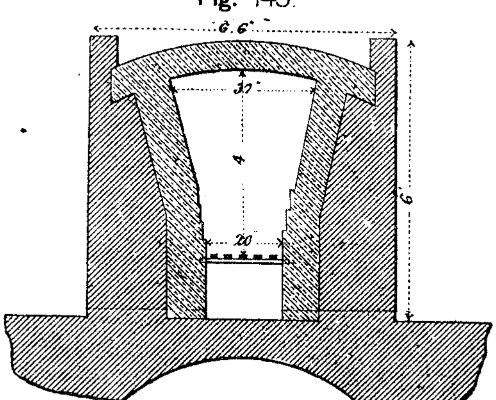
Side Elevation.

Fig. 146



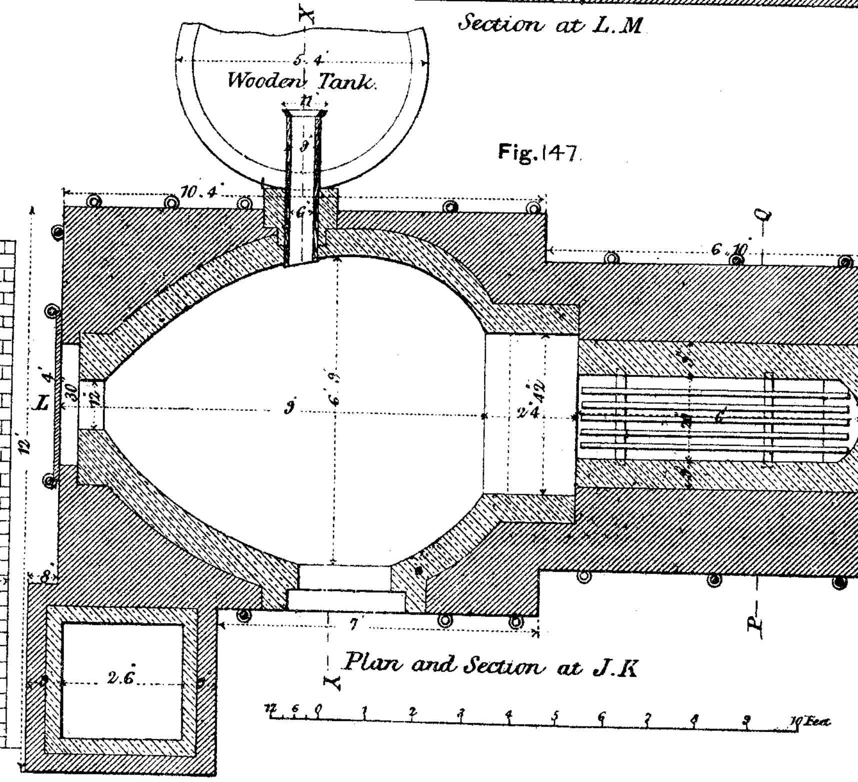
Section at L.M

Fig. 149.



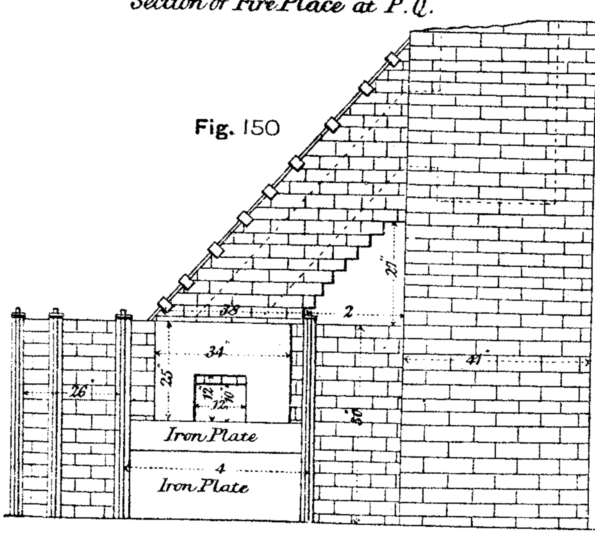
Section of Fire Place at P.Q.

Fig.147.



Plan and Section at J.K

Fig. 150



Zierrogel Process

deep. The fireplace is charged at once, and the temperature is made as hot as the red bricks will bear, and as oxidizing as possible. It is constantly rabbled. At the end of thirty-six hours a portion is taken out and tested to see that it pulverises completely. If it does, the operation is finished; if it does not, the oxidation is continued. The whole of the copper has been transformed by the operation into suboxide, and the charge is increased in weight by 500lb. The grains are black on the outside, but if broken or rubbed the streak is red. The charge is drawn out into an iron barrow, and carried to the store-room. It is placed in bags, packed in petroleum casks, and shipped to Boston. One cask holds 650lb. Three cords of firewood are used for the process, and two men do the work—one man to each twelve-hours shift.

The oxidized product is treated with dilute sulphuric acid. This is done in a conical tub lined with lead, with a false bottom, the bottom being hollowed so as to leave as little space as possible. A charge is 1,500lb. Over this sulphuric acid at 20° Baume is poured. Steam and air are turned on and the boiling continued for four hours. The whole is not dissolved, but 90 per cent. of the copper will be in solution. It is allowed to settle for an hour, and is siphoned off and a fresh charge put in. Two charges are made in a day. This is repeated until all the oxidized products have been treated. This work is not done at night. The residues are boiled two or three times in the same way, to get out all the copper possible. The tub is then cleaned up, and what remains is melted in plumbago crucibles. The bullion is from 600 to 800 fine of mixed metals. It contains from 40 to 50 per cent. of gold and 20 to 30 per cent. of silver. This is sent to the mint.

The sulphate of copper is crystallized and sold. The mother-liquor is used to dilute the acid used for the solution of the oxides.

The working of these alloys of gold, silver, and copper was first tried at the works, but was given up on account of the high price of sulphuric acid. It was carried on for more than a year in Boston, but has quite recently been abandoned. The separation of gold and silver is now to be done at the works by a process invented by Professor Pearce.

BRICK-MAKING MACHINERY.

Amongst the exhibits at the Melbourne Exhibition there were various classes of brick- and pipe-making machines which would be useful in New Zealand, and more especially in many parts where there is not good clay to be obtained for the manufacture of bricks. In England it is now proved that superior bricks and pipes can be made from the *débris* of a slate-quarry. In 1878 a patent was taken out in England for the manufacture of bricks from slate-*débris* by Mr. Thomas Evans, and since that time machinery has been perfected to such an extent that bricks now made of this material are far superior to those made from ordinary clay. Their adamantine character will, it is claimed, bring them into general use for tunelling and sewerage works, where strength, form, and imperviousness to wet are points of importance.

It is claimed that bricks made of slate-*débris* have the component parts most desirable in good bricks—namely, alumina and silica, with an entire absence of fluxibility; and from this material bricks can be made perfectly sound, free from flaws and cracks, and have the hardness to withstand fractures under great strains, while they can be made in any shape or size, and all perfectly regular and uniform. They also will resist the greatest extremes of heat and cold, and stand, it is said, a crushing-strain far in excess of any description of brick, having been proved up to 1,000 tons to the superficial foot.

Some of the brick-making machinery for manufacturing bricks out of this slate-*débris*, as well as from clay, shale, &c., were working at the Exhibition, and the bricks made by the semi-dry process and pressed were everything that could be desired, and by adopting this process there is no need of sun-drying or putting them in sheds: they are taken direct to the kiln.

The following is an estimate of the cost of manufacturing 65,000 bricks of slate-*débris*, as given by the patentee as the number that the machine will make a week, the price of labour being taken as the rate that would have to be paid in the colonies:—

	£	s.	d.
Two labourers supplying slate to grinding-mills, at 8s. per day ...	4	16	0
One boy attending water-spray	1	0	0
Two boys taking off bricks as finished, at £1 per week	2	0	0
Two boys running the bricks on trollies to kiln, at £1	2	0	0
Two men setting in kiln, at 8s.	4	16	0
Two men drawing the kiln, at 8s.	4	16	0
One engineer, if steam-power is applied, ...	3	10	0
One man firing the boiler, at 7s.	2	2	0
One foreman	3	10	0
One additional labourer, 8s.	2	8	0
Coal for boiler and fuel for burning bricks, oil, &c.	18	0	0
Total for 65,000 bricks	48	18	0

Or, say, the total cost was £50, this would make the cost of 1,000 bricks about 15s. 5d. In New Zealand, where water is plentiful, and water-power could be obtained, the cost would be considerably less. The greatest advantage in dry or semi-dry pressed bricks in a moist climate like New Zealand is that the bricks require no sun-drying before putting into the kiln to be burnt.

The following is a description of the machines required—namely, William Johnson, Castleton Foundry, Armley Road, Leeds, England; taken from "The Royal Album of Arts and Industries of Great Britain, 1887":—

Pulverising-and-sifting Mill.

We will first of all refer to the pulverising-and-sifting mill (Fig. 151). This consists of a revolving pan with two edge-runners, which latter are varied in weight from 20 cwt. to 60 cwt. each, according to the nature of the material to be dealt with. This mill is designed to crush marl, slate, shale, or other refractory material. This is tipped into the mill as it comes in its natural state from the quarry or waste-heaps, and, immediately coming in contact with the dead-weight of the edge-runners, is crushed. The bottom of the revolving pan in the outer surface is perforated, and as the raw material becomes crushed it passes through into a dish or saucer, in which are scrapers continually circulating and gathering the dust, and delivering it at one point, where it is caught by an elevator and conveyed to the

Patent Mixing-and-kneading Machine,

as illustrated by Fig. 152. This most thoroughly mixes and kneads the clay, and is designed to meet the requirements of those materials used for brick-making which require extra working, such as hard marls, shale, slate, &c., for either plastic or semi-plastic brick-making. The clay and the water are fed into the centre of the pan, where it forms a cone, the base of which is continually being rolled down into a cake, which is spread out and repeatedly cut up again by a series of ploughs, which are placed in convenient positions. The pan is 9ft. in diameter, and the clay, being delivered in the centre, must travel to the outermost edge, during which it is being continually operated upon by the four edge-runners and the ploughs, and is passed through perforations or slots in the bottom, and drops into a receiver in which are working scrapers bringing the clay and delivering it to the part required. The whole process is simple and automatic, and requires but little power, and attention only for regulating the supply of water.

Moulding-and-pressing Machine.

The brick-moulding-and-pressing machine (Fig. 153) represents a machine which makes two bricks at a time, and will produce 12,000 per day. Such a machine takes from four- to five-horse power to drive, and weighs about 4 tons. It is generally acknowledged to be the simplest and most effective ever introduced to the trade. It is designed for making semi-dry bricks ready for going direct to kiln, also concrete-bricks, cement-balls ready for burning into clinker, peat-blocks, patent fuel, seed-cake, &c. The moulds are changeable, so that various sizes and shapes of bricks can be made by the same machine by changing the moulds. The machine is fitted with a regulating apparatus, by which the thickness of the brick can be varied as the machine is at work. The pressing-plates and moulds are heated with steam, which renders unnecessary the use of oil for lubrication. This machine, it is stated by the maker, is the only one found to make Lyttle's patent compound into bricks, which some brick-machine makers pronounced impossible to be made. This is due to its simplicity of construction and its immense power and strength.

The mode of working is as follows: The clay or material to be formed into bricks is fed into a hopper in a loose and granulated state, at the base of which, working in a slide on the face of the table, is a bottomless box or charger, which receives the clay or other material from the hopper, and, sliding over the face of the moulds formed in table, fills them, and, to insure a certain and dense feed, the pressing-head drops with its own weight (3cwt.) upon the material while the charge is still over the moulds; the pressing-head is then raised to allow the charger to return again under the hopper to be refilled, and in its passage it strikes off the loose material level with the face of the moulds. The pressing-head then drops a second time with its own weight upon the material in the moulds, and during the time it remains there two additional distinct and powerful pressures are given, thus making the brick or article pressed dense and in perfect form. The presser then leaves the moulds, and the bricks are raised for delivery, which is performed by the charger returning again to refill the empty moulds, and delivers the finished bricks, &c., at the same time, ready for the attendant boy to remove on to a wagon or barrow to be taken direct to the kiln. Each brick receives four distinct pressures, thereby thoroughly discharging the air, whereas, it is said, no other machine of this class gives more than two pressures. The peculiar mode of charging the moulds insures a regular and even feed of clay, thereby enabling any kind of granulated clay to be fed, and also in a much damper state than is possible by other machines of this class. The machine is most simply arranged, and the wearing-parts easily renewable. The labour required is one man to tip the clay into the grinding-pan, and one boy to take the bricks off as the machine delivers them. The working of the machine is automatic and constant.

But for obtaining the best results from slate-*débris* or other coarse materials the bricks, after leaving the brickmaker, are fed into the patent lever steam brick-pressing or calcindering machine.

Brick-pressing or Calcindering Machine.

The peculiarity of Mr. Johnson's brick-pressing and calcindering machine (Fig. 154) is that the pressure is given by a powerful lever at $2\frac{1}{2}$ to 1 with a minimum of loss by friction, the whole power produced by the gearing and lever being brought directly upon the substance pressed. The motions also for feeding and delivery of the brick are direct and firm in their action, and can be regulated in a very simple manner, so that there is no varying, and are only affected by ordinary wear-and-tear, which is also provided for. This machine is equal to the finest work, such as the manufacture of white and coloured glazed bricks, which it is performing most successfully, while the small cost of its working makes it specially valuable for producing ordinary building-bricks. Weight of $3\frac{1}{2}$ tons, a 4in. belt only is required for driving.

The cost of these machines is about as follows, delivered f.o.b. in London or Liverpool:—

	Weight, Tons.	Price. £	s.	d.
Pulverising-and-sifting mill	14	210	0	0
Kneading-and-mixing machine	12	170	0	0
Brick-moulding and -pressing machine	$4\frac{1}{2}$	172	10	0
Calcindering-machine	$3\frac{1}{2}$	125	0	0

Some of the other brick-making machinery exhibited combined all the three processes—namely, grinding, pugging, and moulding, and were capable of turning out from 12,000 to 20,000 bricks per day. The price of these varied from £360 to £420. These machines could also make drain-pipes.

GENERAL REMARKS.

There was a variety of different classes of American wood-working machinery exhibited, and also a large assortment of agricultural implements in the German, English, and American Courts. To give a description of these would entail a great many illustrations in order to give a clear idea of their construction; and this class of machinery does not refer to mining, which this report is entirely confined to. It may be, however, of some interest to those engaged in agricultural pursuits to state that none of the agricultural implements at the Melbourne Exhibition surpassed those exhibited by New Zealand manufacturers. Indeed, many of the Australian colonists were surprised at the progress made in New Zealand in the manufacture of farm-implements, and greatly admired those exhibited by Messrs. Reid and Gray. There was a large collection of implements of this description exhibited in the German Court, but both the workmanship and their usefulness was far inferior to those exhibited of colonial manufacture, and many of them were of an antiquated stamp. The manufacturers of agricultural implements in the colonies have nothing to fear as regards competition with those coming from Germany, unless they improve greatly on their present manufacture.

The American machinery for wood-cutting was far superior to any other exhibit of this class at the Melbourne Exhibition. Its lightness, combined with strength, and no labour being wasted in its manufacture by polishing any parts where polish is not actually required, are the chief features of its construction, whilst the simplicity of construction in the different machines, their portability, and the large amount of work they are capable of doing, together with the true and accurate manner in which each machine performs its work, are deserving of notice. They are bound to command a market in the Australian Colonies, unless colonial manufacturers bestir themselves and produce similar machines of equally as good material and workmanship at a less price than American machines can be sold for. It is only by this means that colonial manufacturers can compete against the American article, as heretofore American tools of all descriptions for wood-working have been looked on by almost every one as being superior to those of English manufacture.

The exhibits of the H. B. Smith Machine Company, Smithville, New Jersey, were very numerous, and it may be of interest to quote the prices of some of their machines. A description of them could not be given unless there were illustrations showing their construction; and this report is getting too voluminous to do justice to machinery of this description. Their weight and price may, however, be of some use to those requiring wood-working machinery.

Improved Planing, Matching, and Moulding Machine, No. 3 Size.—This machine will plane two sides of a board 24in. wide and up to 6in. in thickness, tongue-and-groove up to 16½in. in width, and cut mouldings on four sides. All necessary shafts and gearing, belts, &c., included, the weight is 8,000lb., and the price is about £375; speed, 850 revolutions per minute.

Planing-and-matching Machine, No. 5 Size.—This machine will plane on one side and tongue-and-groove boards to any desired thickness or width up to 14in., and surface one side. With all the necessary belting, shafts, and gearing; the weight is 3,400lb., and the price about £125. Speed, 1,000 revolutions per minute.

The following is a list of machines, their weight and cost, &c., all necessary gearing being included.

Name of Machine.	Weight.	Price.	Average Speed per Minute.
	Lb.	£	Feet.
Improved heavy surfacing-machine, 16in. surface	2,300	56	750
" " 20in. "	2,300	68	..
" " 24in. " (double gear)	2,800	87	..
New surface planing-machine, up to 24in.	1,500	62	900 to 1,000
Endless-bed surfacing-machine, up to 28in.	4,000	125	850
Hand-planer, or joining-machine, up to 12in.	1,200	31	900
" " 16in.	1,400	37	..
" " 20in.	1,600	43	..
" " 24in.	2,000	50	..
Improved panel-raising machine, with self-feeder	600	34	600
10in. moulding-machine	4,250	175	1,000
9in. "	4,000	150	..
8in. " to work four sides	2,200	112	900
7in. " "	1,900	93	..
E. 6in. " "	900	56	..
G. 4in. " "	800	50	..
Improved double-end tenoning-machine, from ¼in. to 6½in. long	4,400	225	800
No. 1 car-tenoning machine—cuts tenons to 7½in. long	2,000	87	850
No. 2 " " " 6½in. "	1,200	60	900
No. 3 " " " 5¼in. "	800	51	..
Wheel-tenoning machine for oval and round tenons, 2½in. long	800	50	..
Car-mortising and -boring machine, capable of making a mortise 2½in. through 12in. timber	3,500	125	300
No. 2 Power mortiser—2in. mortise through 7in. timber	1,350	66	500
Foot mortising-machine	175	9	..
Band-sawing machine, 36in. wheel	1,500	37	350
Wabble sawing- or grooving-machine	475	25	1,000
Universal saw-bench, with 18in. saw	900	37	400
Spoke-and-handle machine, or lathe for irregular forms	2,100	100	..
Rod and dowel-pin machine—No. 2, from 1in. to 2in. diameter	800	41	500
" " " No. 3, from ¼in. to 1¼in. diameter	600	35	900
Sand-papering and universal vertical-boring machine	500	12	400 to 500

These are the prices quoted to be delivered at the railway-stations, but liberal discounts on all the prices mentioned are allowed for cash. There were several exhibitors of this class of machinery, amongst whom was Messrs. Fay and Co., of Cincinnati, whose machines are greatly used in the colonies, and highly spoken of.

With regard to the improvement in mining machinery in the Australian Colonies, they are ahead of New Zealand in concentrating and rock-boring machinery; but, as far as the reduction-plants are concerned, they are not superior to any of those in New Zealand. All the reduction-mills chiefly consist of stamping-batteries of the old type, and the mill-men and mine-managers are loth to try any reduction process beyond that they have been accustomed to use. The style of working, stoping, and timbering the lodes is similar to that in which our quartz-mines are worked. The only digression from this system is that in the Broken Hill Proprietary Company's mine at Broken Hill, in New South Wales, where the lode is 160ft. wide in places, and could not be stoped out on the same system as that adopted in the quartz-mines; and it also takes a rich lode to pay for the style of timbering adopted at Broken Hill.

When the present new reduction machinery and plants in course of construction for the treatment of ores in New Zealand are completed there will not be their equal in the Australian Colonies, and if further information with regard to the treatment of the refractory ores is required, America is the country where it is most likely to be obtained, inasmuch that it is by far the largest gold- and silver-producing country in the world, where there are various combinations of metals in the lodes in which gold and silver are found, and where there are very extensive works for reducing and treating the ores. The plant that has recently been erected at Waiorongomai by Mr. John Howell, a gentleman well known on the Pacific Slope of the United States, is likely to treat the refractory ores successfully. At all events, it is a much superior plant to any in the Australian Colonies for dealing with the class of ores found in the North Island of New Zealand. When the question is solved with regard to the proper treatment of ores, which must be by some inexpensive method, there is a bright future for New Zealand, as many lodes now untouched will be taken up and worked, giving profitable employment to a large population.

In concluding my remarks on mining in the Australian Colonies—after visiting many of the principal mining centres and seeing the different mining works—there is nothing to induce a miner to leave New Zealand to embark in mining in Australia. There are many men on the goldfields there who are barely making a livelihood; and to find employment in the mines is equally as difficult as in New Zealand. In all the mining centres there is a large population, and the day is past when the individual miner without a good deal of capital can carry on mining successfully. The ground is principally held by companies, and large sums have to be expended before any returns can be expected. There is a far better field in New Zealand for the individual miner, and the day is not now far distant when low-grade ores will be made to pay for working, and thereby open up a large field for labour. This, together with the healthy and invigorating climate New Zealand possesses, will attract the attention of those who have left their native land to make one of the Australasian Colonies their home.

I have, &c.,

HENRY A. GORDON, Assoc. M. Inst. C.E.,
Inspecting Engineer.

APPENDIX.

Professor ROBERTS-AUSTEN, F.R.S., to Sir FRANCIS BELL.

Smelting-works, Freiberg, Germany.

SIR,— Royal Mint, 22nd January, 1889.

Since we visited together, in company with Sir Saul Samuel, the smelting-works at Freiberg, I have read many of the more recent official papers relating to the mining and metallurgical industries of New Zealand, which have been submitted to both Houses of the General Assembly of that colony. I have also read various reports and handbooks bearing on the same subject, which contain statements of much interest and importance. I understood that your main object in visiting Freiberg was to ascertain how far the conditions which prevail at the great Saxon works correspond with those of New Zealand and New South Wales; and, in accordance with your request and that of Sir Saul Samuel, I append a report upon the processes conducted at Freiberg. It may be well, however, for me to add a few remarks bearing upon the subject generally.

Any one in the older countries familiar with mining and metallurgical practice who reads the reports to which I have just referred cannot fail to be impressed with the anxiety of the miners to possess trustworthy information as to improved methods and processes. On the other hand, it is impossible not to fully recognise the value of the strenuous and successful efforts which are being made by Professor Black and other technical officers of the New Zealand Government to meet the wants of the industrial population. There is evidently no lack of accurate metallurgical knowledge in the colony, but it would seem that the ablest men feel that the most pressing claims on their attention arise from the needs of the miners for more or less elementary technical instruction; and they cannot, therefore, devote themselves to the study and practical development of processes suited to the actual treatment of complex ores. For instance, in a report to the Minister of Mines dated May, 1886 (Paper C.-4B, pp. 15 and 16), Professor Black calls attention to the "great variety of valuable metals that are found in the Coromandel Peninsula," and to the "complexity of the ores containing the metals." He points out that "processes for treating such ores are well known theoretically in the colony, but what is needed is the presence of some one connected with the Mines Department who has a practical knowledge of the processes, and has seen them at work;" and he suggests that a qualified person should be sent "to the Pacific Slope to collect and bring to New Zealand the information that is so urgently required on the Coromandel Peninsula, as well as

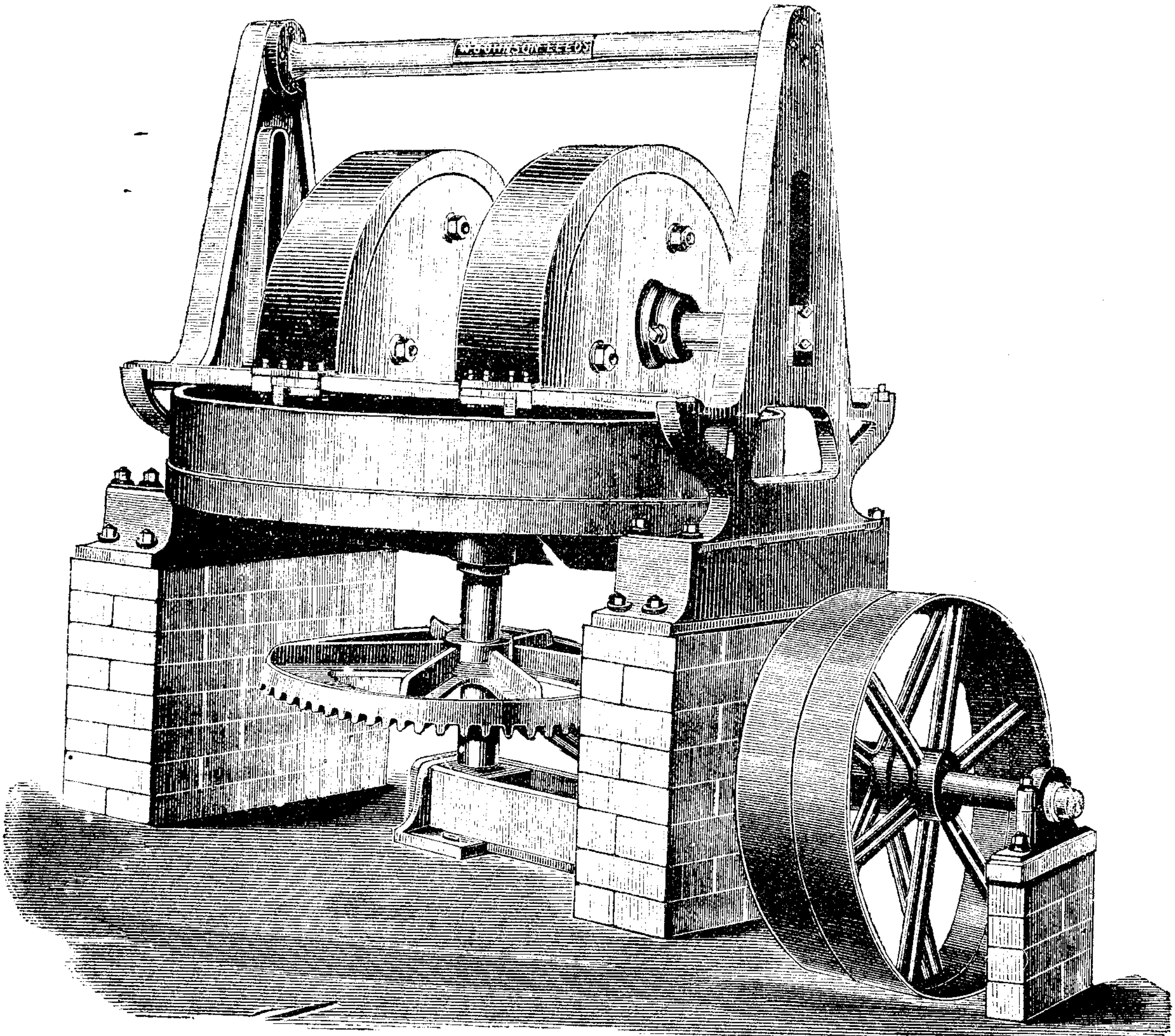


FIG. 151. PULVERISING AND SIFTING MACHINE.

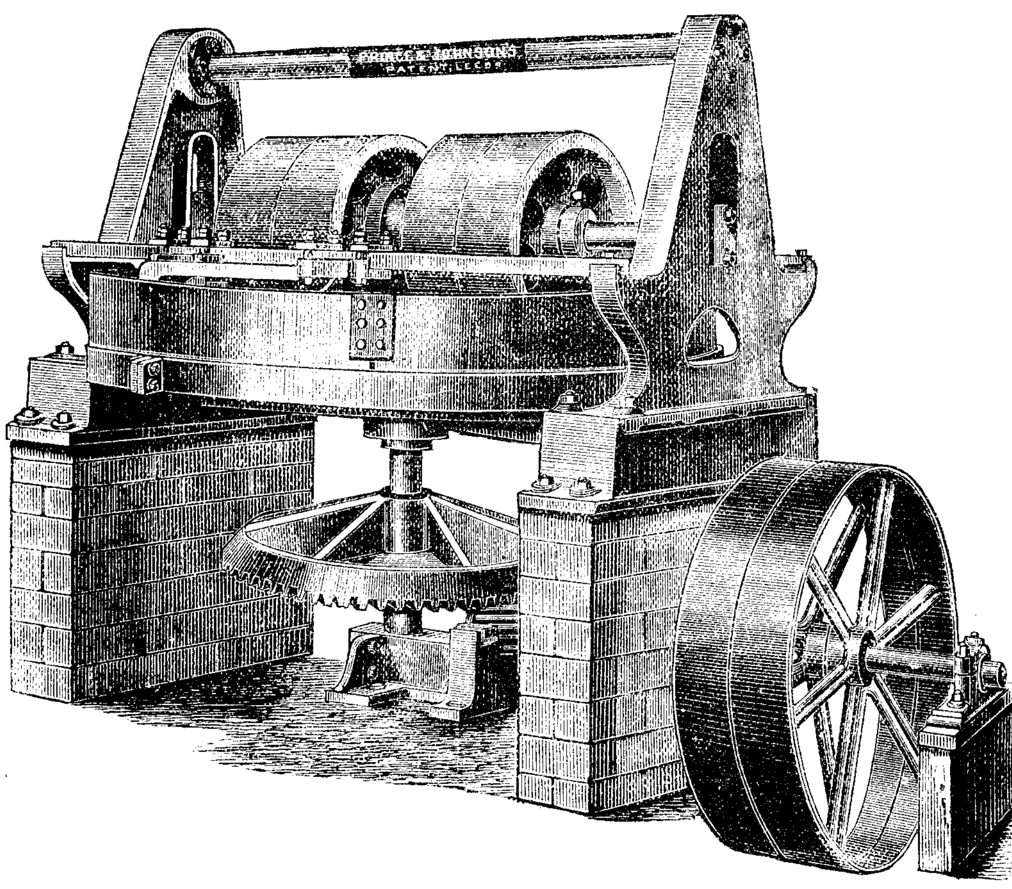


FIG. 152. PATENT MIXING AND KNEADING MACHINE.

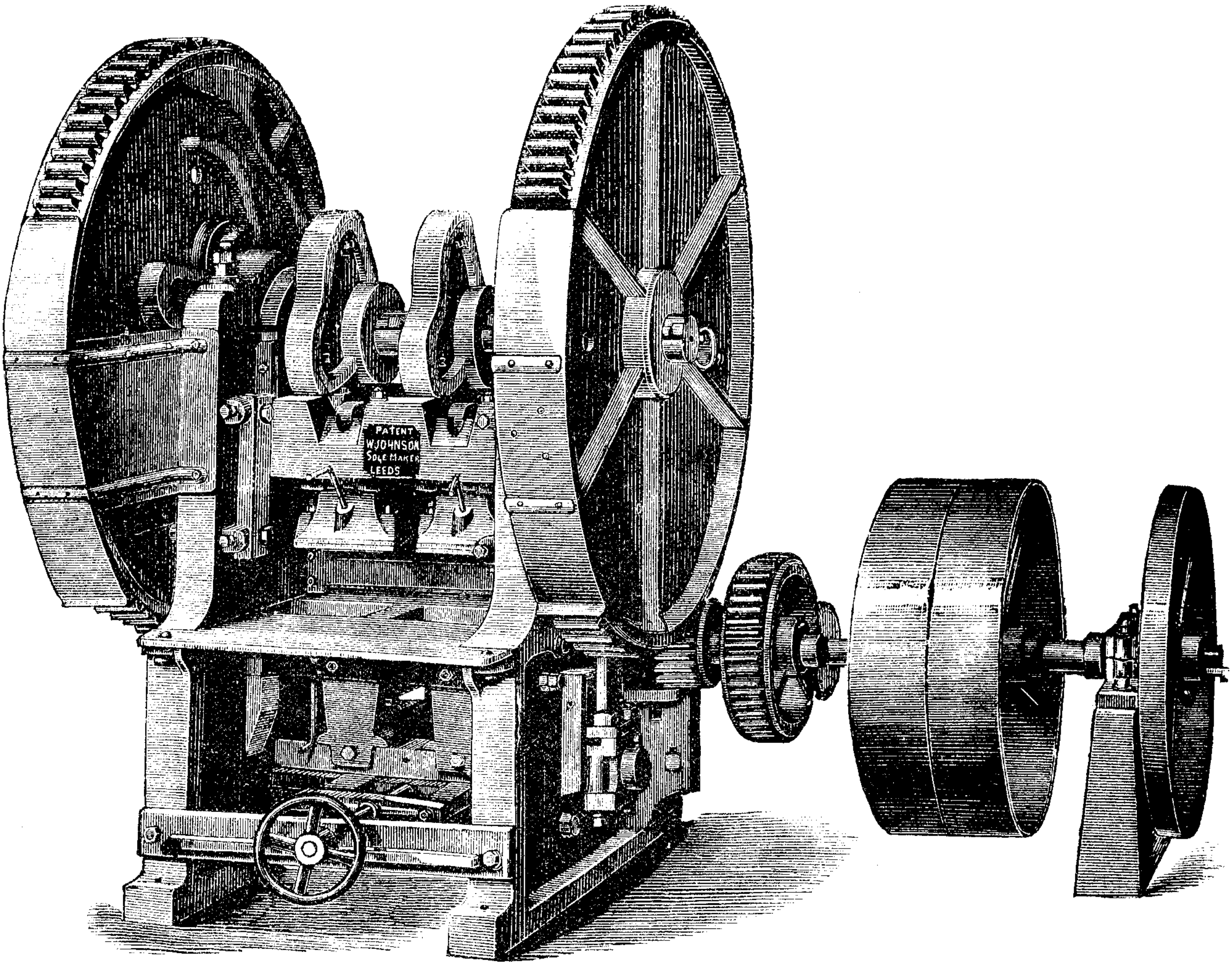


FIG. 153. MOULDING AND PRESSING MACHINE.

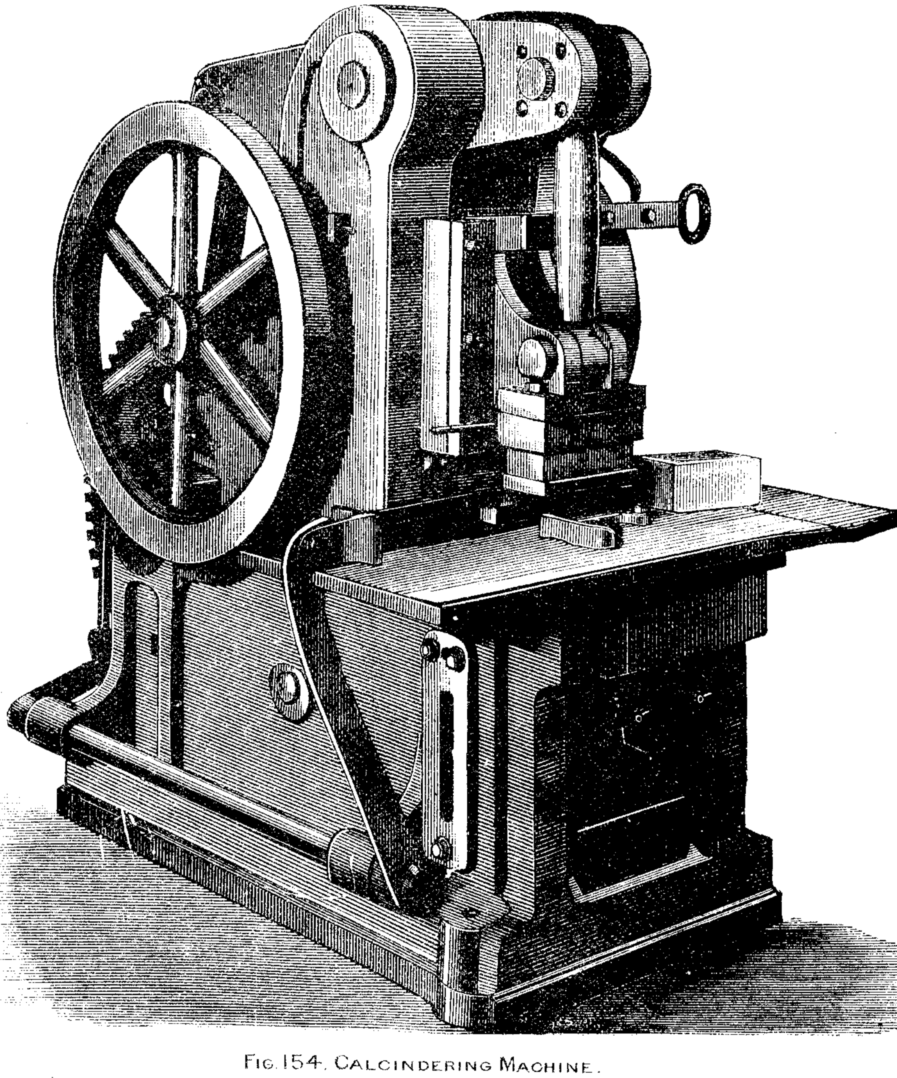


FIG. 154. CALCINDERING MACHINE.

at Collingwood and at other parts of the colony." Again, in a later report (Paper C.-5, 1888, p. 8) Mr. H. A. Gordon, Inspecting Engineer to the Department of Mines, observes that "it is not so much a want of capital as a want of knowledge how to deal with our silver-ores that is keeping the field back just now; for if people could satisfy themselves by actual trial of the efficacy of any process, they would find capital to put up all necessary machinery." He refers to the preference which is shown to "sending the ores all over the world, instead of putting up plant that may prove useless," pointing out that "help from abroad is costly."

It is impossible for me to say how far it may prove to be practicable to adopt a system of smelting complex ores in blast-furnaces, with a view to the concentration of the precious metals in lead, as is done at Freiberg. It may, however, be pointed out that what is called the "La Monte," process, which is based on the same principle as that adopted at Freiberg, does not appear to have been successful. On the other hand, a somewhat similar scheme, proposed by Mr. Parkes, for the treatment of the very complex ores of the Thames district, would appear to have given better results; but, with reference to this process, Mr. Gordon, in the report to which I have already referred, says that he is "very doubtful whether the smelting process will ever be carried on with New Zealand ores at a cheap rate, as the cost of obtaining the necessary fluxes will always make it an expensive process here."

I cannot offer suggestions as to the treatment of such ores until I obtain further information respecting the nature and relative abundance of the ores, the variety of fuel it is proposed to employ, the cost of transit, and the local conditions generally. I have only at present described the process of smelting complex ores as conducted at Freiberg, but I propose to send you later on some further observations pending the arrival of the particulars to which I have just referred. In the meanwhile I may say that I am of opinion that if lead-ores are sufficiently abundant it ought to be possible in the future, if not at the present time, to establish in New Zealand and New South Wales some large central smelting-establishment similar to the Muldenhütte at Freiberg, to which the complex auriferous and argentiferous ores of your colonies might be transmitted for treatment.

I should note that the diagrams (Figs. 1, 2, 3,) which are referred to in the report do not represent quite the latest form of appliances in use at Freiberg. The Pilz furnace (Fig. 1), for instance, is very important, but you will remember that during our visit I ordered a model representing the latest dimensions of the furnace, and this model, from which I intended to have had a drawing made, has not yet been received. This will be explained in my final report, with further diagrams.

I am, &c.,
W. C. Roberts-Austen.

Sir Francis Dillon Bell, K.C.M.G., C.B.

Freiberg Process.—Report by Professor Roberts-Austen, F.R.S., Assayer to the Royal Mint, and Professor of Metallurgy at the School of Mines, upon the Processes in Use at the Saxon Government Works at Freiberg.

From the various reports made by the direction of the New Zealand Government, as well as from the statements appearing in the New Zealand Press, it would appear that, besides being rich in free-milling auriferous quartz, New Zealand possesses considerable deposits of antimonides, tellurides, and other refractory ores, both of gold and silver, as well as complex sulphides containing, in addition to the precious metals, lead, copper, zinc, and iron.

Reference to the map accompanying the "Handbook of New Zealand Mines," 1887, will show how widely distributed are the ores, both of lead and of copper, which form one of the main sources of production of gold and silver.

With reference to the applicability of the Freiberg process to the treatment of the ores occurring in New Zealand, it may be pointed out that, in order to render such a process a commercial success, there must either exist a sufficient amount of suitable ores and fuel within a reasonable distance of the works, as is the case at Freiberg, or the ores must be sufficiently rich and abundant—and, further, the means of transport must be sufficiently cheap—to counterbalance the expense of carrying ore from mines distant from the smelting-works. The network of railways in the United States has led to the system of sending ores to "custom" works, which works are situated in the centre of mineral districts, and obtain their supplies of fuel by railway-carriage at reasonable rates. Such "custom" works appear to have been erected at several places in New Zealand, one example being that given in the "Report on the Mining Industry of New Zealand," 1887: they do not, however, seem to have proved commercially successful. The La Monte process, employed at the works referred to, did not essentially differ from the cupola treatment of argentiferous and plumbiferous materials in use at Freiberg. Ores suitable for such treatment exist, it is true, in considerable quantities; but the districts in New Zealand where all the materials required for the furnace-charge can be obtained in sufficient quantities and at reasonable rates appear at present to be but few, the country being still comparatively wanting in the means of transport which render "custom" works profitable elsewhere.

As the means of comunication become extended, it would appear to be probable that ores from all parts of the colony will be treated at a few large works suitably situated. It may be well, therefore, to consider the process adopted at the Freiberg smelting-works, which receive from all parts of the world argentiferous ores, the same being paid for in accordance with a certain fixed tariff.

The Process of Smelting at Freiberg.

The smelting-works were originally established at Freiberg by the Saxon Government, to treat the ores obtained from the mines of the district. There are two works, of which the more important, the Muldenhütte, is situated about three miles from the town of Freiberg; the other, at Halsbrücke, being somewhat more distant. The Muldenhütte is situated on the slope of a hill just beneath the railway connecting Freiberg with Dresden. On the summit of this hill is the main stack, and also a reservoir, the water from which is available as one of the main sources of power for driving and hoisting at the works. Power is obtained from the river Mulde, which flows at the foot of the hill, and steam is also employed.

The various operations conducted in these works have often been described, as will be seen by the list of papers noted below.* An excellent account was published in 1880, in English, but since that time several important changes have been introduced into the works. A brief introduction is here given, showing the general nature of the scheme and processes adopted at Freiberg, fuller details being supplied in the respective sections into which the description is divided.

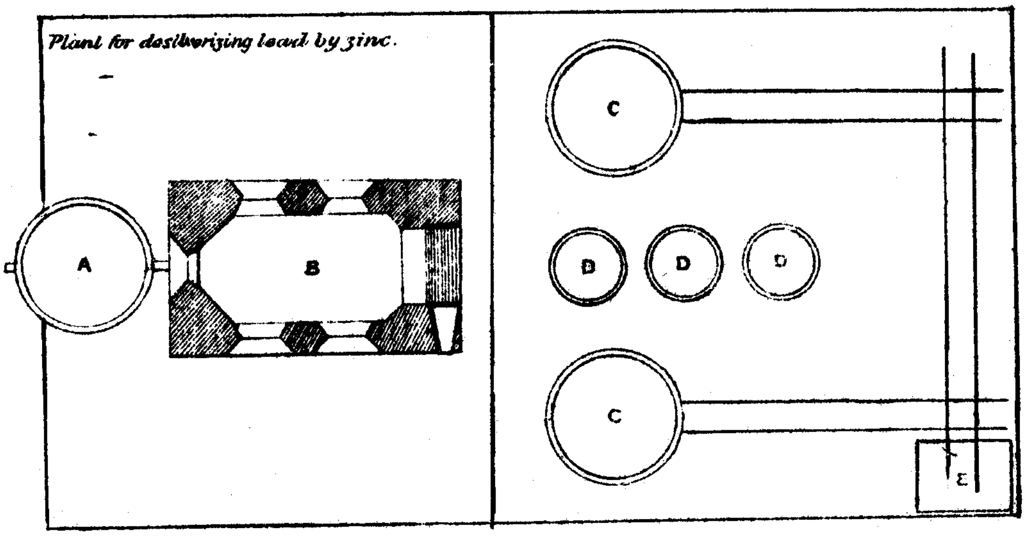
The ores treated consist chiefly of the sulphides of lead; but silver and copper are always present in the mixture of ores which constitutes the furnace-charge. The plant also includes furnaces for the treatment of ores of zinc, arsenic and antimonial fume. As an incidental process, sulphuric acid is made both by the ordinary lead-chamber method, and by a patented and secret process, in which the gases containing the sulphurous anhydride are stated to be passed over perforated clay slabs coated with platinised asbestos. Bismuth is also extracted from the portions of the cupellation hearths in which the greater part of the bismuth collects that was originally contained in the lead treated. Provision is also made for the extraction of copper, nickel, and cobalt present in the ores. The relation borne by the respective sections to each other and to the general plan of operations is indicated in the diagrammatic scheme which accompanies this paper. The various departments of the works are kept distinct, and the treatment as a whole centres around the smelting of the lead-ores, the various residues from the treatment of other ores, frequently rich in the precious metals, being added to the lead-smelting charge. It is in this latter treatment that the distinctive features of the Freiberg smelting-process are found, the lead-ores being smelted in admixture with the copper-ores and argentiferous and auriferous residues. The furnaces used for this purpose are water-jacketed cupolas (Fig. 1). They were introduced originally about the year 1865, to replace the older types of blast-furnace which had long been employed. The extraction of silver by the classical Freiberg amalgamation process, conducted in barrels, was also abandoned about this period, and was replaced by the method still in use, by which the precious metals are collected in the smelted lead. The main features of this process are as follows: In order to impart the necessary degree of strength and coherence to enable them to be treated in the blast-furnace, as well as to obtain them in pieces of a size adapted to cupola treatment, they are first roasted at a temperature sufficient to clot the mass when roasted. This is effected by subjecting a suitable mixture of ores to an oxidizing roasting in a long-bedded reverberatory furnace. The sulphur of the ore is in this way oxidized, and the metals are left chiefly in the form of oxides. Towards the end of the roasting, the temperature of the furnace is raised to a sufficient degree to partly fuse the oxidised charge then present; the roasted material can in this state be withdrawn from the furnace into sheet-iron wheelbarrows, in which it is allowed to solidify. The solidified material is afterwards broken up into pieces of the desired size, and classified according to the completeness with which the roasting has been effected, as indicated by the presence or absence of unroasted lead-sulphide. It is then smelted down in the cupolas, in admixture with coke and brown coal. Large quantities of iron are present in the material charged into the furnace, and cold blast has therefore to be employed to prevent too great a reduction of that metal and the consequent formation of unfused masses of reduced iron in the furnace. The products of this smelting are (1) lead, (2) a "regulus" or mixture of fused sulphides of lead and copper, (3) a slag, and (4) a lead fume. A small quantity of speisse, containing nickel and cobalt, is also occasionally obtained. The lead produced contains the greater part of the precious metals originally present in the charge, together with some copper and other impurities. Such lead has always to be subjected to a refining process before being desilverised by the method in use at the works, and, if very impure, a secondary refining process, termed "liquation," has also to be used. To avoid this the charge is usually maintained as free from impurities as possible, the percentage of the copper being at the same time kept low. The slag from the ore-smelting always contains lead in too large a quantity to admit of its being thrown away. It has therefore to be re-treated, and, as the lead obtained from this source is always very impure, any impure materials that it may be necessary to deal with are generally treated simultaneously with the slag, provided they are not too rich in the precious metals. The percentage of copper admissible in the furnace-charge for this slag-smelting is also greater than in that for the treatment of the ore. The ore-slag is treated in admixture with materials poor in silver and in lead, in a similar manner to the method adopted when smelting the ore. The products are the same as before—lead, speisse, regulus, slag, and fume. The lead, however, is much less in relative quantity and more impure than that obtained from the ore-smelting; speisse is a more frequent product; the regulus obtained is richer in copper, and the slag is so poor in silver and lead that it may be thrown away.

The further treatment of these various products is as follows:—

Lead.—The slag-lead, and occasionally the ore-lead, is first submmitted to a liquation process, which consists in melting the lead, at as low a temperature as possible, on the sloping bed of an ordinary reverberatory furnace in a deoxidizing atmosphere. By this means the lead melts and flows into a sump or hollow at the foot of the sloping bed of the furnace, leaving the greater portion of the copper it originally contained, together with iron and other impurities, the melting-points of which exceed that of lead, on the furnace-bed. Nearly all the silver present in the lead charged into the furnace, and with it the gold, pass into the liquated product, owing to the low fusion-points of the silver-

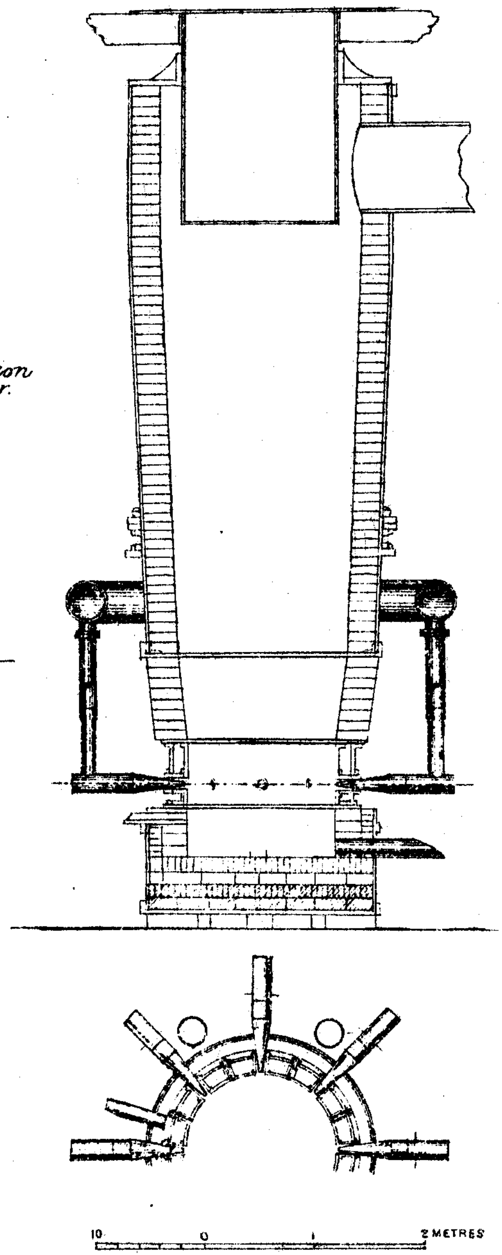
* Bergm. Verein in Freiberg; Freibergs Berg- und Hüttenwesen: Freiberg, 1883. Festschrift; Zum Hundertjähr. Jubiläum der k. Bergakademie zu Freiberg: Dresden, 1866. Die Fortschritte der Berg- und hüttenmännischen Wissenschaften: Freiberg, 1867. Beust; Das Freiberger Berg- und Hüttenwesen vor 100 Jahren and jetzt: Dresden, 1866. Jahrbuch für das Berg- und Hüttenwesen im Königreiche Sachsen, 1881, &c.: Freiberg. Percy—Lead, p. 203; Silver and Gold, p. 543.

FIG. 2.



A. Tapping pot. B. refining or improving furnace. C. desilverizing pots. D. liquating pot. E. hoist.

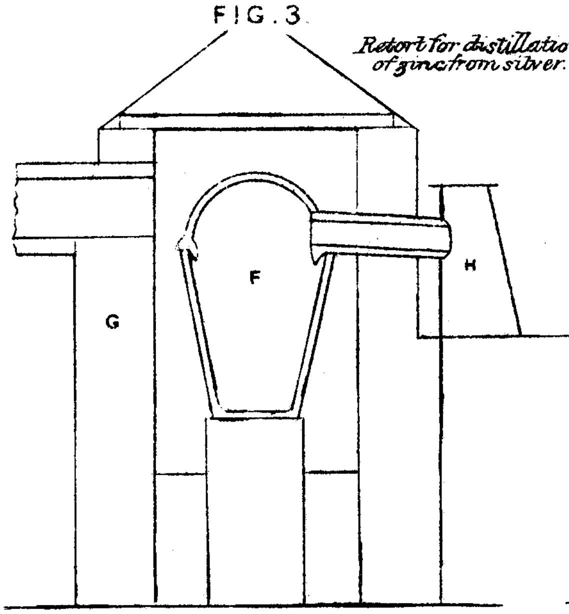
FIG. 1



PILZ FURNACE

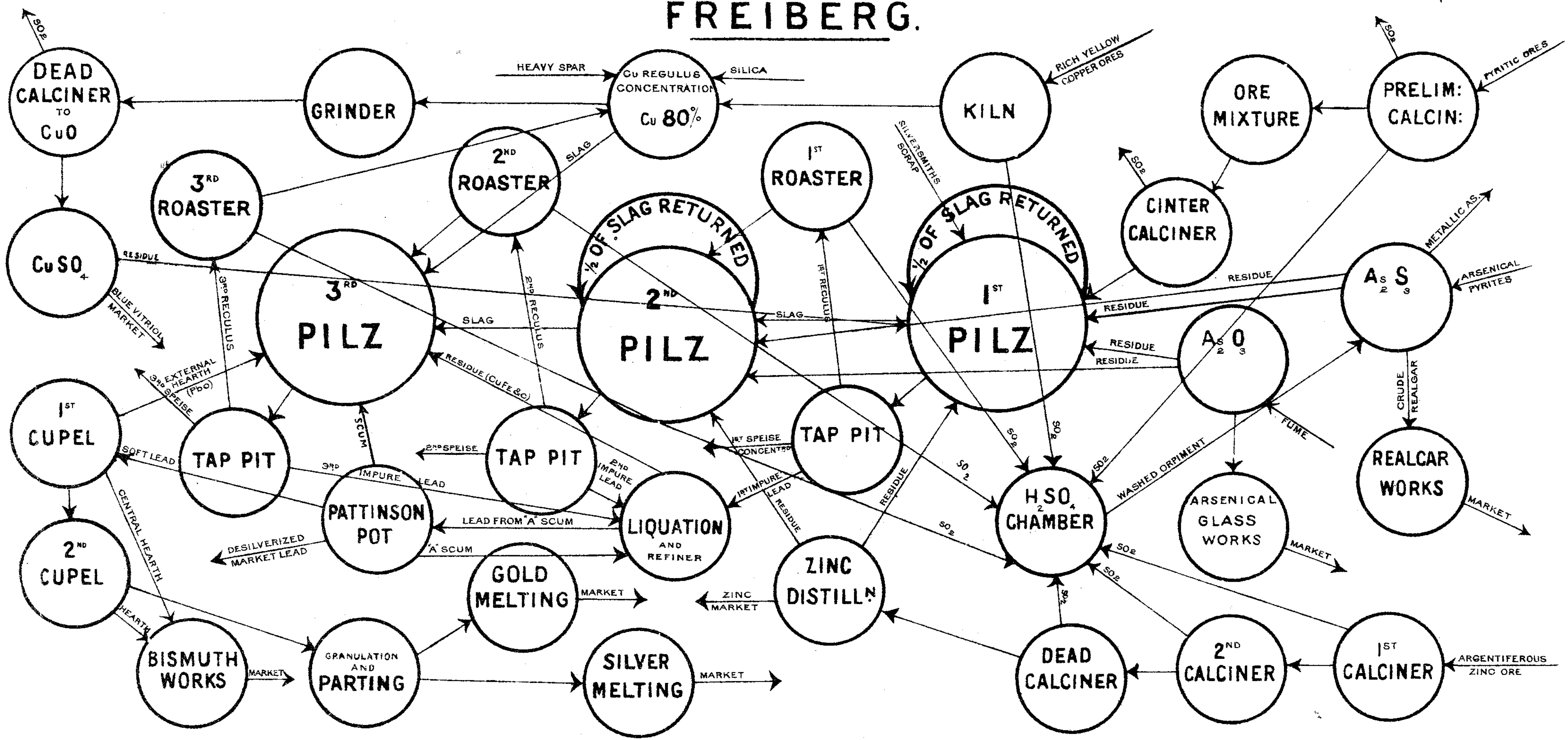
FIG. 3

Retort for distillation of zinc from silver.



F. graphite crucible. G. furnace. H. condenser.

FREIBERG.



DEAD CALCINER TO CuO
GRINDER
Cu REGULUS CONCENTRATION Cu 80%
HEAVY SPAR
SILICA
KILN
RICH YELLOW COPPER ORES
ORE MIXTURE
PRELIM: CALCIN:
PYRITIC ORES
CINTER CALCINER
2ND ROASTER
3RD ROASTER
1ST ROASTER
SILVERSMITHS SCRAP
½ OF SLAG RETURNED
3RD PILZ
2ND PILZ
1ST PILZ
CuSO4
BLUE VITRIOL MARKET
METALLIC AS.
As2S3
ARSENICAL PYRITES
As2O3
FLUME
CRUDE REALGAR
REALCAR WORKS
MARKET
RESIDUE
SLAG
1ST REGULUS
2ND REGULUS
3RD REGULUS
RESIDUE (CuFe &c)
EXTERNAL HEARTH (PbO)
3RD SPEISE
1ST CUPEL
2ND CUPEL
TAP PIT
SCUM
2ND SPEISE
1ST SPEISE CONCENTRD
SOFT LEAD
3RD IMPURE LEAD
2ND IMPURE LEAD
1ST IMPURE LEAD
PATTINSON POT
LIQUATION AND REFINER
LEAD FROM "A" SCUM
"A" SCUM
DESILVERIZED MARKET LEAD
GOLD MELTING
SILVER MELTING
CENTRAL HEARTH
HEARTH
BISMUTH WORKS
GRANULATION AND PARTING
ZINC DISTILLN.
ZINC MARKET
H2SO4 CHAMBER
WASHED ORPIMENT
ARSENICAL GLASS WORKS
DEAD CALCINER
2ND CALCINER
1ST CALCINER
ARGENTIFEROUS ZINC ORE
SO2

lead and gold-lead alloys there present. This liquated lead, together with the ore-lead, is then submitted to an oxidizing fusion in a reverberatory furnace, the result being that the arsenic, antimony, tin, and other impurities present, which had failed to be eliminated by the liquation process, are oxidized away, and, a portion of the lead being simultaneously oxidized and fused, they pass into this oxide scum, and can be removed with it. Owing to the varying degree of oxidizability of the several impurities, the fused litharge obtained at different stages of this process contains relatively larger or smaller quantities of the various impurities. These litharges, after removal from the furnace, are kept separate, and are known by the name of the more important impurity they contain. Each of these is afterwards reduced by itself in a small blast-furnace, the product being in each case a readily-saleable hard lead, the impure litharge being first fused with a small percentage of carbon to reduce a portion of the lead, and in this way to collect any silver that may be present in the reduced lead formed. Any copper, too, that may have passed into the liquated lead is also oxidized away, but the bismuth remains with the lead. Small quantities of the precious metals always pass into the fused-oxide layer on the surface of the lead, but by far the larger proportion of that originally charged into the furnace remains with the purified lead. This lead is then treated by the ordinary Pattinson process, the lead being first melted at a low temperature and then allowed to cool gradually, the lead crystals poor in silver being separated from the richer molten portion. This treatment is repeated in the ordinary manner until the lead contains 0·1 per cent. of silver. Instead of continuing to treat this lead by the Patinson process, the Parkes process (Fig. 2) of desilverisation by the aid of zinc is employed for the purpose of extracting the remaining silver and gold, the lead from this treatment being dezincified by an oxidizing fusion, after which it is ladled into moulds and is ready for the market. The rich Pattinson lead is cupelled on cupellation-hearths of the German type, with non-absorbent marl beds. The cupellation is so conducted that the lead, being charged into a hearth of large size, is oxidized away until the residual material contains about 80 per cent. of the precious metals. This requires a higher temperature for its further treatment, and is therefore removed from the large furnace to a similar but much smaller one, in which the remaining portion of the lead is eliminated, the fused gold and silver being granulated by pouring into water, the granules dried, and the gold and silver parted by the aid of sulphuric acid.

The litharge, if of a yellow colour, is reduced to the metallic form by a reducing-fusion in a small cupola, but any red litharge that is produced is sold as such. The beds of both the large and the small cupellation-furnaces show green spots at the places where the final products, rich in gold and silver, collected: the bismuth, not being removed by oxidation until nearly all the lead has been oxidized, passes into these portions of the marl beds, colouring them green. These green patches are carefully removed, dissolved in hydrochloric acid, and bismuth oxychloride precipitated by dilution with water. This is either sold as such after purification, or else is reduced to the metallic state by fusion in crucibles with iron. The other portions of the marl cupellation-beds, being rich in lead, are on this account added to the various smelting-charges.

The zinc rich in silver and gold obtained by the Parkes process is distilled in the ordinary Morgan furnace (Fig. 3). This completes the lead-smelting process proper, both the gold and silver present in the original materials treated having been collected, and the desilverised lead obtained in a form in which it is ready for the market.

The *cupriferous regulus* resulting from the ore-smelting contains usually but a few per cent. of copper, and consists chiefly of the sulphides of iron and lead. It may be mentioned that it is necessary to have considerable quantities of iron present in the cupola-charge, partly on account of the desulphurising action of the reduced metal, and partly because considerable quantities of zinc are usually present, and the zinc-oxide passing into the slag would render it exceedingly pasty and difficult to fuse were it not for the counteracting influence of large quantities of ferrous oxide.

This regulus, if its composition is such that this treatment is admissible, is roasted in kilns, the sulphurous anhydride produced being utilised in the manufacture of sulphuric acid. When roasted the regulus is added to the slag-smelting charge; the lead-oxide contained in the roasted regulus is then, for the greater part, reduced to the metallic state and a second regulus is produced, which is poorer in lead but richer in copper than was the one resulting from the ore-smelting. This regulus is too rich in copper to admit of its being roasted in kilns, the tendency for the pieces of regulus to clot together during the roasting being too great; it is therefore roasted in "stalls" instead. These stalls resemble, as their name implies, ordinary cattle-stalls; they are roofless, with low surrounding walls and a slightly-sloping bed. On this bed wood is placed, the regulus to be roasted being piled up on it, and then covered over with a compact layer of finely-divided roasted pyrites or regulus. The stalls are placed in rows side by side and back to back, a tunnel being left between every two such rows. Into this tunnel the gases resulting from the roasting are drawn through perforations in the back walls of the stalls, and are led away to the sulphuric-acid chambers. The combustion is started by lighting a small fire on a grate outside the stall; this kindles the wood, the heat evolved by the combustion of the sulphur in the regulus being afterwards sufficient to continue the process without the addition of any other fuel. This roasted second regulus is then treated as before, until a regulus is produced which contains about 35 per cent. of copper.

Instead of this repeated roasting and treatment with the slag-charge, the regulus, after having been roasted, may be treated in a cupola together with lead-slags and other fluxing and reducing additions. The product is a comparatively rich regulus, the lead originally present being reduced to the metallic state, taking with it the silver, the enriched regulus retaining but little of the precious metals. When enriched to the degree mentioned above, the regulus is brought by a single fusion in an ordinary Welsh reverberatory furnace to a "white" or "purple" metal containing about 75 to 80 per cent. of copper. This is not further treated at the Muldenhütte, but is sent to Halsbrücke Works, where it is roasted sweet, and the copper converted into copper-sulphate by treating the roasted material with sulphuric acid. Any lead and gold that may have been present remain undissolved, as also does the greater portion of the silver, the small quantity that passes into solution being reprecipitated by metallic copper. The undissolved lead residue is added to the blast-furnace treatment.

Speisse.—This is a comparatively rare product. The nickel and cobalt it contains are concentrated by a process resembling that for the concentration of the copper in the regulus, the speisse being first roasted and then remelted with fluxes, such as lead-slags, and with reducing agents in a small cupola. Metallic lead is obtained, which contains the greater portion of the precious metals originally present in the speisse, and this latter is much richer in nickel and cobalt after this treatment than it was before. In this concentrated state it is sold to outside works, where it is treated by the wet process, the residues, which contain the gold and silver of the speisse, being bought back by the Government works.

The *ore-slag* is treated in the manner that has been described, the resulting slag, poor in lead, being thrown away. This contains about 2½ per cent. of lead and 0·0045 per cent. of silver, the zinc-oxide also present occasionally reaching some 20 per cent.

The *fume* contains large quantities of lead-oxide, together with zinc-oxide and arsenious anhydride. It is first subjected to a sublimation treatment for the separation of the arsenious anhydride, which is collected and sold after a further sublimation. The oxides of lead and zinc remain on the bed of the furnace, and are added to the cupola-charge.

In detail the general scheme of treatment is as follows: The ores are mainly those of lead, copper, zinc, and silver. The products are lead, zinc, bismuth, silver, gold, arsenic, as metals, and copper, mainly recovered as sulphate. Besides these there are also incidental products, such as sulphate of iron and sulphate of manganese.

Nickel and cobalt are recovered as arsenides—that is, in the form of speisse; and the arsenic is recovered partly as metal, but mainly as powdery arsenious anhydride, and as red, yellow, and white arsenical glass. The ores come partly from surrounding mines and partly from foreign sources, the latter being mainly worked for the copper, silver, and gold they contain. Almost all the ores are argentiferous. The principal portion of each charge consists of lead-ores, and these are divided into galenas with 30 per cent. or more of lead, and poor lead-ores with 15 to 20 per cent. In the mean they contain generally 40 per cent. of lead and 0·015 per cent. of silver. The following, however, is a general statement respecting the ores:—

	Lead.	Silver.
Galenas (1)	79 per cent.	0·011 per cent.
(2)	74 "	0·012 "
(3)	65 "	0·010 "

and an ore richer in lead, 80 per cent., but poorer in silver, only 0·008 per cent.

	Lead.	Silver.	
Poor lead-ores	26 per cent.	0·06 per cent.	
	27 "	0·040 "	
Quartzose ore	...	0·026 "	
		0·05 "	Sulphur.
Quartzose and pyritic ...	... ...	0·010 "	16 per cent.
		0·015 "	13 "
Pyritic ores	... ...	0·540 "	20 "
Sulphuretted ores	... ...	0·600 "	31 "
		0·240 "	30 "
		0·035 "	35 "
	Lead.		
Lead-ores with arsenic ...	23 per cent.	0·450 "	Arsenic.
	39 "	0·100 "	16 per cent.
	Zinc.		
Zinc-blende	38 per cent.	0·044 "	
	40 "	0·020 "	

The lead in the ore is only paid for when it reaches 15 per cent., and they pay for every 5 per cent. above 20 per cent.—that is, 25, 30, 35, and so on. The ores in which only the silver or gold is of a value sufficient to cause them to be paid for are called "dürrerze," and the class is subdivided into—pyritic, 20 to 40 per cent. sulphur; quartzose, 10 to 19 per cent. sulphur; spathose, 0 to 9 per cent. sulphur.

The sulphur is paid for when it exceeds 24 per cent., and every additional 5 per cent. is paid for. Each per cent. of copper is paid for, but fractions of 1 per cent. are not taken into account.

The zinc-ores must not contain more than 5 per cent. of lead, and in ores that are not confessedly zinc-ores, only 3 to 4 per cent. of zinc must be present.

The copper-ores contain from 1 to 15 per cent. of copper, arsenical ores from 10 per cent. arsenic, zinc-ores from 30 to 40 per cent. zinc, sulphur-ores from 25 per cent. sulphur.

The arsenic in an ore is paid for when it reaches 10 per cent., and fractions of 5 per cent. are accounted for.

Gold is paid for from $\frac{5}{10000}$ and upwards, and silver $\frac{4}{1000}$ and upwards.

The weighing takes place in the presence of representatives of the mine and of the works. About 5cwt. is weighed at a time. The accuracy of the weighing depends on the richness of the ore: 0·01 to 0·05 per cent. silver, accurate to 10lb.; 0·50 to 5 per cent. silver, accurate to $\frac{1}{10}$lb.; 5 per cent. and upwards, accurate to $\frac{2}{100}$lb. Ores rich in gold are always weighed to $\frac{2}{100}$lb.

Sampling.—A small scoopful is taken from every 2 cwt., and is thrown into two or more wooden dishes, according to the amount weighed off. The ore is then put into the ore-house, and a board is inserted in the ore-heap showing the character of the ore, its quality, &c. The moisture is determined by heating 75 grammes in a copper shovel; the calculation of the moisture is accurate to ½ per cent. The mixing takes place in the ore-house. The various ores are spread out in layers, the thickness of which depends on the nature of the material, layer over layer. Then a definite portion is cut off one end of the heap, and the ore so removed is thrown up into a heap, which is then considered to be sufficiently mixed.

Roasting.—A charge for roasting and smelting may contain 2,500cwt. roasted material from the Gerstenhöfer furnaces, 600cwt. roasted residues from kilns and arsenic furnaces, 3,000cwt. raw ore, 200cwt. residues from zinc-smelting.

Heaps of from 6,000cwt. to 7,000cwt. supply four roasting-furnaces for a week and a half; one furnace will treat 180cwt. a day. The roasting-furnace is a single-bedded reverberatory, upwards of 40ft. long and about 10ft. broad; it has eleven doors on each side, and one which is used for roasting pure galena has fifteen doors on each side. They are worked from both sides at the same time. The fire-bridge is hollow, and the bed consists of a layer of Chamotte bricks, made from two parts of raw and one part of burnt clay; they contain 60 to 70 per cent. of silica. These bricks rest on common bricks supported by an iron plate, resting in its turn on pillars. If the hearth or fire-bridge becomes worn, they do not stop working, but with a long iron spoon insert a ball consisting of one part of clay and two parts of poor quartz ore. This is beaten down so as to renovate any defective place.

Each charge of ore consists of 34cwt., and there are five such charges in different parts of the furnace-bed at a time. The charge near the fire-bridge is allowed to clot, and is then removed. The other charges on the furnace-bed are then advanced towards the fire-bridge the distance of two doors at a time. The fireplace is divided in the centre, so that really there are two fireplaces or grates to each furnace, as the breadth is too great to permit one fireman to stoke the whole.

These Fortschaufelungsöfen are in connection with long brick chambers for the condensation of the fume, the collection of which takes place every six months, when some 2,700cwt. to 3,000cwt. of material is collected, containing—0·01 to 0·02 per cent. silver, 10 to 28 per cent. lead, 40 to 50 per cent. arsenious anhydride. The sulphurous anhydride in these flues cannot be used, as the gases are far too dilute and impure.

The charge for roasting generally consists of—galena, 30 to 40 per cent.; poor lead-ore, 20 to 30 per cent.; poor quartzose ore, 10 to 15 per cent.; residues, 5 to 10 per cent.

The cost of roasting each hundredweight of ore is as follows: Fuel, 8pf.; transport, 6pf.; wages, 20pf.; carriage of coke, 0·25pf.; wear-and-tear, 0·8pf.; tools, 3·9pf.: total, 38·95pf., or about $4\frac{3}{4}$d.

The roasted ore still contains from 5 to 7 per cent. of sulphur when the charge is withdrawn from the furnace into barrows of sheet-iron. The semi-fluid roasted charge is tipped out when solid, and broken into pieces about the size of the fist, and sorted by sight into (1) well roasted, (2) ordinary, and (3) badly roasted, according to whether much or little undecomposed galena is seen to be present. The men are paid accordingly.

At the Muldenhütte there are several large Pilz cupolas, each with eight tuyeres, and one small one with four tuyeres. The following is a charge for the ore-smelting: Ore, 450cwt.; pyritic ore, 80cwt.; roasted residues, 40cwt.; slag, 550cwt.; quartzose ore, 20cwt.: total, 1,140cwt. This gives 88cwt. of work-lead, containing 0·4 to 1 per cent. of copper, and 20cwt. of regulus, and very often a speisse forms as well. In charging, the coke is thrown towards the centre, and the larger pieces of ore to the sides.

If a "bear" forms, it is usually a ferruginous one; but it may contain much zinc-sulphide, and it generally forms at the top of the "boshes"—that is, the restricted part sloping to the tuyeres. In order to remove it, bricks are removed below the bear, and the obstruction is knocked away while hot. The bear always has the form of a ring. The hearth of the furnace is built upon an iron plate. Then follow three layers of common bricks placed flat, and then two layers of fire-brick; these together being about 16in. thick. Then follows another layer, 16in. thick, of the Chamotte bricks already described. The pressure of the blast is about 20 to 25 millimetres of mercury when smelting ore, and 15 to 20 millimetres when smelting slag. For driving the blast at the Muldenhütte there exists a twenty-horse-power water-wheel, used during the day for the liquation- and refining-furnaces, and during the night for the blast-furnaces; it also drives a small water-pump. There is also a twenty-horse-power steam-engine for the cupolas and liquation-furnaces, a fifteen-horse-power steam-engine for the cupolas and smithy, and an eight-horse-power steam-engine for the cupolas and steam-lift.

In twenty-four hours each cupola smelting ore treats about 1,200cwt. to 1,500cwt. The charge would consist of—roasted ore, 600cwt. to 700cwt.; slag from previous smelting, 500cwt. to 560cwt.; and other additions, 250cwt. to 300cwt. This charge requires some 115cwt. to 120cwt. of coke, or about 1cwt. of coke to 12cwt. of the ore-charge. The result of the smelting is about 115cwt. of lead, 36cwt. of lead regulus.

The charge described above is worked by five men at each cupola. They work in shifts of twelve hours. Below, 1 smelter and 2 assistants; above, 1 charger. The wages are about $\frac{1}{2}$d. for each hundredweight of ore.

The hearth is allowed to fill until some of the regulus is seen to come out with the slag. This is tapped continuously from one or other of two tap-holes.

The lead contains	0·5 to 0·6	per cent. of silver.
The lead regulus,	25 to 30	per cent. of lead.
	6 to 15	per cent. of copper.
	0·2 to 0·25	per cent. of silver.

The slags contain 3 per cent. of copper, 4 to 5 per cent. of lead, and 0·03 to 0·04 per cent. of silver.

Smelting the Rich Slag.—The slags may be viewed as three equivalents of monosilicate with one equivalent of bisilicate:—

$$3(MO_2\ SiO_2) + (MO_2\ 2SiO_2).$$

Such slags are thrown away as do not contain more than 0·001 to 0·002 per cent. of silver, and 1·5 per cent. of lead.

The charges in slag-smelting are very varied. The following are examples:—

(1.) Slag, 900cwt.; plumbiferous material, 45cwt.; broken-up hearths, 12cwt.: which yielded 52cwt. of poor lead, 85cwt. of coke being required.

(2.) For every 2cwt. of a mixture of 5cwt. roasted regulus from slag, 12cwt. roasted ore-regulus, 2cwt. roasted copper-regulus, 5cwt. roasted liquation residues of lead, is added 4½cwt. of slag from galena-smelting, and 4cwt. of common slag.

(3.) 3cwt. of ore-slag, 3cwt. of various residues rich in lead, 1½cwt. once-roasted ore-regulus, 0·5cwt. of coke. To this charge there is added, in the working-day of twenty-four hours, 12cwt. of marl, 12cwt. hearth, 12cwt. lead skimmings.

(4.) 6cwt. of slag, 25cwt. of twice-roasted regulus, 40cwt. of slags from the smelting of the litharge.

Products of the Slag-smelting.—Slag-lead containing 0·4 per cent. of silver, regulus containing 10–20 per cent. of lead, regulus containing 20–30 per cent. of copper, poor slag containing 0·0015 per cent. of silver and 1·5 to 2 per cent. of lead.

Sometimes a speisse of nickel and cobalt is also formed. The regulus is only about one-third of the amount of lead produced; the slags often contain 9 per cent. of oxide of zinc. The regulus is broken up and roasted in kilns or stalls, and is then added either to a smelting-charge similar to that by which it was formed, or it is concentrated in a cupola, or goes to a reverberatory for concentration with silica and barium-sulphate.

Smelting the Speisse.—The charge is 150cwt. of speisse, 675cwt. ore-slag, 4cwt. impure litharge, 44cwt. lead-residues, 5cwt. hearth, 40cwt. fluor-spar, 75cwt. coke. The products are 54cwt. lead, 121cwt. concentrated speisse.

The speisse is concentrated by continuous and alternate roastings and smeltings until it contains 20 per cent. of nickel, when it is sold. The poor lead goes to the liquation-furnace. The regulus is roasted in kilns and stalls to concentrate it until it contains 30 per cent. of copper; it is then called "copper" regulus. It is afterwards treated in a reverberatory. Incidentally, large quantities of impure lead, and substances containing lead-oxide, are produced.

The process employed for the reduction of the lead from such substances consists of a reducing-fusion. One such substance, the litharge derived from the rich argentiferous lead, contains about 78 per cent. of lead. The furnace used is a small Pilz furnace with four tuyeres. The charge in twenty-four hours is 1,500cwt. litharge, 450cwt. lead-slags, 50cwt. slags from a previous "revivification," 130cwt. coke. The products are a variety of lead—which, according to its purity, is either first liquated and then refined, or is taken to the Pattinson pots—and a slag containing 10 per cent. of lead. Part of this slag goes to the next charge, and part to the first ore-and-slag smelting.

Liquating the Impure Lead.—This is done at Freiberg on the inclined bed of a reverberatory furnace, 500cwt. being treated daily. The copper, with some lead, remains solid in the form of liquation-residues. The lead before the liquation contains 0·5 per cent. copper; afterwards it still retains 0·07 per cent. The residues are about 5 per cent. of the total lead; they consist principally of lead with 15 to 18 per cent. of copper, and are added to the slag-smelting in the Pilz cupola.

The scum from the Pattinson pots is also treated in this reverberatory, in admixture with lignite. When antimonial lead is liquated the temperature must be still lower, but a little wood must be kept kindled in the well of the furnace to prevent the lead solidifying.

If the lead contains 1·5 per cent. of silver, and is fairly pure, it is cupelled at once—that is, it is added to lead that has been already enriched by the Pattinson process. If it is impure, the lead is refined and Pattinsonised.

In refining, the lead is fused in an oxidizing atmosphere on the bed of a reverberatory furnace. The products are a variety of different kinds of litharge, to which the name of "abstrich" is given. It is nothing but impure litharge, and, as the nature and proportion of the impurities vary greatly, the appearance is very dissimilar: some are dark-grey and stony, others are light-yellow and crystalline in structure:—(1) Powdery-tin abstrich; (2) fused-tin abstrich; (3) arsenical abstrich; (4) antimonial abstrich; (5) litharge, impure, abstrich; (6) litharge, pure, abstrich. There is a fractional oxidation of the various impurities, the tin and arsenic going first. The time required to refine 40cwt. of lead is about fifty hours.

The products, in the case of lead smelted from ore, are as follows: 10cwt. of tin abstrich, or about 3 per cent. of the total lead, containing 11·3 per cent. of tin, 14·4 per cent. of arsenic, 2·8 per cent. of antimony; 20cwt. arsenical abstrich, or about 5 per cent. of the total lead, containing 8·5 per cent. of antimony, 8·9 per cent. of tin, 8·7 per cent. of arsenic; 30cwt. antimonial abstrich, about 3 per cent. of the total lead, containing 6–8 per cent. of antimony, 1·3 per cent. of tin, 4·4 per cent. of arsenic; 10cwt. impure litharge, about 3·2 per cent. of the total lead, containing 3·1 per cent. of antimony, ½ per cent. of tin, 2·0 per cent. of arsenic.

In refining lead produced from the smelting of slag, very similar varieties of abstrich are obtained, except that they are not so rich in tin as when lead from ore is refined. At the Halsbrücke Hütte a hollow pipe through which steam is passed is sometimes used as a mechanical stirrer when refining. In desilverising the tin abstrich, about 90cwt. of slag-lead is placed on the bed of the refining-furnace, and then, on that, 15cwt. of the abstrich, mixed with 4 per cent. of coal; the charge is melted down in three hours, and then another 15cwt. of abstrich and 4 per cent. of coal is added, and so on until the furnace is full. The desilverised abstrich is taken off (the silver passing into the lead) and run down in a small Pilz furnace.

The lead is refined and Pattinsonised. The antimonial abstrich is desilverised in the same manner, only 24cwt. of it are added instead of 15cwt., as it melts easier. The arsenical abstrich and impure litharge are directly revivified by being run down in a small Pilz furnace for metallic lead, the lead obtained being then refined. The impure litharge is added in part as a plumbiferous addition to the Pilz cupola smelting-ore.

The desilverised tin abstrich is smelted in a Pilz furnace, the charge being 100 parts of abstrich to 100–150 parts poor lead-slag, 50 per cent. of slags from the same working, and 20 per cent. of limestone. The lead obtained is revivified and poled in a Pattinson pot. The various kinds of abstrich obtained are desilverised over and over again, until they contain comparatively no silver. The composition of the hard lead obtained from the stanniferous abstrich is very varied. One variety contains 18 per cent. of tin, 10 per cent. of antimony, 2 per cent. of arsenic.

The desilverised antimonial abstrich is smelted in a Pilz furnace with slag and 10 per cent. of limestone. In twenty-four hours 100cwt. to 150cwt. are sent through. The hard lead is liquated and poled, and contains about 10 per cent. of antimony, 3 per cent. of arsenic, and 1 per cent. of tin. Each refining-furnace is worked by one refiner and two assistants.

Pattinson Process, as conducted at Freiberg.—The pots have a thickness at the bottom of 60–70 millimetres, and 48–50 at the sides. They are 1·56 metres wide at the top, and 0·85 metre at the bottom. They hold about 15 tons, and last about 500 crystallizations. Each pot has a separate fire; there are sixteen pots, sometimes worked in two batteries of eight pots each. They are worked by the one-third system. Each pot is kept at a determined percentage of silver, and assays are made daily. When fresh lead is added it is introduced into the pot containing the like amount of silver. At Freiberg it is usually the third pot from the left. The rich lead contains from 1½ per cent. to 2 per cent. of silver, and the poor 0·0018 per cent. The scum from pots 1 to 4 by itself is liquated, and similarly that from pots 5 to 8, and from 9 to 15 by themselves. In a case in which the enrichment had not been carried far enough, the following were assays of the pots: (1) 1·07 silver, (2) 0·80 silver, (3) 0·42 silver, (4) 0·33 silver, (5) 0·30 silver, (6) 0·26 silver, (7) 0·18 silver, (8) 0·11 silver, (9) 0·08 silver, (10) 0·04 silver, (11) 0·02 silver, (12) 0·012 silver, (13) 0·007 silver, (14) 0·0035 silver, (15) 0·0015 silver, (16) 0·001 silver. The poor lead never contains more than 0·05 per cent. of copper, 0·2 per cent. of iron, and traces of arsenic and antimony. The fuel used is a mixture of lignite and small coke. To each battery of sixteen pots there is one fireman, and to every two pots there are two men. There is also one lead-ladler. The men work for eleven hours, but the fireman works for twelve hours.

Cupellation at Freiberg.—The hearth is formed of 48cwt. of fresh marl mixed with ½cwt. of clay. The marl is of three kinds.

	I.	II.	III.
$CaCO^3$	50	68	66
$MgCO^3$	33	27	6
$FeCO^3$	2	2	2
Clay	14	3	35

Any pyrites present is carefully removed.

The greatest depth of the hearth from a line on a level with the tuyeres ought not to be more than 20 cent metres. The fire-resisting material with which the "hat" is lined consists of one part of clay and two parts of silica; 100cwt. to 200cwt. of lead is placed on the damp hearth, and this is covered with sawdust and chips; the "hat" is then put on, the wood lit, and the "hat" luted on with clay; the fire is kindled, and the blast turned on; the lead melts down in sixteen to eighteen hours. The temperature is gradually raised, and lead is added, 700cwt. being the total charge. The litharge, as it forms, is removed until the remaining lead on the bed contains 60 to 80 per cent. of silver. This is taken out, and the extraction of the silver completed in a similar but much smaller hearth. If the red litharge obtained contains less than 0·02 per cent. of silver, it is sold. It is said that if the lead contains 0·2 per cent. of bismuth no red litharge will be produced.

Where the concentrated lead and silver settle on the hearth there is a dark spot which indicates the presence of bismuth, and this part of the hearth is broken away and the bismuth subsequently extracted, this portion of the hearth having been previously hollowed to collect the rich lead. There are two men to a furnace, and they are paid a little over a penny for every hundredweight of lead treated. It takes 120 to 140 hours to cupel 700cwt. of lead.

The hearth on which the remaining lead is driven off is about one-fifth the size of the larger one. The silver is granulated by running it into water. It is about 990 fine, and is sent to the Halsbrücke works to have the gold parted from it.

Solution of the Regulus.—The concentrated copper-regulus is sent to the Halsbrücke works, where it is then roasted nearly sweet. It still contains about 1 per cent. of sulphur. It is then passed through a rather fine sieve, and the larger pieces are crushed and reroasted. The roasted regulus is dissolved in wooden vats lined with hard lead. The sulphate of copper is crystallized out, and after purification by recrystallization is sold. Such copper-sulphate as does not crystallize out but remains in solution is removed by scrap-iron. The solution of copper-sulphate before crystallization is made to pass over metallic copper, in order to remove any silver it may contain. The residue, after the treatment of the regulus with sulphuric acid, contains the lead and most of the silver originally present in the regulus treated. This is sent back to the Pilz furnace, with a view to concentrate the silver in the lead produced from the lead-ores.

The scheme (Fig. 4) accompanying this report comprises other metallurgical processes which are conducted at the Freiberg works. Their descriptions would not, however, vary in any important particulars from those which are to be found in every metallurgical text-book; and, as their relation to the scheme as a whole is indicated, they need not be further referred to here. They do not form a part of the distinctive Freiberg process, and are merely incidental to it.

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