Mining and Ore-Treatment at the Talisman Mine, Karangahake, New Zealand.

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Mining.

The Talisman mine is situated on the Waitawheta (pronounced Wytaffetta) river just above its junction with the Ohinemuri (pronounced Ohinnemurre) river near Karangahake, New Zealand. The explored reef-bearing portion of the claim includes the angle formed by the junction of the rivers and extends southwards for slightly over a mile. The country is mountainous - both rivers flowing through precipitous gorges. The trig station on Karangahake mountain is almost directly above the workings on the Bonanza section and is 1680 feet above river level, the adit from the gorge giving 1200 foot of backs measured vertically.

The officers of the N.Z. Geological Survey are engaged in a detailed survey of the Waihi district and the rocks of Karangahake are under examination.

The rocks of the district are andesites and dacites much altered and silicified. One of these exhibits a columnar structure in the mass in the Waitawheta gorge, near the surface workings on Taukani hill, also on the southern bank and in the bed of the Ohinemuri river just above the Crown mines battery.

Country-Rocks.

In the zone of oxidation, good "country" is a light bluish-grey compact rock which drills well ("lively country"), while in the lower levels it is darker and harder, and partial alteration of the mass has given it a patchy mottled appearance, the patches being only slightly lighter in colour than the rest of the rock. Poor country may be light bluish-grey and soft enough to clog the holes when drilling by machine ("dead country") or it may be less decomposed and hence darker and hard, so that it drills well, but owing to its jointed nature often breaks in rough slabs when blasted.

The principal reef (the Maria) does not necessarily carry payable values when traversing good country and its patchy nature may be partly due to secondary enrichment having transferred the values from barren blocks to the enriched zones below. In the Bonanza section, the richest portion of the lode, the variations in assay values are more noticeable than elsewhere and it is found that after stoping excellent ore the values sometimes decrease until no longer payable, though the change is not accompanied by any change in the macroscopic character of the country-rock. Thus, the vertical projection shows that no stooping has proceeded between levels 6 and 8. Again, between levels 11 and 12 and 12 and 13, in the northern portion of the Bonanza chute, there are horizontal patches which are unpayable for a vertical depth of about 70 feet in each driving levels and crosscuts.

Maria Reef.

The only reef which is of any importance at present is the Maria. It strikes approximately N. and S. The outcrop is seen on both sides of the Waitawheta gorge, and the reef has been driven on southwards for a length of over 4,000feet. It is a fissure reef formed by simple deposition and also by replacement of brecciated material in the fissure.

In the upper levels it is almost vertical. The northern half has a slight tendency to dip to the west; a small section of the southern portion dipped slightly to the east and crossed the boundary of the Crown mines down to the No. 7 level. Elsewhere in the southern section the dip was always westerly, and below No. 6 level it decreased considerably.

Pay-Chutes.

There are four pay-chutes, termed respectively the Woodstock, Talisman, Bonanza and Dubbo, which were formerly worked near the surface by separate companies. The width of the reef is very variable, and where payable, generally ranges from 2 to 12 feet.

Woodstock & Talisman Chutes: - In the Woodstock and Talisman chutes the ore has been subjected to oxidation down to the lowest levels of the mine and is not of high grade. It is generally white quartz with a faint bluish veined structure, due to sulphides of silver, &c., where values are present, and as a rule the valuable portion of the reef does not extend right across it, but occurs on either wall or in the centre.

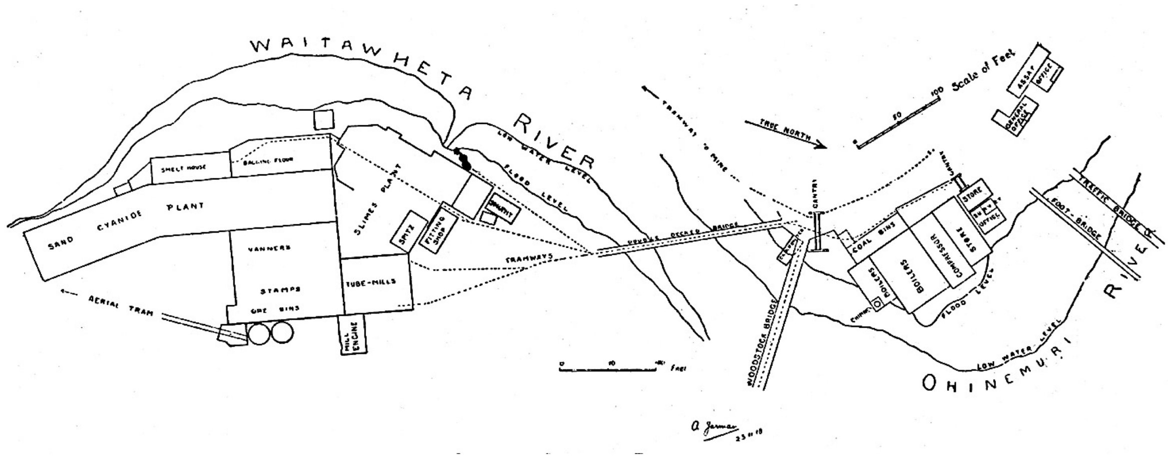
Bonanza Chute: - The Bonanza chute is the chief ore-producer at present. It is in this section that the change of dip has been so noticeable. Below No. 6 level it flattened until in places it was only 30 degrees from the horizontal. Below No. 12 level the dip again becomes steep, and this change was accompanied by increased values. In most places the reef was good where steep, and small and poor where flat, and a clay selvage, 3 inches or more thick, generally existed on the hanging-wall of the flattened reef. The length of pay-chute at No. 8 level was only some 20 feet, until at No. 12 it is 600 feet long, and at No. 13 over 1,000 feet, though it should be noted that there are stretches of unpayable material in these lengths.

The zone of oxidation does not extend below midway between Nos. 11 and 12 levels which is just about river-level and 1,250 feet vertically below the outcrop. In this zone the high-grade ore has evidently possessed the characteristic banded structure of the sulphide ore of lower levels, but oxidation has converted much of the sulphides into brown and red oxides of iron, thus giving brown and red streaks alongside the black.

Where masses of sulphide have been converted into haematite the gold is sometimes found scattered throughout the mass. It appears to be about 900 fine.

At No. 10 level the general proportion was 4 Ag (silver) to 1 Au (gold) and at No. 11 it had risen to 15 to 1, although the ore did not carry heavy black sulphide streaks.

At No. 12 level the ore is quartzose with streaks of soft black sulphides of lead and silver running parallel with the walls of the lode thus giving the reef its characteristic banded structure. Some of these sulphide streaks at No. 12 and 13 levels are 12 inches wide and very rich, so that some of this material was bagged and shipped direct to the smelter. The bullion carried from 10 to 30 Ag to 1 Au.



General Plan of Buildings. Fig. 1.

Two specimens of these sulphides were analysed and showed the following compositions: -

	From No. 12 level			From No. 13 level.		
	Oz.	Dwt.	Gr	Oz..	Dwt	Gr
Gold	41	1	13	38	2	18
Silver	781	4	11	163	1	18
				(Per long ton.)		
	%			%		
Gold	0.126			0.117		
Silver	2.391			0.116		
Copper	3.24			2.00		
Lead	6.00			2.21		
Antimony	0.42			traces		
Zinc	2.40			3.40		
Manganese	1.07			0.90		
Aluminium	1.20			1.35		
Iron	8.74			8.54		
Sulphur	7.40			5.50		
Silica	65.64			74.54		

Table B

At No.13 level the sulphide streaks are slightly harder and copper pyrites is more noticeable. Rhodonite (the silicate of manganese) which appears first at No. 12 level, is a true indicator of values in and below No. 13 level.

Where the percentage of copper pyrites was high the ratio of gold to silver dropped to 3 to 1, but in the southern portion of the chute copper is not so noticeable, the sulphides are more closely grained and the ratio of silver to gold is 8 to 1. At the bottom of winzes Nos. 6 and 12, which are now (December 1910) 80 feet and 150 feet respectively below No. 13 level, the proportion of copper has increased further and the ratio of silver to gold has fallen to 1 to 1. The variability of the gold-silver ratio and the existence of barren or poor parches in this rich Bonanza chute are both notable features.

At Nos. 5 and 6 levels, above the uppermost blank, there are considerable deposits of friable quartz with black oxide of manganese, and below there is the rich sulphide zone, strongest at No. 12 level and again below this is the rhodonite gold zone at No. 13 level. These zones, arranged as described, suggest very forcibly that secondary enrichment has taken place after the manner recently explained in detail by W. H. Emmons of the Geological Survey of U. S. A. In this connection it should be noted that there have been no alluvial workings in the district.

The Dubbo Chute: - This chute gave a small run of values on No. 10 level but was poor on each of the succeeding levels. At No. 11 there is about 200 foot run of friable quartz with manganese oxides, which is unpayable, and, in view of the experience on the Bonanza section it will be most interesting to watch developments on this chute at greater depths.

Branch Veins.

A branch vein diverges from the Maria reef in the Talisman section at various places above No. 9 level on the foot-wall side in the upper levels and on the hanging-wall side at Nos. 8 and 9. Below the latter it died out. A considerable tonnage was obtained from this branch.

An east branch vein, about 3 feet wide, commences above No. 11 level in the Bonanza section. At No. 11 level it runs alongside the main reef for 500 feet at a distance of 10 to 50 feet, but junctioning with it at both ends. The values here are good. On No. 12 level it diverges up to 200 feet from the parent reef for a length of 500 feet. The values here are not so good as on No. 11 and do not extend the full length of the branch. At No. 13 it has been followed for 250 feet and only carries payable values for a very short section, which has been winzed on for 80 feet and still carries payable ore.

A hanging-wall branch vein has been cut by a crosscut on No. 13 level and driven on for 200 ft in which two small sections were payable. These will be tested by winzes.

Other Veins.

In the Woodstock section there is the Woodstock reef which runs parallel with the main reef at a distance of about 550 feet. It is 4 to 5 feet wide and was only payable near the outcrop. Between these reefs are two smaller ones termed Shepherd's and Cornes' reefs both of which have recently been cut in crosscut No. 8 from No. 12 level. Cornes' reef is of low grade, but Shepherd's was high grade near the outcrop and values continued for a depth of 350 foot.

These reefs which have been worked on the north side of the Waitawheta river on Taukani Hill are probably continuations of Shepherd's and Maria reefs. The workings extend nearly to river-level and when the Woodstock shaft has been sunk to 400 feet below the river, a level will be extended under the river to explore these reefs at depth. The summer flow of the river is 21.3 sluice heads or cubic feet per second (a sluice head in New Zealand = 1 cubic foot per second).

Welcome Reef.

In the Crown mine, which adjoins the Talisman on the eastern boundary, the main reef is the Welcome lode. At river-level it runs approximately parallel to the Maria (North and South) about 1,000 feet from it. It has been intersected in the Dubbo section of the Talisman mine by a crosscut from No. 11 level and some 530 feet of driving was done on it without meeting payable ore. This reef dips west, i. e. towards the Talisman boundary.

Production of the Crown Mine.

	Long Tons	Value of Bullion
Since 1891 to end of 1905	274,559	£625,434
1906	22,000	40,735
1907	22,072	57,242
Shut down November, 1908	14,921	25,260
1909	from cyanide plant	250
Restarted, May 1910	6,661	13,600

Table C

Output of the Talisman Mine.

Year	Long Tons	Value of Bullion		Remarks.
Ending:- Mar. 31, 1895	280	£2,390 12s 0d.		Trial crushing. Co. registered Nov.5, 1894
Ending:- Mar. 31, 1897	285	1,510 19s. 0d.		Trial crushing.
			Per ton	
1897	4,194	£13,681	£3 5s. 3d.	12 months to Mar. 31.1898
1898	8,696	32,648	3 15s. 1d.	12 months to Mar. 31.1899
1899	9,558	33,028	3 9s. 1d	12 months to Mar. 31. 1900
1900	7,961	21,980	2 15s. 3d	9 months ended Dec. 31, 1900
1901	11,252	17,256	1 10s. 8d.	12 months Dec. 31, 1901
1902	13,396	20,163	1 10s. 2d.	
1903	47,267	94,134	1 19s. 10d.	
1904	44,888	84,826	1 17s. 10d.	Woodstock Co. absorbed.
1905	44,725	129,088	2 17s. 9d.	
1906	49,573	152,011	3 1s. 4d.	
1907	46,025	184,445	4 0s. 2d.	
1908	46,417	218,975	4 14s. 4d.	
1909	46,456	208,886	4 9s. 11d.	
1910	45,020	210,264	4 13s. 5d.	
14 years		**£1,421,385**		

Table A

Faults and Breaks.

The Maria reef is intersected by several faults which run east and west and displace the reef from 5 to 15 feet. Two of these, the Woodstock and Bonanza faults, pass through the pay-chutes and consequently have been followed with interest. The Woodstock fault dips south at about 60 degrees from the horizontal and keeps a very steady course through the Talisman chute. Between Nos. 10 and 11 levels, good ore existed only on its south side, but between levels 11 and 12 there was good ore on both sides, that on the north being of lower grade than that on the south. Between Nos. 12 and 13 it passes through the stopes and 80 feet below No.13 it cuts the incline shaft, and the water from it was too much for the pumps. The fault-material is clayey and oxidised and the stopes on either side contain a considerable quantity of it either as horses or false walls.

The Bonanza fault dips to the north, the amount of dip varying as shown on the vertical projection. Above No. 10 level it is in the centre of the Bonanza chute, between Nos. 11 and 13 it runs approximately on the northern boundary, but below No. 13 level, in No. 6 winze, high-grade ore occurs on its northern side. It is interesting to note that what is probably the same fault passes through the principal pay-chute of the Welcome reef in the Crown mine, these chutes being opposite to one another on parallel lines of reef.

System of Mining.

Access to Workings. Down to No. 8 level, access to the workings in the Talisman, Bonanza and Dubbo sections was obtained by adits only.

Talisman Shaft. From this level an incline shaft was sunk dipping west 62 degrees from the horizontal and also pitching slightly south. This was put down by the old Talisman company. Down to No. 11 level it is divided into three compartments, viz., a ladderway and two quartz skipways. From No. 11 downwards a fourth compartment was added, the dimensions then being:- shaft overall, 18 feet x 6 foot, containing two quartz skipways together occupying 7 foot x 4 foot, one mullock skipway 3 foot 6 inches x 4 foot, and a 4 foot 10 inch x 4 foot compartment accommodating the ladderway and pump pipes. At present, ore is hoisted by air-winches up to the top of this incline shaft at No. 8 level, from which it is trucked by horses to the top station of the aerial tramway connected with the mill.

Woodstock Shaft. In the Woodstock mine (absorbed by the Talisman Co. in 1904) access was obtained by adits driven on the reef from the precipitous edge of the Waitawheta gorge. The river level, which is about 30 feet above ordinary water-level, now provides the chief entrance to the Talisman mine. The tramway to the mouth of this level is cut out of the rock in many places. After driving this level 100 feet, a station was cut in which headgear was erected and a shaft then sunk vertically for 240 feet.

The shaft has three compartments, a pumping and ladderway compartment 6 foot 6 inches x 6 foot and two winding compartments, each 4 foot x 6 foot inside timbers. Sinking is to be continued for another 200 foot and a straight haulage level will then be driven to connect with No. 14 level which will be the next one to be opened up on the Talisman and Bonanza chutes.

Ore is now being obtained on the Woodstock chute at No. 12 level, and should this be found to continue downwards, the haulage level would be driven on the course of the load. Electrical haulage will be used. This shaft will be the pumping shaft and will be sunk in advance of other development workings so that there should be no further trouble occasioned by water. To accommodate the pumping-engine, winding engine and pump-winch, the existing chamber had to be considerably enlarged. An account of this work is given later. (See Chamber)

Considerable economy should result from using this shaft as a main travelling and haulage way. At present, the miners walk in along the river level to the Talisman shaft (1,650 feet) and those who work on Nos. 12 and 13 levels then descend the ladderway. All this travelling is done in the company’s time, i. e., wages are paid from the time of entering to the time of leaving the mine. When the new arrangements are completed they will descend by cages to No. 14 level and use trolleys to get to their work. Apart from the time saved there will be considerable husbanding of energy.

System of Development.

The horizons of former levels are shown on the vertical projection. The present system of development is to open up a new level for each 200 feet sunk in the Talisman shaft, measured along the dip. The levels are driven 8 feet x 5½ feet in the clear and rise 9 inches per 100 feet. Connections are established from 100 foot to 150 foot. Should this development work disclose a barren zone (which would generally be horizontal) and should subsequent stoping prove it too be extensive, an intermediate level is driven from the rise at the place where values again become satisfactory. In this way the breaking of unpayable material is avoided as far as possible.

Overhand stoping is used; the men standing upon broken country-rock obtained from development work or by robbing the filled abandoned stopes overhead. Any mullock obtained from No. 13 level is hoisted by the mullock skip to a hopper at Nos. 11 and 12. Between Nos. 12 and 13 the only material available is that obtained on No. 12 level, and when this is insufficient, or when the reef dips at less than 45 degrees from the horizontal so that mullock will not pass down the rises, it is necessary to project rises into the hanging-wall country to obtain material for filling. These mullock rises are driven at a grade sufficient for the material excavated to run down into the stope with little assistance (about 50 degrees) and the contractors are paid per foot run of rise measuring 8 foot x 8 foot, or its equivalent in cubical contents.

After starting, they prefer to open out to not less than 9 foot x 12 foot. The price paid (25shillings per cubic foot) includes the distribution of the material in the stopes. The length of such a rise depends upon the amount of material required and may be up to 80 feet. Though this is dead work and expensive as a filling, such rises have occasionally been useful in proving branch veins in the hanging-wall country. Stoping proceeds from the main rises in a series of slices sloping at the angle found to be most convenient for shovelling the quartz and filling. Should a stope disclose an extension of values horizontally, either north or south, then the end of that stope is carried on as an intermediate level until the ore is no longer payable. For convenience in handling the ore it may then be necessary to rise from the level below up to this point.

The filling is covered with planking before ore is shot down, and, when rich, sacks are placed underneath the planks to prevent loss of fine friable rich sulphides.

Rock drilling in stopes is performed by machines as far as possible. The machine used is the Improved No. 2 National made by Taylor Horsfield of Bendigo, Victoria, Australia. The piston is 3 and one eighth inches in diameter and has a 5½ inch stroke, giving approximately 400 strokes per minute, with a pressure of 80lb. of air at the machine. The men object to anything lower than 70lb. The valve of the receiver at the river-level of the Talisman shaft is adjusted to blow off at 90lb, and it is assumed that the loss by friction from here to the stopes is about 10lb.

The machine weighs 300lb. and the columns vary from 2 foot 6 inches to 11 foot 6 inches long with a screw bar at one end.

The drill steel used is M. M. Toledo. Star bits are employed, the pitching bit being 2¾inch in diameter and 1 foot 6 inches long and the finishing drill 1¾ inches and 7 foot long. The same machine and drills are also used in development work. In ordinary faces a 4 hole pyramidal centre-cut is drilled and from 15 to 18 holes for the whole round.

Nobel's 1 and one eighth inch diameter gelignite, blue-jacket fuse and No. 6 detonators are used.

During boring, a water jet is employed, the water being taken from the rising main of the pump, the pressure varying up to 130 or 140lb. per square inch according to the height of the stope. When blasting, a nozzle three sixteenth of an inch in diameter is screwed on to the end of the compressed-air supply pipe and with this safe-guard against waste, compressed air is used to clear away the smoke.

It is interesting to note here that one of the above machines was employed for drilling 6 inch holes for the foundation bolts of the large Cornish pump. The drills were of 2 inch diameter hexagonal steel, the pitching bit was 6 and five eighths in diameter and the finishing drill 6 inches in diameter and about 7 foot long, weighing about 84lb. The rate of progress was 1 foot per hour in hard andesite. A tripod was not obtainable and a jury rig was substituted in the form of a bar of 4 inch steam-piping clamped on to two chairs or pedestals each 3 foot 6 inch high made from 2½ inch x five eighth inch iron bar and bolted on to an 8 inch x 4 inch wooden base-piece.

Excavation of the Pumping Chamber.

In order to accommodate the pump-capstan, new winding engine and pump-engine, the chamber at the head of the Woodstock shaft had to be enlarged. That portion which accommodates the pump-engine is 100 feet x 40 feet x 25 feet high and its excavation was interesting.

On the south end, the rock was a firm blue altered andesite, decomposed to a softer brown rock in patches, especially adjacent to a small quartz-vein which ran through the chamber at an acute angle to its longer axis. This softer material made the roof treacherous and therefore it was deemed advisable not to use machine drills. The whole of the work was done by hand drilling and day labour.

From the existing chamber, a drive was projected to the full length of the required extension and this was met at right-angles by a crosscut from the river-level, thus providing the necessary ventilation. Leading stopes, with a height of 4 foot, were carried along this drive and also along that portion of the crosscut which was to form the back or end of the chamber (fig. 2).

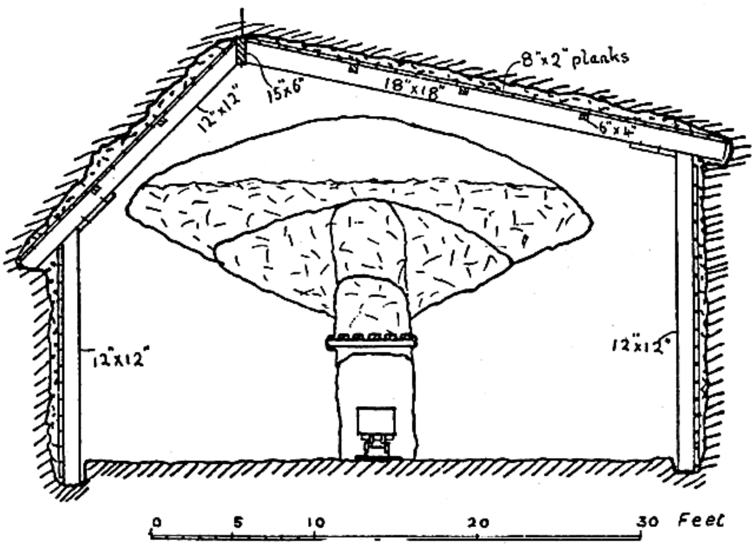


FIG. 2.

Section of Pump-Chamber, Showing outline of cuts made in excavating.

Light stulls were placed in hitches cut at 7 foot from the floor and at intervals of 3 foot 6 inches. Across these were laid rough slabs to support the broken rock and the excess was drawn off into trucks by moving or withdrawing one or two of the slabs after the manner of a “Chinaman.” Another stope about 5 foot high was carried along and then the sides were broken into about 2 foot above the stulls, the floor formed on either side being given an upward pitch. This procedure was repeated until the height of the centre of the arch was reached and the necessary width obtained. Sufficient broken rock was always left in the stope for the men to stand on and conveniently sound the roof for treacherous or loose ground. A 6 foot barrier was allowed to remain between the new chamber and the existing one until the timbering of the former was completed, an inclined rise being put through it so that timber could be hauled into the chamber by a hand winch.

The roof at the centre and western ends was at all times heavy and it was thought safer to catch it up by rafters and close slabs before excavating the floor. Also, at this stage of the proceedings, the timber was more easily handled than would have been possible had the chamber been completely excavated.

Twenty sets of rafters were put in at intervals of 5 foot. These are of kauri 20 feet x 12 inches x 12 inches at the south end where the span is 30 feet. The rafters increase on the west side of the chamber to 30 feet. x 18 inches x 18 inches towards the north end where the span widens to 40 feet as shown in Fig. 2.

Hitches 2 foot high x 2 foot 6 inches wide were cut to receive the rafters, the depth of the hitches varying from 2 foot 6 inches to 4 foot according to the state of the rock. A centrepiece of 15 inch x 6 inch hardwood was suspended by wrought-iron bolts, 10 feet apart, let into the rock and wedged firmly in place. Against this centre-piece the upper ends of the rafters rested; 6 inch x 4 inch spreaders or distance pieces were put between rafters to prevent end movement, two or more being used to each space. Planks, 8 inch x 2 inch, were placed across the rafters and the space behind them filled with broken rock.

The roof having been secured, the end barrier was removed and the timber over this portion completed and joined to that of the existing chamber, which was 40 feet high.

The broken rock was then removed from the chamber and the excavation of the floor portion commenced from the south end, the work proceeding in benches from the centre so as to take advantage of the rill as far as possible. The walls were next secured by uprights of rata 16 foot x 12 inches x 12 inches, let into hitches 12 inches deep in the floor and the top end of the leg was cut to fit against the rafter and kept in place by a block nailed on to the rafter. Kauri planks, 8 inches x 2 inches, were placed behind these uprights and the space behind them filled with waste rock. An inside roof was constructed of 12 inch x 1 inch boards lined behind with ruberoid in order to prevent slightly-acid water from dripping on the machinery.

The winding station was then widened on the east and west sides to accommodate a new winding engine and a capstan engine for handling pump rods. The roof was also heightened to enable the 40 foot rods to be lowered.

The completion of this chamber together with the erection of two Babcock & Wilcox boilers on an outdoor site excavated on the side of the gorge will enable the existing 240 foot shaft to be utilised in the near future. Any outside site for a shaft would have been exceedingly difficult of access owing to the precipitous nature of the country and would also have entailed much extra sinking.

Ventilation.

The arrows on the mine plan and elevation indicate direction of air currents.

The ventilation is natural, the river-level being the intake and the Talisman inclined shaft the upcast.

The ordinary air-current at the intake = 13,000 cubic feet per minute, and the compressed air used = 3,500 cubic feet per minute, giving a total of 16,500. The number of men underground is 180 to 200.

At the north end of No. 13 level the temperature is 71 degrees and at the south end 75 degrees fahr (November, 1910).

Sanitation.

Sanitation is carefully attended to and no case of ankylostomiasis has been suspected.

Drainage.

All levels rise 9 inches per 100 foot. Surface water is picked up at No. 7 level and is utilised to drive the smithy blower at the mine buildings at the entrance of No. 8 level.

Practically no further water is encountered until below No. 13 level, where the average flow is 25,000 to 30,000 gallons per hour, partly from the bottom of the shaft and partly from No. 12 winze. When the Crown mine was idle in 1909, there was much more water to be handled, about 41,000 gallons per hour, and it is quite probable that a cross-fissure connects the Maria reef-system with that of the Welcome reef, the flow through it being somewhat restricted. The above water is dealt with by the following pumps:-

Suspended at the bottom of the Talisman shaft are No. 12 and a No. 9a Cameron pumps which are constantly in use and a No. 8 which is kept in reserve; the capacities of these three being respectively 18,000, 12,000 and 11,000 gallons per hour at a piston speed of 100 foot per minute as ascertained by pumping into a cistern of known capacity at the plat of No. 13 level, the lift being 80 feet. From here the water is lifted 200 feet to the river-level by two Austral Otis horizontal station pumps, each of a tested capacity of 22,500 gallons per hour at 45 r. p. m. These pumps have two air-cylinders, 12 inches in diameter x 25 inch stroke, with cranks at 90 degrees. Behind each air-cylinder are two water-cylinders (i. e. 4 water-cylinders to each pump) 5¾ inch in diameter x 24 inch stroke each on the same centre line. The plunger of the first is driven directly by a tail piston rod from the air-cylinder and that of the second is driven by two outside rods connected with the crosshead and passing one on each side through the cylinder flanges, which act as guides, to the rear end of the water-cylinder. Each water-cylinder with its valves, is a separate unit, and can be disconnected if necessary.

To prevent the formation of ice at the exhaust ports, and dense fog in the exhausted air, the exhaust chamber has been tapped and a ½inch pipe supplies water from the rising main to each exhaust port. This has effectually prevented the formation of ice and fog, but, if the water supply be cut off from the ports, frost forms in less than two minutes. The air exhaust pipe is fitted with a drain pipe to carry this water back to the cistern.

A No. 12 Cameron pump is installed in winze No. 12 below No. 13 level, and is supplied with air by a 4 inch pipe from the 6 inch air main in the Talisman shaft. After the fire destroyed the old compressor house, this pump had to be stopped as the air was required for ordinary drill work. It was then found that running at 80 strokes (16 inch) per minute, it took about 161 cubic foot of air at slightly under 80lb., which was equal to the air required for 18½ of the drill machines. This amount of air pumped 19,000 gallons per hour through a lift of 120 feet. The Austral Otis station pumps use the air expansively and take 190 cubic feet of air at 80lb. for a capacity of 45,000 gallons per hour through a lift of 340 feet.

Other pumps held in reserve are as follow:-

Cameron No. 12, at No. 13 level to pump from cistern to the river-level, a lift of 340 feet.

Cameron No. 9, at same place to pump to No. 12 level.

Cameron No. 10, at cistern on No.12 level to pump to river-level.

Tangye pump- 2 foot stroke, 12 inch air-cylinder and 6 inch water cylinder. This is a leather-packed piston pump which delivers to the river-level.

Cornish Pump. - The new Cornish pump which is being installed in the Woodstock shaft is estimated to have a maximum capacity of 124,000 gallons per hour in three lifts of 250 feet each.

92,500 gallons per hour through 1st lift.
99,500 " " " " 2nd lift.

124,000 "   "   "   "   3rd lift.

It has three plungers, each 26 inch in diameter and with strokes of 10, 8, and 6 foot respectively; at 10 r.p.m. of the crank-shaft, this gives plunger displacements of 124,000, 110,400 and 102,800 gallons per hour or, allowing 10% slip, which is probably excessive, a capacity of 124,000, 95,500 and 92,500 gallons respectively. The engine driving this pump is a cross-coupled compound Corliss engine with cylinders 18 inches and 34 inches in diameter and, by a rope drive of sixteen 2 inch ropes, it drives a 20 foot rope-wheel connected to the pump gear, thus giving a reduction of speed of 2 to 1. The big spur flywheel of the pump-gear is 17 feet 6 inches in diameter and weighs 43 tons. The spur gearing, by which it drives the pump, effects a further reduction in speed of 5 to 1, a total reduction of 10 to 1. By this means the piston speed of the engine is kept reasonably high. Steam is supplied at 150lb. per square inch by two Babcock & Wilcox (W. I. F. type) boilers each of 1619 square feet heating surface; super-heaters each with 250 square foot, and chain-grate mechanical stokers. The exhaust steam goes to a jet condenser of barometric type.

The pump rods are of Oregon pine, in 40 foot lengths, and are respectively 24 inch, 22 inch and 18 inch square for the three lifts. Butt joints are used with 15 inch x 1 inch side-plates, 18 feet long. The balance bob has a radius of 16 feet and will weigh about 40 tons. The pump column is 24 inches in diameter.

Haulage and Winding.

The mine trucks have a capacity of 16 cubic feet and the broken ore averages 25 cubic feet per ton of 2240lb. One of the wheels on each axle is loose and the other fixed, one free wheel being on the right hand side and the other on the left. This ensured easy running. The gauge is 18 inches and the rails are 14lb. Ore and mullock pockets are used, from which hoisting material is done in skips. From the mine trucks the broken material is tipped between the rails directly into the hoppers and, to prevent large lumps from going through, a 4 inch x 1 inch iron bar is secured between the rails, thus reducing the width of opening to 7 inches. Large lumps therefore have to be spalled, and in this way the doors of the ore-pockets are kept in good order. The quartz-hoppers have been made as large as could conveniently be arranged - about 75 tons capacity. Mullock hoppers are smaller - 20 tons. They are excavated in the hanging-wall. Formerly, the bottoms of these pockets were lined with boiler-plate, but this wore away quickly. They are now protected by old rails laid closely together after the fashion of grizzly bars. Fine stuff packs itself between them, and the protection afforded has prevented further wear.

The average load for an ore-skip is 25 cubic feet (25 cubic feet = 1 long ton) and for the mullock-skip. 15.5 cubic feet. The bearers are 50 foot below the level floors and the tracks are laid with 30lb. rails set to 2 foot 5 inch gauge. Winding of ore is done chiefly on the day shift, but is not confined to it. A good average for the 8 hour shift is 150 long tons from the hoppers on No. 12 and 13 levels to No. 8 level. The winding engine is operated by compressed air. It has two cylinders, 12 inches in diameter x 15 inch stroke, and gearing 5 to 1. The drum is 3 foot 6 inches in diameter and rope 1 inch.

From the hoper at No. 8 level the ore is trucked by a contractor to the aerial tram terminus outside the mine. The trucks are 21 cubic feet capacity and the way is laid with 28lb. rails at 2 foot 5 inch gauge. The horse is loaned by the company and the contractor has to get out 160 tons per day. Another contractor despatches the ore to the battery by the aerial tram. A skip-load is 15 cubic foot or 12 cwt. of ore. The grade of the line is 18 degrees and the usual time taken for filling, is 16 seconds and, travelling, 31 seconds, total 47 seconds - say 40 tons per hour.

Power (Underground).

With the exception of the Cornish pump, which is steam driven, all pumping and winding below ground is done by compressed air. The pressure at the receivers at the compressor houses is from 90 to 100lb per square inch. The main to the receivers in the mine is 8 inch and the fall of pressure to No. 13 level, a distance of over 4,500 feet, is about 6½lb. By the time the air reaches the working faces the fall is about 15%. The pipe-line is closely inspected every day and two pairs of fitters make the rounds of the working faces and pipe-lines to effect repairs.

Sampling and Mine-Plans.

Development progress is recorded by an assay plan and longitudinal section. Each stope is also recorded by an assay plan and section. Samples are taken from the face in all cases, the width of section represented by a sample never exceeding 5 foot.

Labour.

The contract system is employed wherever possible. One contractor, who usually has from 6 to 12 partners, will be solely responsible for the whole of the mining and exploration on his particular level, with the exception of the timbering for shoot, and leading stopes, which work is done by timbermen paid by the company. The contracting party may have under them some 50 men. The contractor obtains his steel, dynamite, candles &c, from the company and these, plus drill-sharpening, repairs, breakages, depreciation of hose, loss of steel, &c., are entered as a contra to his account. The contractor delivers the ore at the ore-hopper of his level, and the tonnage is estimated by counting the number of skiploads received when hauling from there to No. 8 level. Payment is made on this tonnage, which is checked by the number of truckloads taken out by horse-tram to the aerial ropeway.

In driving, rising, or sinking, payment is made per foot of progress and if the material excavated is sent to the mill, payment is then made at an alternative price per foot, which is included in the original tender.

The filling of stopes is done by the same contractor, and timbering, such as cribbing of passes and manways, and stulls to support weak hanging-walls, is paid for separately at per running foot and per stull. When sets are required for drives, or framed sets for winzes or rises, payment is made for their erection at per set.

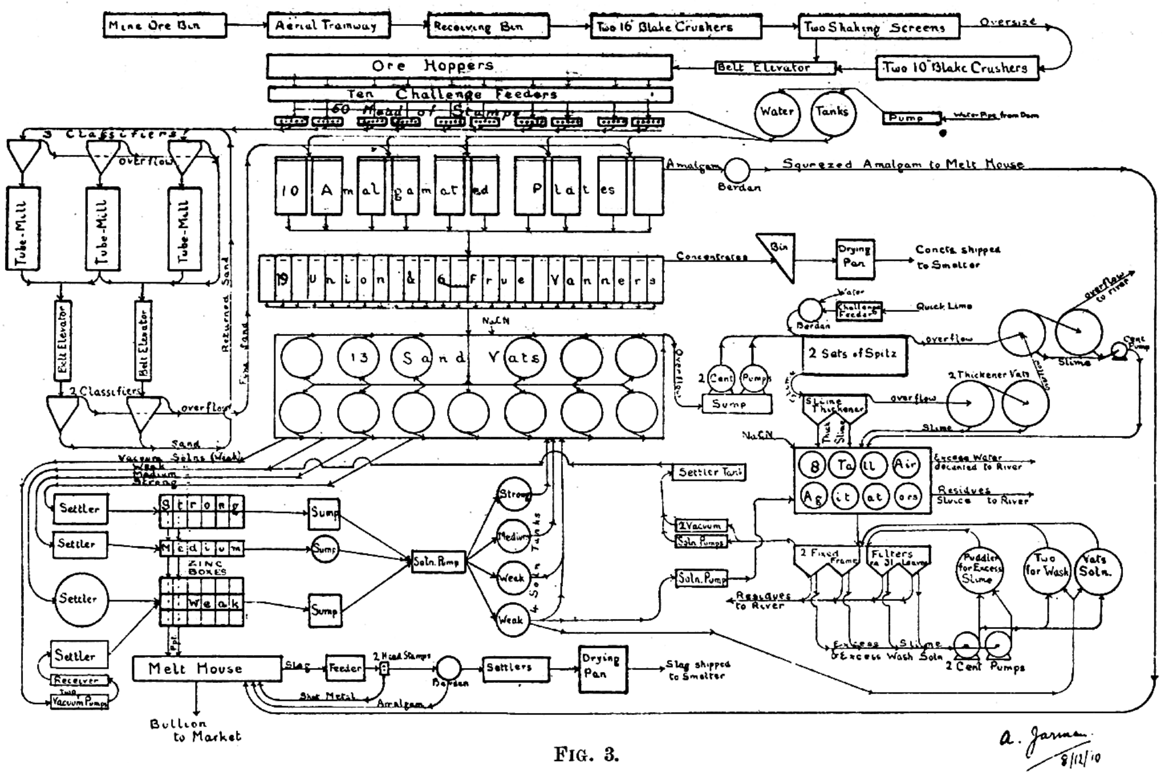


FIG. 3.

Flow Sheet, December, 1910.
Talisman Consolidated G. M. Co. Ltd.
Fig. 3

Ore-treatment.

The general scheme of ore-treatment is shown by the flow-sheet, Fig 3. Fig. 4 is the plan of the mill; Fig. 5 is a vertical section through the battery, concentrating and vat floors, and Fig. 6, a section through the tube-mill house and slime-plant.

Preliminary Crushing.

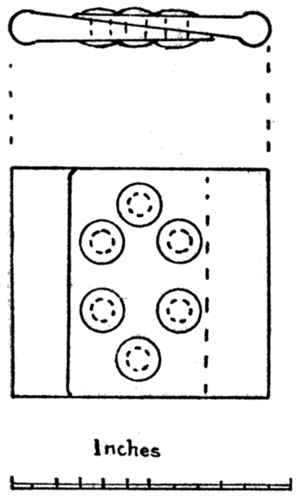


Fig. 7
Safety Toggle-Piece.

An ore-bin, which will hold about 220 long tons, receives the ore from the aerial tramway. From this it is shovelled down two shoots into two 16 x 10 inch Blake crushers, running at 240 r. p. m. and the crushed material falls on to two shaking screens of 1¼ inch steel plate perforated by 1½ inch diameter holes. They are set at a slope of 29 degrees and make 200 of 1½inch strokes per minute. The undersize goes directly to the elevator boot and the oversize falls into two 10 x 7 inch Blake crushers running at 218 r. p. m. and thence to the elevator. The shaking screen eccentrics are on a shaft driven from a pulley on an extension of the shaft of one of the large breakers. Any stoppage of this breaker stops both screens, but otherwise the two crushing sets are independent of each other.

Shearing toggles, composed of two halves held together by six 1 inch rivets, are used on the small breakers. The strain occasioned by a hammer head getting between the jaws is sufficient to shear all these rivets and thereby save the main parts of the machine.

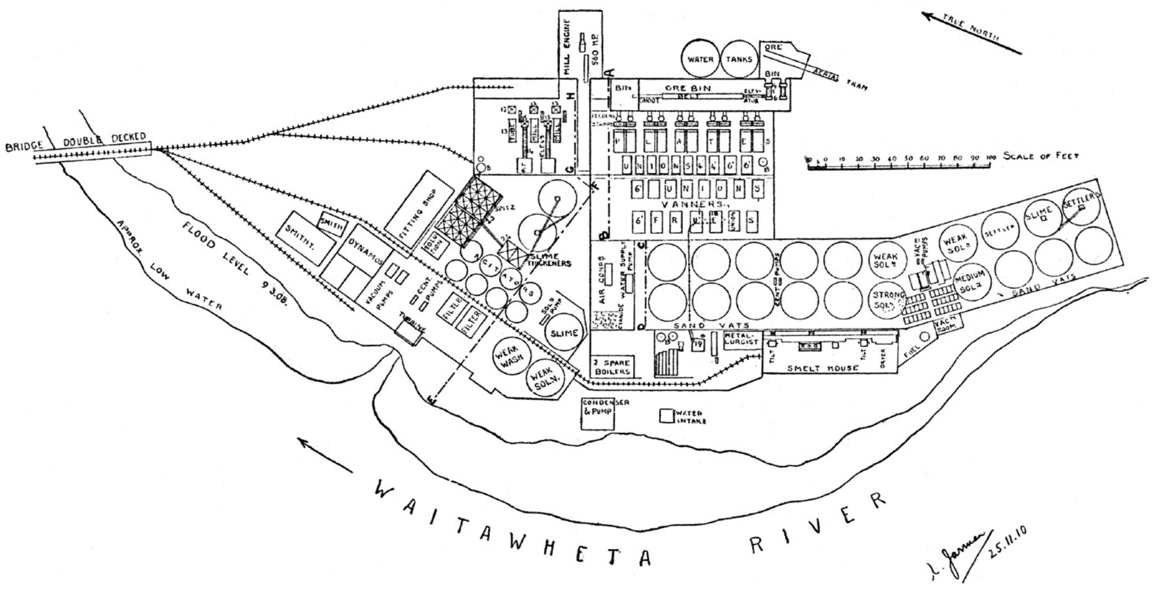
The elevator runs at 170 buckets p.m. of 0.2 cubic feet capacity. When dealing with clayey material (such as that found near the Woodstock fault) the buckets have to be cleaned out occasionally. The elevator delivers to 15 inch belt conveyor 52 feet long, running at 260 ft. p. m. which delivers the ore through shoots to various parts of the main hopper with a capacity of 400 long tons (1 long ton = 25 cubic feet approx.).

That section which feeds the first ten stamps is partitioned off from the rest so that tributers' ore may be kept separate from general ore until it has passed the stamps.

The floor of the hopper is inclined at 45 degrees and the sides slope similarly near the bottom, so that it is almost self-discharging.

50 Head Stamp Battery.

The usual rack and pinion discharge doors deliver ore to 10 suspended Challenge feeders each actuated by the central stamp tappet of the 5 head mill it feeds. The friction block of the feeder is of the double groove type. The battery was erected in 1901. Each mortar rested on a block of kauri 20 feet x 5 feet x 2foot 6 inch set in a block of concrete running right across the site. The mortar weighs 3½ tons. It rests upon a double sheet of tarred felt and is anchored to the block by eight 1½ inch cotter bolts, 3 feet 11 inches long, the cotter and washer (4½ inches x 4½ x ½ inch) being accommodated in a recess 6 inches x 5½ inch x 4 inch in the side of the block. Steel lining-plates are used inside the mortars, the two end ones being ½ inch thick, those on either side of the feed opening are 1/8th inch, and those on the feed opening and the lower half of the back of the mortar are 5/8th inch. The latter lasts from three to six months.



Plan of Talisman Mill

Fig. 4.

The stamps when new weigh 1250lb. per head.

	Ft	in		in		lb.
Head	1	2	x	9	diam	363
Shoe	0	9	x	9	diam	200
Steam	16	0	x	3½	diam	500
Tappet	1	1	x	9	overall	187
Total						**1250**

To compensate for wear and keep the weight per head fairly even the last ten stamps have heads 2 foot in height and are fitted with partly-worn shoes from other stamps.

Fraser and Chalmer's cast-steel heads and shoes are used and the dies are of local (Price Bros, Thames) cast-iron. They are 9 inches x 6 inches high and have an octagonal base. Formerly steel dies were used and the wear was consequently very uneven. The order of drop for the adjacent sets of 5 head is 1, 3, 5, 2, 4, and 10, 8, 6, 9, 7. The log-book rate is 106 drops per minute, but it is usually from 108 to 110; the height of drop is set to 7 inches which is reduced to 6 inches by the material in the die. When this increases by wear to 7 inches the tappets are raised to adjust it to 6 inches again. The height of discharge is ½ inch when the die is new. The cam-shaft for each set of 5 -head is separate and has its own driving pulley and friction clutch on the main shaft, the clutch being operated either from the cam-shaft floor or the stamp floor near the foot of the plates. Blanton cams are used, of the single-wedged type, and iron guides as well as plain wooden ones - the iron ones being preferred. Woven steel-wire screens are employed, 40 head being on 15 mesh and 10 head on 25 mesh (December, 1910). The stamp duty is 3.4 tons per head per 24 hours. The stamp duty during 1909 when 30 mesh screens were used (tube mills not being available) was 3.18 long tons (=3.56 short tons) per head per 24 hours, 44,800 tons being treated in the 12 months. The height of discharge was variable, being raised for rich ore to crush it more finely than general ore.

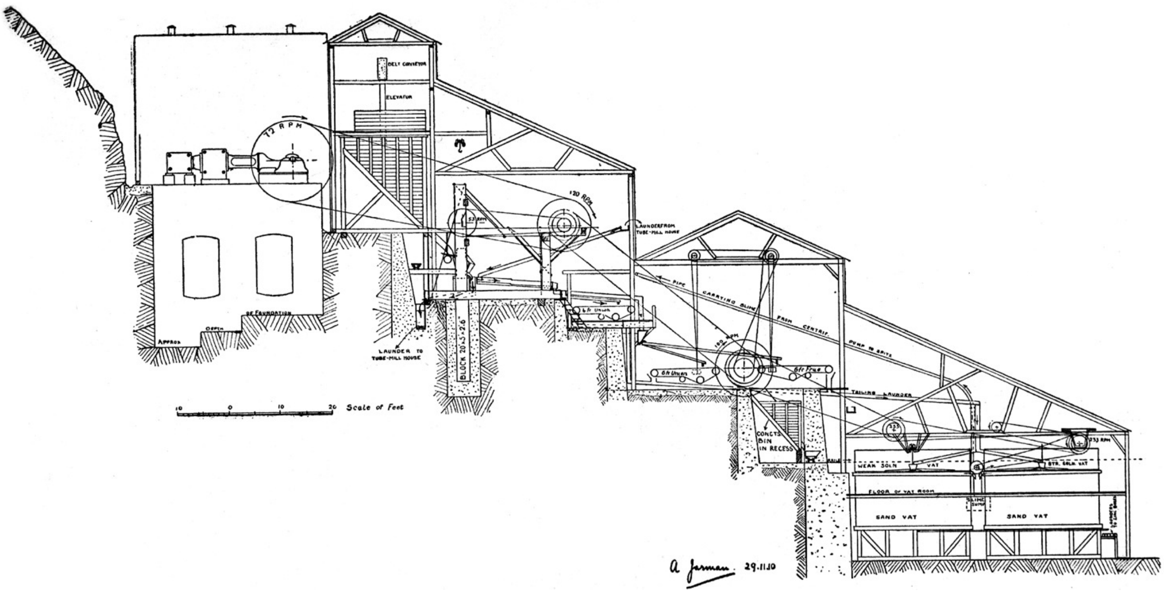
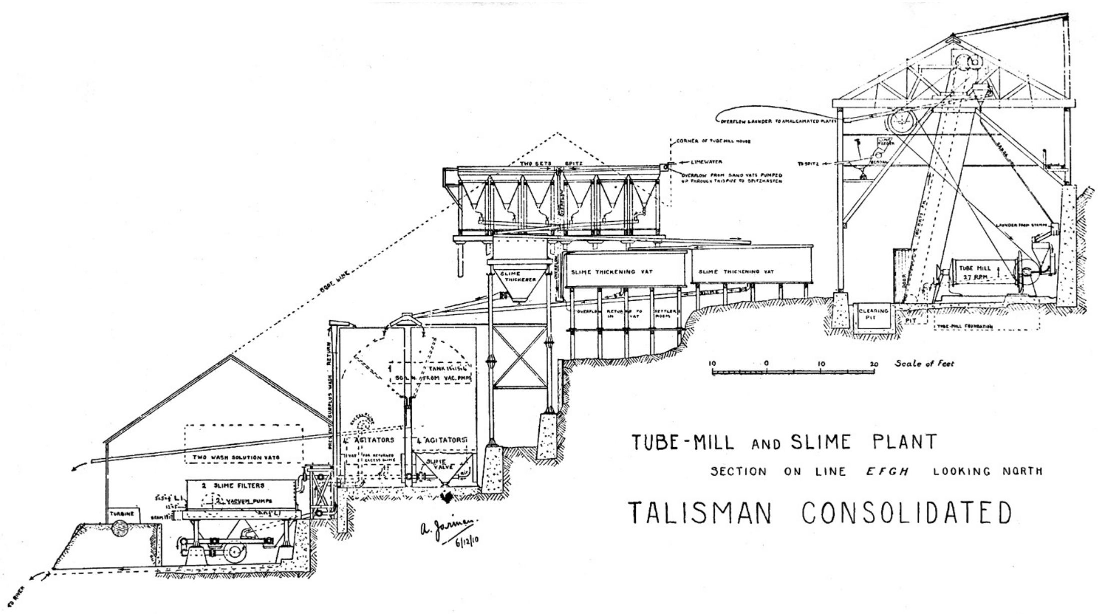


Fig.5
Section through Mill and Sand Plant on line A, B, C, D Looking South.
Talisman Consolidated G. M. Co. Ltd.



Tube Mill & Slime Plant.
Talisman Consolidated.
Fig. 6.

Tube Mills.

The installation of three tube mills has eased the work of the stamps and enabled them to be kept in better order than formerly without effecting repairs on Sunday, the output being about the same as before but more finely ground.

Grading Tests Before and After Installing Tube Mills.

Mesh		**May, 1910 before tube mills.**	**November, 1910 Tube mills started September, 12th.**
On	30	0.08	Nil
On	40	3.24	Nil
On	50	12.37	0.03
On	60	6.63	0.30
On	80	14.74	10.45
On	90	9.68	
On	100	0.67	6.32
Through	100	52.56	
On	150		12.46
On	200		13.19
Through	200		57.24

Table D

N.B.- The screens used in these tests were not of special manufacture.

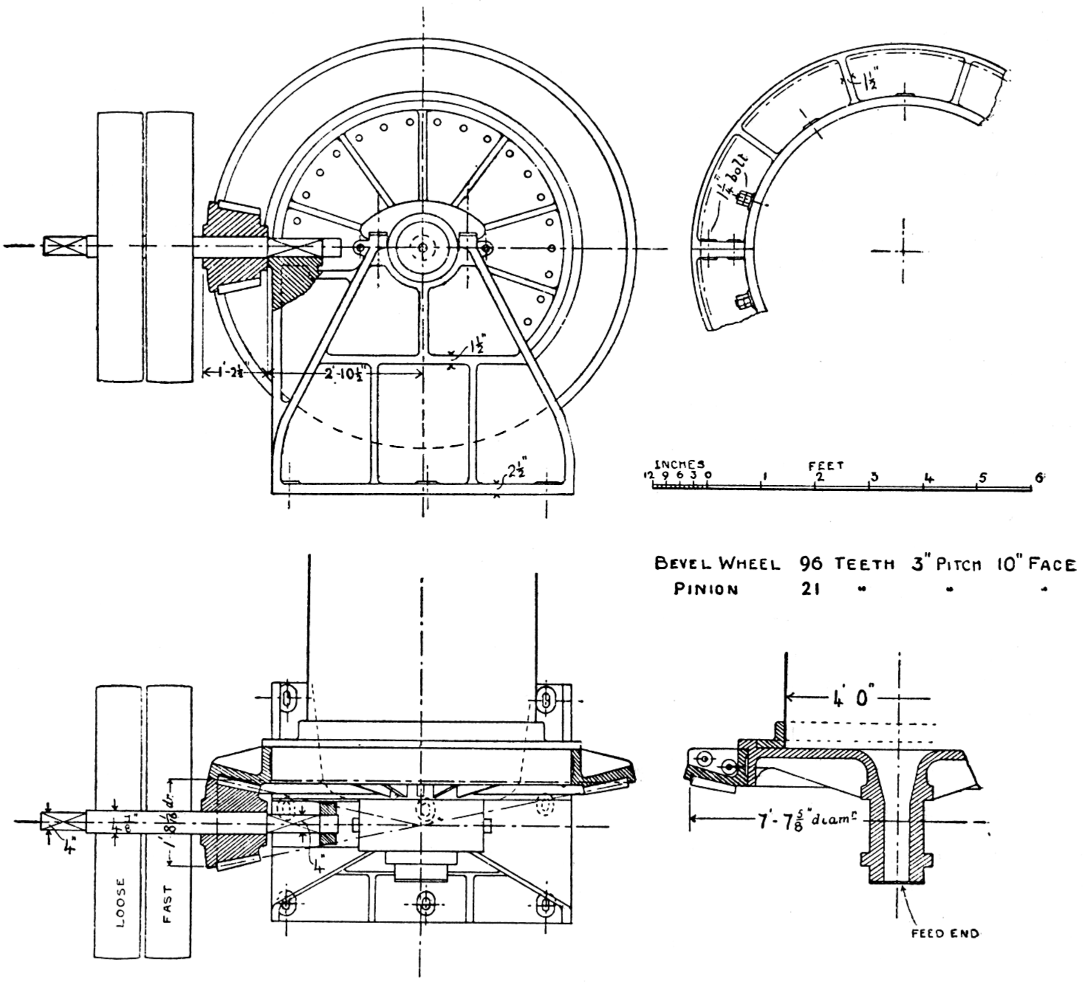


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FIG. 8.

Tube Mill Driving Gear. Talisman Consolidated. Fig. 8

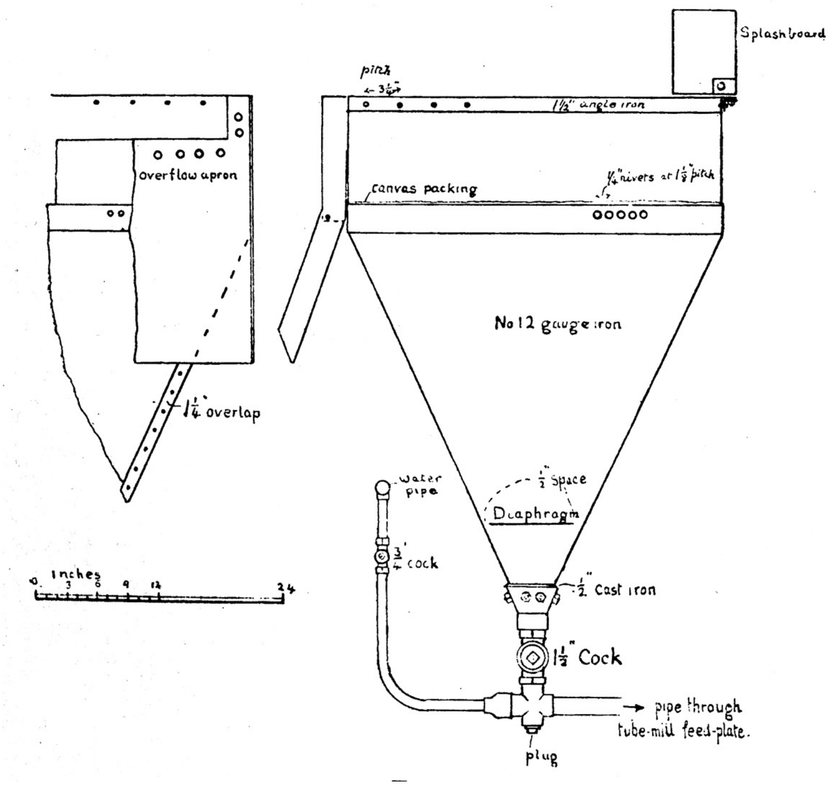
When first erected the tube mills were run by friction rollers carrying the body of the tubes on tyres. The wear was excessive and the present drive (Fig. 8), as arranged by Mr. J. A. Turnbull, mechanical engineer to the company, was substituted in September, 1910, since when there has not been the slightest trouble. The teeth of this drive on both pinions are of ample size (10 inch face) for the power to be transmitted. The failure of bevel driving-gear is generally due to the teeth being to short. The diameters at the bases of the cones are 1 foot 8 inches and 7 foot 8 inches, a ratio of 5 to 23, and the mill starts easily when the belt is only one quarter of its width on the driving pulley. The mills are run at 26 to 27 r. p. m. The shells are ½ inch boiler plate and are 12 feet long by 4 foot internal diameter. Two are lined with H. P. Barry's patent sectional honeycomb liner and one with F. C. Brown's iron plate and rib liner, the object being to compare the two when running side by side upon the same product, no strictly comparative tests having yet been published.

So far as the mill-man is concerned, the cast-iron liners are the best, for should a rib or a plate break or loosen, a small leak starts promptly at the bolt holes, thus giving timely warning that something is wrong. The ribs are the first to give way and when this occurs the shell of the mill is still protected by the lining plates. With the Barry liner the contents of a quarter section may be dislodged without the defect being discovered until the mill is opened up for examination.

Further, in replacing or renewing the lining, time must be allowed for cement to set thoroughly. E. g. 3 or 4 days when ordinary cement is used. With ribbed liners a new plate or rib can be put in in half an hour or less.

The arrangements in the tube-mill house are as follow- the pulp from the battery passes to a launder behind the stamps and so to the tube-mill house. A ¼ inch screen, intercepts stray coarse material which is removed occasionally by the attendant, the coarse sands from the elevator classifiers join the pulp and after passing another ¼ inch mesh screen, the launder is divided into three equal sections feeding three square pyramidal diaphragm classifiers of the form and size shown in Fig. 9. The diaphragm is inserted near the bottom of each pyramid leaving a ½ inch space all round, as advocated by Caldecott, the object being to ensure an even thick pulp. A water supply is arranged at the throat for use when the pulp packs. Four tests were done to determine the thickness of the feed, the Brown and one Barry mill being tested simultaneously. The percentages of dry ore present were 55 and 49.5% for the first and 50 and 45% for the second mills, so that the pulp is practically 1 to 1, and fairly even in character.

The tailings from the tube mills are joined by the overflow from the three classifiers and then elevated by two bucket elevators. The buckets are 0.2 cubic feet capacity and run at 160 p. m. A clearing pit 6 foot x 6 foot x 5 foot is provided to each elevator and if the boot chokes up, the door communicating with the pit is opened so that its contents can be run out, and the elevator restarted. The pit can be cleared afterwards. Each elevator delivers its pulp to a classifier of the same design as that shown in Fig. 9. The overflow from these passes by launder to the plates in the battery house and the coarse sand returns to the tube-mill feed-launder.



Classifier. Fig. 9

Grading Tests of Tube Mill Products.

There was no arrangement to ensure that the pulp passing to each mill should be of the same thickness, and the quantity fed could only be adjusted by altering the launder divisions. However, it has been shown that the thickness of pulp does not vary greatly and the feed was divided so that the two mills appeared to get equal quantities.

Sampling was done by passing a square tin across the stream of pulp from one side to the other. The sand adhering to the outside of the tin was rinsed off and the contents then transferred to a sample bin of about 2 cubic foot capacity, only the water of the sample itself being used for rinsing out the inside of the tin. The thickness of feed was determined by measuring and syphoning off the clear water from the tailing sample and weighing the remainder before and after drying. It took just three hours to take 19 cuts from each stream, viz., feed and tailings from one Barry and the Brown mill. The weights of dry ore obtained represent from 1/280th to 1/475th of the pulp flowing through one mill during the time of sampling. The samples were allowed to settle for 20 hours and the clear water syphoned off, the remaining material being transferred to large pans and dried in an oven. The dried samples were rubbed through a 10 mesh sieve and reduced to 1/8th of the original bulk by riffles. After transport to Auckland, they were re-dried, mixed thoroughly, riffled down to about 300 grm., and then graded by sieving with standard I. M. M. screens. This sieving was performed dry until apparently finished, and the process was then completed with the aid of water. The results are given in Table E in percentages.

The amount of grinding effected by the tube mills is seen by comparing the grade of the battery pulp (screen samples) with that of the fine pulp returned from the tube-mill house (plate heads) see Table E, remembering that about 160 long tons are crushed per 24 hours. These screen and plate samples were taken over a space of 2½ hours and consist of 11 cuts across the stream at regular intervals. The dry weights of the samples only represent 1/2740th and 1/4180th respectively of the total flow of pulp in this time, but the cuts could not be made more rapidly without risk of bad work.

Grading Tests.

Mesh I.M.M. Screens.	**"Screen Sample."**	**"Plate Heads."**	**Feed.**		**Tails.**	
			Barry.	**Brown.**	**Barry.**	**Brown.**
20	7.6	0.02	2.0	2.6	0.36	0.42
40	17.8	0.18	12.3	13.4	7.9	9.0
60	11.4	2.2	24.6	25.7	25.3	25.9
80	8.8	9.6	26.6	28.6	31.0	32.7
100	5.4	11.8	9.6	9.1	9.9	9.1
120	3.0	5.8	3.6	3.1	4.0	3.9
150	3.0	5.6	2.8	2.1	3.2	2.5
Through 150	42.7	64.3	18.2	14.7	17.5	16.2
Totals	99.7	99.5	99.7	99.3	99.16	99.72

Table E.

Amalgamation.

The pulp from the tube mills is elevated to two classifiers (Fig. 9), the overflow from which passes to the amalgamated plates, water being added en route. There are ten plates, each 4 foot 9 inches by 12 foot long with 1 1/8th inch drop in the centre. The fall of the plates is 1 3/8th inches per foot, and at the end of the plates there are two riffles 3½ inches wide by ½ inch deep.

About 28lb. of mercury is used every morning in cleaning and dressing the plates and the loss sustained is 0.25 ounces per ton of ore crushed. It is still too early to say what the effect of fine grinding will be upon the percentage recovery by amalgamation. The tube mills started work on the 12th September, 1910, and a fire on the 15th September put two compressors out of service and consequently stopped the pumps below No. 13 level. Rich ore from development work was therefore cut off and the tonnage had to be maintained from above No. 13 level. The amalgamation and cyanide returns immediately before and after starting the tube mills are therefore not comparable. In addition to this the ore varies so much in nature and in its gold-silver ratio that it would be necessary to take the average over periods of at least six months in order to get reliable figures.

Percentage Recovery of Values by Amalgamation.

Year	%	
1906	37.3	
1907	39.1	
1908	40.1	
1909	40.7	
1910	40.3	for 6 months prior to starting tube mills.
1910	31.0	for 3 months after starting tube mills.

The reduction of the recovery by 10% since September is probably due to the alteration in the character of ore treated. No amalgam traps are used, but each Vanner has a muntz-metal box inside its feed-distributing box, which catches mercury, becomes coated with amalgam and is cleaned up every six months.

Concentration.

From the plates the pulp passes to:-

8 Union vanners 4 foot wide
11 Union vanners 6 foot wide and
6 Frue vanners 6 foot wide.

There is no pulp distributing box but the vanners are ample for the work to be done, about 2 tons of concentrates being produced per day. The running conditions are as follows:-

4 foot Union 180 r.p.m. or side-shakes and 2.5 foot belt-travel per minute.
6 foot Union 194 r.p.m. or side-shakes and 3.3 foot belt-travel per minute.
6 Frue 194 r.p.m. or side-shakes and 4.5 foot belt-travel per minute.
And the inclination of the belt is 3 inches in 12 feet.

There is one attendant per shift and in addition a fitter is engaged in repair work on the day shift. Each shift clears the concentrate from one row of vanners. The concentrate in the storage bin is drawn out twice per week and railed to a drying pan (8 foot x 2 foot, 8 inches) under which a slow fire is maintained. Next day it is dry enough to be bagged, sampled by a sampling iron, and weighed ready for dispatch to the smelter.

The weight and assay value of concentrate produced varies greatly with the class of ore treated. It averages 1.25% by weight and 15% to 18% of the gross value of the ore treated, assaying about £85 per long ton, the ratio of gold to silver in the bullion varying from 1:3 to 1:30.

Cyaniding the Sands.

The tailings from the vanners pass to 13 steel cyanide vats, 22 feet 6 inches diameter, 10 being 10 foot deep and 3 of them 8 foot deep. Butters and Mein's distributors are employed, the vat first being filled with water. The overflowing slime is pumped to spitzkasten by a Tangye 4 inch centrifugal pump having a capacity of 200 gallons per minute, a second pump being kept in reserve. A slotted tube "rod sample" is taken during filling, and the assay of this should be comparable with the corresponding "plate head" sample. The treatment for any vat is not posted until these assays are to hand - the ore being variable in grade and silver-gold ratio.

In consequence of the high silver values, strong solutions are used, the bulk being kept low, about 0.35 ton per ton of ore during contact, and about 1 ton of wash solutions used afterwards. The general outline of the method of treatment adopted by Mr. Deane for a charge of 140 tons of average grade is as follows:-

1. Allow to drain completely.
2. Displace the retained water by an alkaline wash of 4 tons medium solution + 200 lb normal soda carbonate (soda ash).
3. Continue displacement by adding the working solution of 8 tons strong solution + 270lb. cyanide (NaCN), following this by repeated lots of 10 tons of strong solution, until the effluent shows 1.1% total cyanides (equal to 0.7% free KCN).
4. Close the percolation cock and start the injector, i.e., a small air-jet at the bottom of a vertical stand-pipe leading from under the filter. This little air-lift raises the solution to the top of the vat and keeps it in continuous circulation during "contact."
5. After 4 or 5 days "on contact" a rod sample is taken and generally shows the values to be sufficiently reduced to proceed.
6. Drain off the strong solution, and then wash by 5 tons medium solution, followed by 80 to 120 tons weak solution.
7. Drain well, take a rod sample and sluice the residues out into river, if value is satisfactory. When the vat has been half emptied a check sample is taken from top to bottom.

If the stock of week solution is small, a water wash is given to displace the last of it, but not otherwise. The effluent solutions are tested by titration with silver nitrate in the presence of KOH and KI indicator, the total cyanide indicated being expressed in terms of KCN. (Potassium cyanide):-

Strong solution		1.0 to 1.5%
Medium	"	0.85%
Weak	"	0.3%

When titrated without the indicator, the end point is very difficult to determine but the amount of free cyanide is approximately:-

Strong solution		0.5 to 0.7%
Medium	"	0.4 to 0.5%
Weak	"	0.2%

Owing to the variable nature of the silver contents of the ore, the strength of solution used for "contact" has to be varied, and the following examples give a good idea of the practice on rich, medium and poor ore since tube-milling started. The solutions formerly used were stronger, but finer crushing has had the effect of simplifying the treatment. The solutions are cleaner and work more effectively so that less cyanide is required.

Charge Long Tons	DWT. Gold		Gold Extn %	Oz. Silver		Silver Extn %	Lb NaCN added	NaCN Lb. Per long ton	=KCN Lb. Per short ton
	Before	After		Before	After				
140	21.21	1.21	94.5	10.617	1.694	84.1	320	2.28	2.65
138	11.417	0.792	93.1	5.492	0.612	89.1	270	1.95	2.27
145	4.04	0.417	89.7	2.414	0.344	85.8	130	0.90	1.04

Table F.

When a vat filters slowly, washing and draining are assisted by vacuum. There are two pumps with cylinder 6 inches diameter by 6 inch stroke running at 40 r.p.m. The vacuum solutions go to a receiver and then to a settler before passing to the zinc boxes.

Normal soda carbonate (soda ash) was adopted for alkali wash after trials had been made with it and lime, the former being the more efficient. It is preferred to caustic soda, as being less objectionable to handle.

The solutions pass to settling tanks and thence to six 6-compartment zinc-boxes, each having 360 cubic feet zinc space. Three of these are used for week solutions showing from a trace up to 0.3% total cyanide, one for medium solutions with 0.3 to 1.0% and two for strong solutions with more than 1.0%. After passing the boxes the solutions show from a trace to 4 grains of gold and from 8 to 22 grains of silver per ton.

During the author's stay at Karangahake, Mr. Deane gave one vat special treatment to illustrate the necessity for strong solutions to attack the silver. The charge was in a vat 8 feet deep and consisted of 106 long tons, assaying 16.33 dwt. Gold and 8.02 ounces silver.

After draining and sampling, percolation was begun with weak solution, the effluent showing 0.2% free cyanide. After 24 hours the sand was sampled, and showed an extraction of 27.5% of the gold and 4.3% of silver. Week solution was continued for another day and then the vat was drained, having had 58 tons of week solution. The effect of 0.4% solution was next tried by running on weak solution in 10 ton lots, to each of which was added 38.5lb. of sodium cyanide. When the effluent showed 0.4% free cyanide (KCN) "contact" was established, there being 53 tons of solution. The sand was sampled each day, and after three days contact the gold extraction had risen to 90%, but the silver only to 30.8%, the residue being worth 11 shillings per ton in silver alone. The free cyanide had fallen to 0.36%, and to raise it to 0.5% an additional 126lb. of cyanide was therefore added (21lb. every 4 hours). After three more days the extractions were, gold 92.5%, and silver, 67.6%. The vat was then drained, washed with 30 tons strong solution, showing 0.49% free (0.96% total) cyanide, then 5 tons medium and finally 160 tons weak solution. After this it was drained, sampled and discharged, the residues carrying 0.79 dwt gold and 1.63 ounces silver, indicating extractions of 95% of the gold and 79.5% of the silver. The total cyanide added was 192.5 + 126= 318.5lb., and if the whole of this had been added to the contact solution in the first instance better extractions would have resulted.

Treatment of Slime.

The overflow from the sand vat is pumped up to two sets of spitzkasten, each having nine pyramidal hoppers 6 foot x 6 foot square and 6 foot deep, with 1 foot 6 inch depth above the hoppers. To facilitate settling, an addition of limewater is made. The

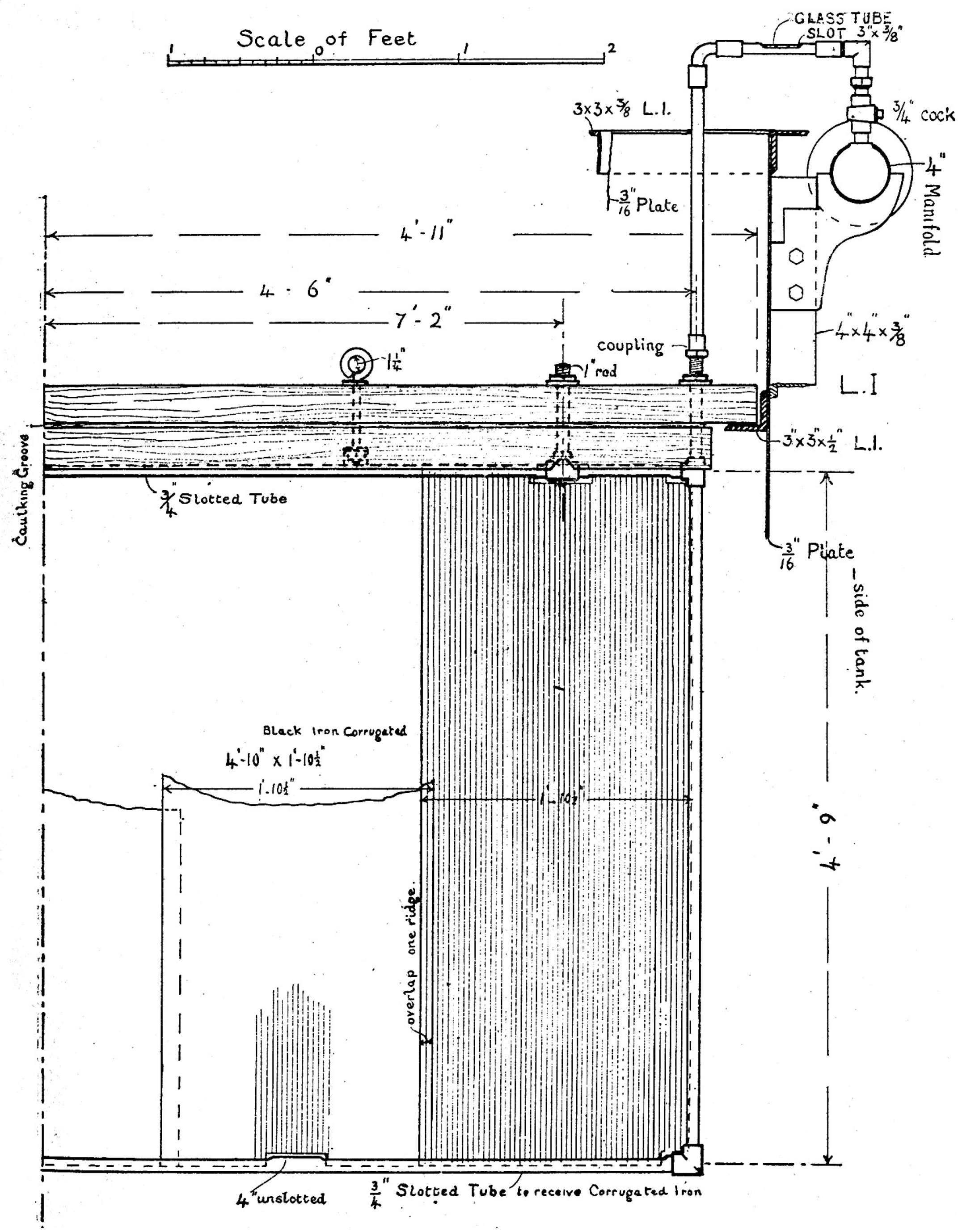


Fig. 10
Showing Details of Construction of Filter Frame.

lime is fed by a Challenge feeder to a Berdan pan, the water from which flows into the top end of the feed-distributing launder. About 10 tons of lime are used per month for 650 tons of slime (dry weight). The underflow from the spitz. passes to thickening vats, the first of which is a steel tank of the same shape and dimensions as the filter-frame tanks, viz. 20foot x 10 foot x 7foot deep, with two pyramidal hoppers at the bottom, 10 foot x 10 foot x 7foot deep.

The overflow from this passes to two settling vats 22 foot diameter and 7 foot deep, from which the overflow passes to two similar vats in the sand-treatment room. The overflow from the latter passes out to the river.

When slime is required for the tall agitator tanks, it is drawn from the steel tank and first two circular vats. The slime caught in the last two vats is drawn off twice a week and pumped to the agitators by a Gwyne 4 inch centrifugal pump of 200 gallon per minute capacity.

There are eight tall air-agitator tanks 12 feet in diameter and 30 feet high, the conical bottom occupying the bottom 5 foot 6 inch. Air is supplied near the base by two ½ inch pipes connected with a main carrying air at 40lb. pressure. These tanks have no central tube as they are unnecessary for agitation. The procedure is as follows- the tank is filled to 18 inches from the top with slime, which is allowed to settle for an hour or more and any clear water then decanted off, the amount varying from nil to 5 foot depth according to the thickness of slime available. When full to 18 inches from the top, the charge is about 120 long tons of 1 to 1 slime, the dry slime assaying about 2/3rd of the value of the sand from which it was derived. Five tons of week cyanide solution are run in (0.18 to 2.0% free KCN) through a wire basket containing from 200 to 250lb. sodium cyanide, thus making up 65 tons of 1.19 to 0.35% KCN. After 24 hours agitation the pulp is ready for filtration.

There are two filters of the fixed frame type, each set consisting of 31 leaves, each 9 foot x 4 foot 9 inches, equal to 2,650 square feet of filtering surface. The frames were spaced 7½ inches between centres, but 8 inches is found to be more convenient. They are connected at each end to a 4 inch manifold leading to the vacuum pump. A gauge glass is fitted in the connections to one of these manifolds so that a defective cloth is readily noticed. Any leaf may be disconnected and removed without interfering with the rest.

The details of the frame are shown in Figs. 10 and 11. Black corrugated iron of 1 inch corrugations forms the support for the cloth and is held by a frame formed of ¾ inch piping, having a ½ inch slot cut longitudinally to receive it. The leaf is 9 foot long and the supporting tube at the bottom sags slightly in the middle, but an extra pipe placed vertically in the centre to support the bottom tube would prevent this sag. This addition will be made in new frames. The tanks containing the filter frames are 20 foot x 10 foot and 7 foot deep and are of 3/16th inch steel plate stiffened by vertical and diagonal lengths of angle iron 4 inches x 4 inches x 3/8th inch. The bottom consists of two square pyramidal hoppers 7 feet deep, formed of ¼ inch steel plate and stiffened with a horizontal rib of 3 inch x 3 inch x ½ inch angle iron as shown in Fig. 10. The weight of the tank is supported by a 5 inch x 5 inch x ½ inch angle iron riveted on the sides and resting on an 18 inch x 7 inch H girder supported on cast-iron columns. At the ends of the tank similar angle iron rests upon 12 inch x 5 inch H girder, the ends of which are supported by the ends of the 18 inch x 7 inch girders (see section, Fig. 6).

The capacity of the filter tank is a little more than half that of an agitator. Slime from an agitator is run in by a 12 inch pipe, and is prevented from impinging on the filters by a 3/16th inch baffle plate. A full charge comes up to 6 inches from the top of the tank. Agitation is effected by compressed air, admitted from ½ inch pipes - two near the bottom of each hopper.

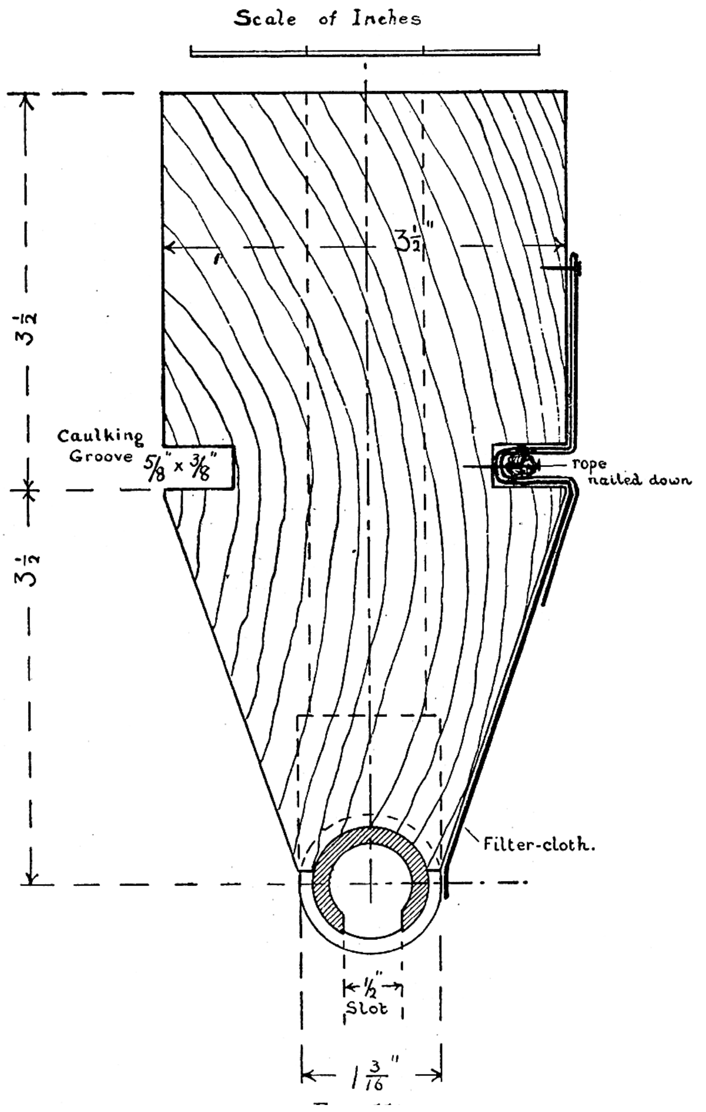


Fig. 11
Showing Details of Construction of Filter Frame.

There are two vacuum pumps, designed by Mr. J. A. Turnbull, to work to a net lift of 40 feet. They are wet, double-acting, belt-driven, and either side of the cylinder can be disconnected. The cylinder is 8 inches in diameter x 18 inches stroke, and at 50 r. p. m. a vacuum of 23 to 25 inch of mercury is maintained.

An hour at this vacuum suffices to build up a ¾ inch to 1¼ inch cake when the cloths are new, and a 1 inch cake is equivalent to about 10 tons of dry slime. Old cloths work slowly owing to the formation of scale. During the formation of the cake the solution is pumped to a 15 foot x 15 foot x 4 foot settling tank, from which it goes to the weak-solution settler, prior to entering the zinc boxes. The solution runs from 0.18 to 0.2% KCN, and goes into stock as the weak solution for all work.

The slime remaining in the tank after the cake is formed is pumped out into a storage puddler-vat, 22 feet in diameter x 5 foot deep, (shown by dotted lines in Fig. 6). Weak-wash solution is run in from one of the two weak-wash vats, 22 feet in diameter x 7 feet deep, (shown by dotted lines in Fig. 6), and washing is continued for 1½ to 2 hours according to the grade of the material under treatment. The remaining wash solution is then pumped back to its tank and the vacuum continued for another 15 minutes to drain the cake as completely as possible, when it carries 18% to 25% of solution. During this 15 minutes the sluicing gates at the bottom of the hoppers are opened and the cake is sampled by taking a grab from one side of every other leaf. Vacuum is then released and the cake assisted to fall by the use of a wooden spatula and by hosing. This residue, which assays about 5 shillings per ton, is sluiced into the river. Discharging takes little more than 10 minutes with new cloths, but old cloths require an hour or more. For the pumping of slime and weak solution, two 8 inch Kershaw centrifugal pumps are used, capacity 1,400 gallons per minute, made by Thompson, Castlemaine, Victoria, Australia.

Clean Up.

Amalgam is retorted and melted in the usual manner, the grade of the resulting bullion being 357 to 535 fine in gold, and 10 to 15 parts of base metal.

In the cyanide work, the first and second compartments of the strong boxes are cleaned up in the middle of the month and a complete clean up of all the boxes is made at the end of the month. The rich slime goes straight to the vacuum filter boxes, and the other slime is washed on a 30 mesh sieve to remove shreds of zinc before being filtered. After draining under vacuum, the slime is dried and oxidized at a dull-red heat, the floor of the furnace being an iron plate heated from below. The roasted slime is cooled and mixed with soda carb. (soda ash), 6%, crushed borax glass, 30%, and fluor, 1%, melted in two tilting furnaces and poured into conical moulds. The bullion is usually free from mate. It is re-melted in crucible furnaces with a little borax, the slag is skimmed of (thickened by a little bone-ash if necessary) and the bullion cast into bars of approximately 1,300 ounces. The fineness is from 52 to 146 in gold according to the ore treated, and from 50 to 60 of base metal. The bars are about 4½ inches thick and are sampled by drilling two holes, one at the top and 1/6th of the length from one end, and the other at the bottom and a similar distance from the other end. The holes are drilled 1½ inches deep and the outer crust is discarded. The clean-up slag is broken up and then crushed without amalgamation in a 2 head (Union Iron Works) stamp mill of 400lb. head. The pulp goes to a Berdan pan and from there to settling pits. The amalgam from the Berdan, and shot metal from the mortar box, are returned to be smelted and the crushed slag is dried and shipped to the smelter, being worth about £30 per ton.

Water Power.

At the dam on the Ohinemuri river the summer flow is 25 sluice heads (cubic feet per second) and provides 320 h. p. by means of a 10 foot 6 inch pelton wheel. The pipe line is 1,800 feet long and 4 foot in diameter and the fall 80 feet. The dam in the Waitawheta gorge has a summer flow of 21.3 sluice heads and supplies water to a 100 h. p. Victor turbine which drives the lighting dynamos, about 46 h. p.

Labour Employed in the Mill.

	Morning	Afternoon	Night
Breakers and elevator	1	1	
Stamps feeder floor	1	1	1
Stamps (shift boss)	1	1	1
Vanners	1	1	1
Vanners repairer	1		
Concentrates, drying, bagging & Zinc boxes	2		
Vat floor (Sand cyaniding)	1	1	1
Slime tanks and spitz.	1	1	1
Filters	1	1	1
Tube mills	1	1	1
Extra hand, screens, sluicing &c.	1		
Engine-driver	1	1	1

Water Supply.

The main from this dam also supplies water to the mill, a feed pump, capable of dealing with 16,000 gallons per hour, forcing it to the two supply tanks at the top of the mill. The total water used is about 9 tons per ton of ore crushed.

Steam Power.

There are four B. & W. boilers with 1,619 square feet of heating surface and fitted with chain-grate mechanical stokers. Steam from these boilers is supplied to the mill engine by a pipe line about 660 feet long. This engine is a compound-tandem Corliss engine of 560 h. p. by Messrs Fraser & Chalmers. The cylinders are 18 inches and 34 inches in diameter with 42 inch stroke. It runs at 72 r. p. m. and drives the main shaft by 12 ropes 2 1/8th inches in diameter.

Compressed Air.

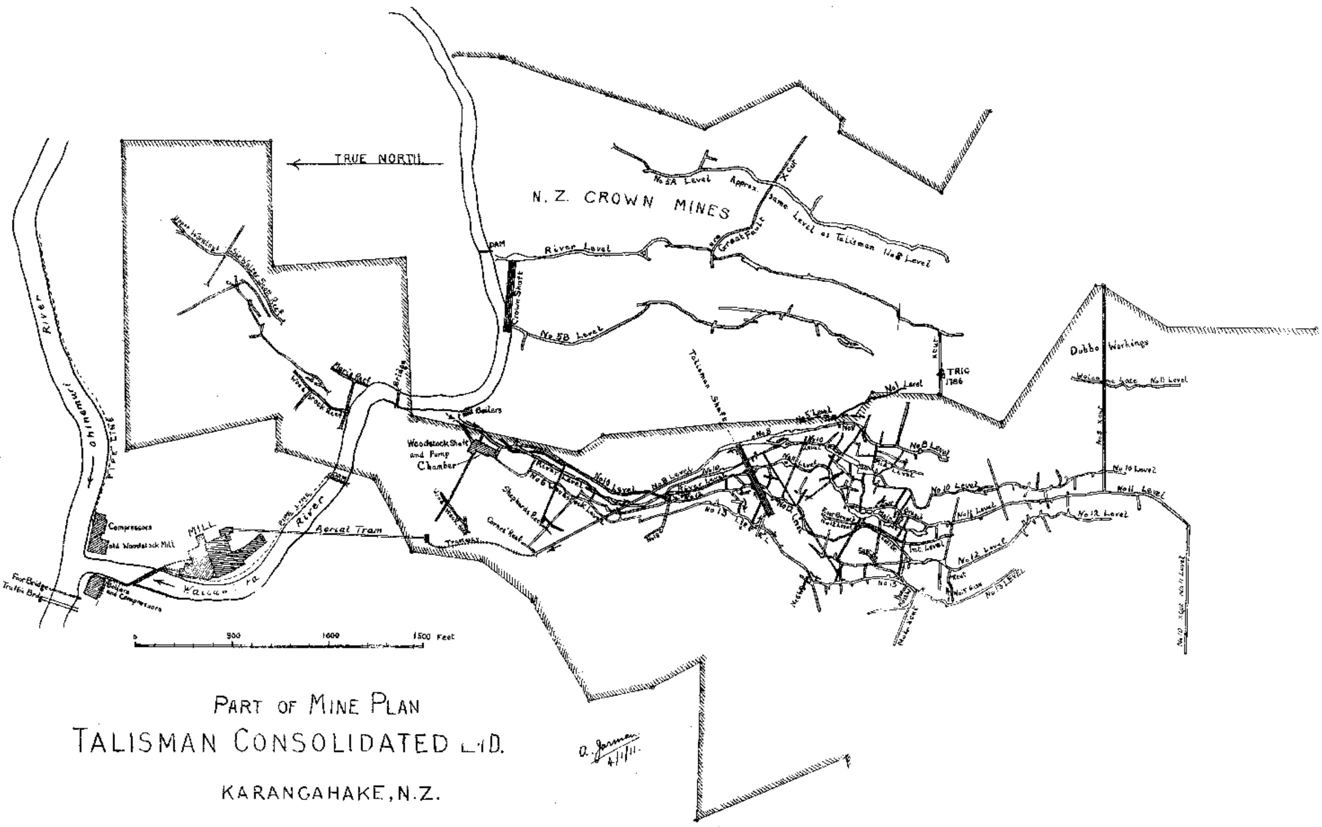
The B. & W. boilers also supply steam to a Riedler air-compressor having a rated capacity of 3,100 cubic feet of free air p. m., and the blow-off valve of the receiver is set at 100lb. per square inch. An Ingersoll Rand compressor of a capacity of 3,700 cubic feet p. m., has been ordered and will be installed alongside it. The power from the Ohinemuri dam is utilized to drive an Ingersoll Rand compressor giving 1,250 cubic feet free air p. m. at 90lb. pressure and also a Union Iron Works Co machine giving 600 cubic feet free air p.m. at 40lb.

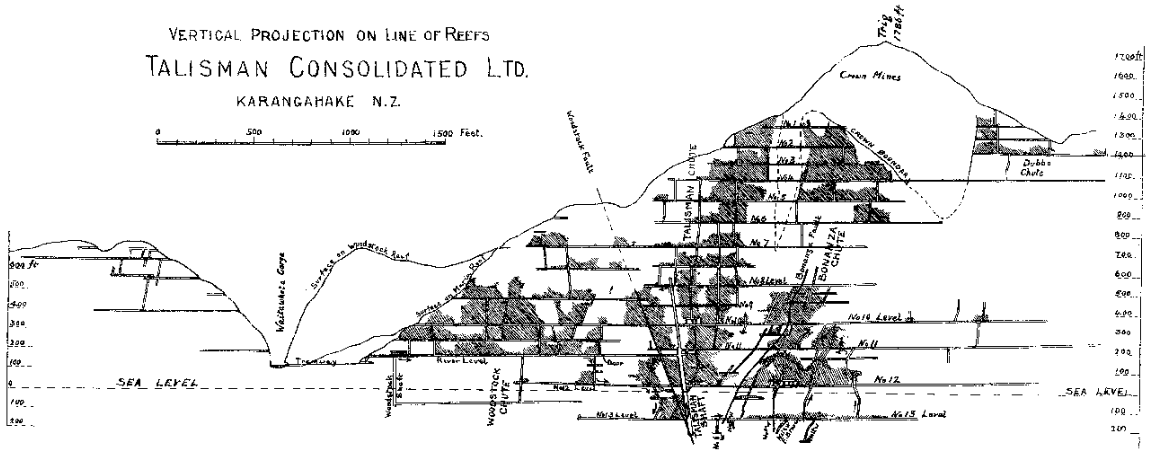
Costs.

A statement of costs is published each month together with the monthly return, e. g., on November 9th, 1910, "the costs for the past month totalled 35 shillings and 3 pence, made up as follows: Mine Development, 9 shillings; Mining, 11 shillings and 10 pence; Milling, 12 shillings and 4 pence; General Expenses, 2 shillings and 1 penny."

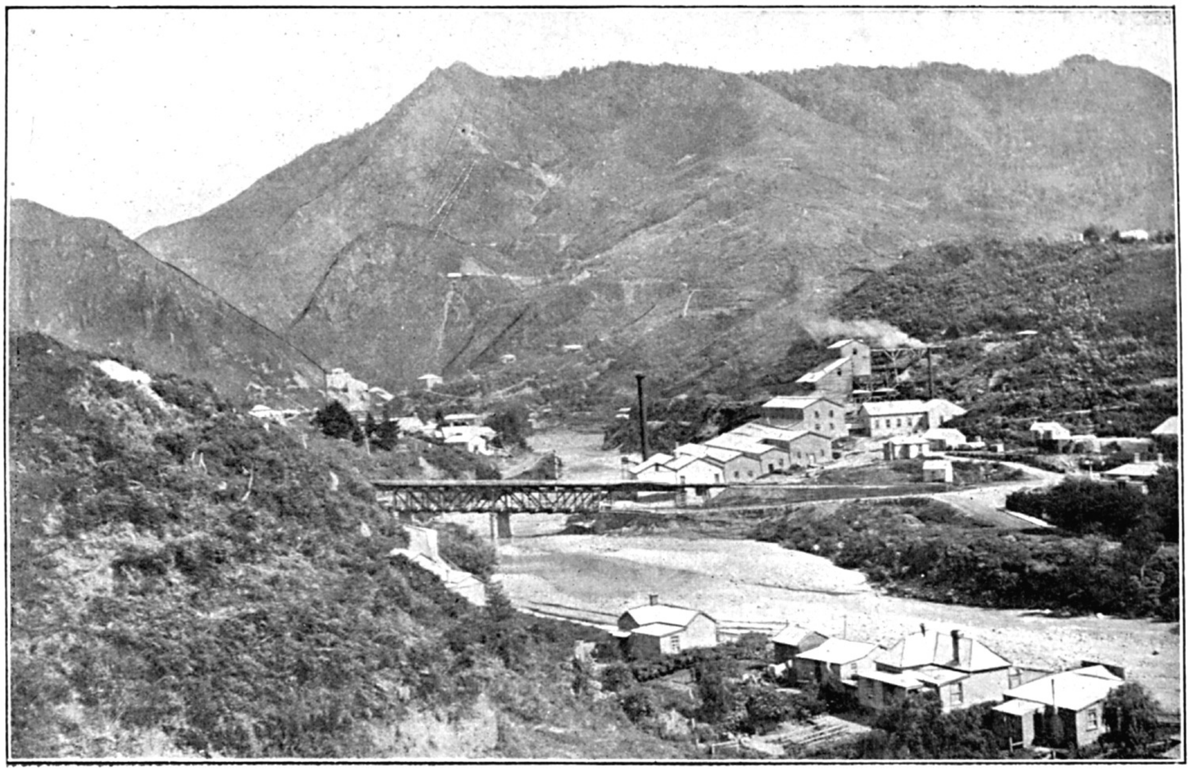
Conclusion.

The writer is much indebted to Mr. H. Stansfield (of Messrs. Bewick, Moreing & Co.), General Manager for the Co., Mr. D. M. Deane, Metallurgist, and other officers of the company, for particulars and information herein contained and for many courtesies extended to him whilst at Karangahake.

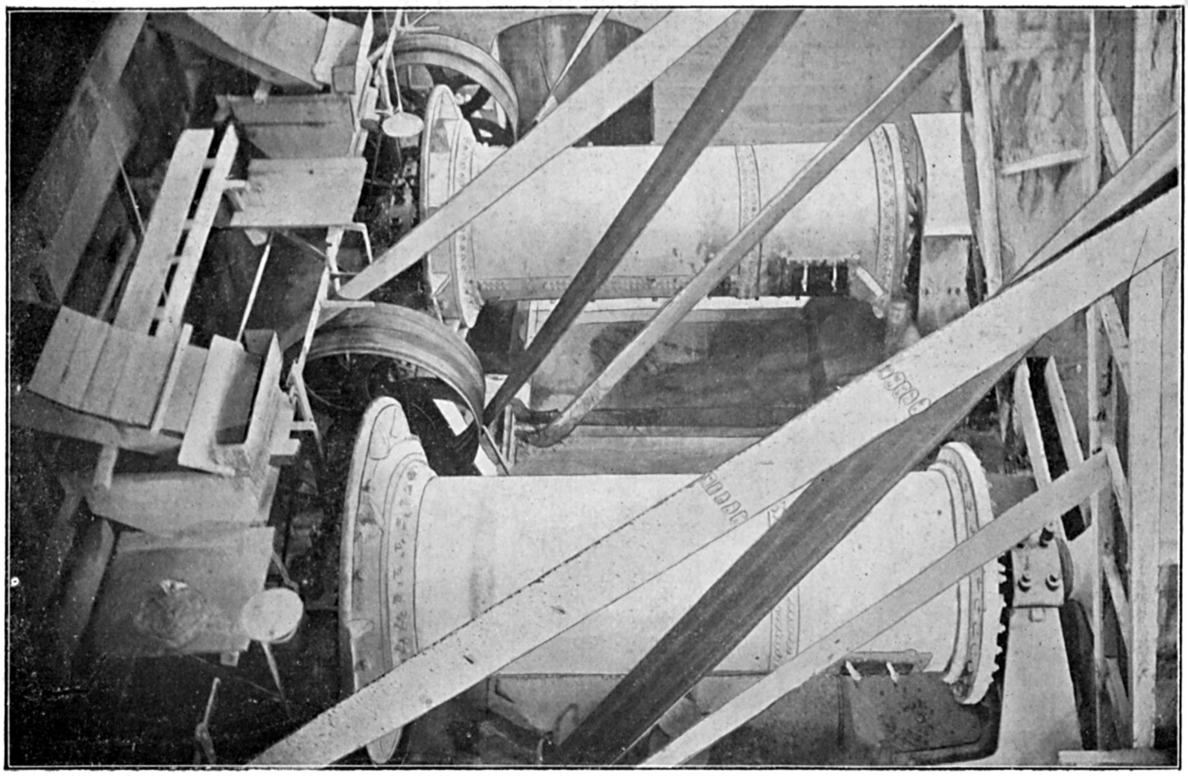




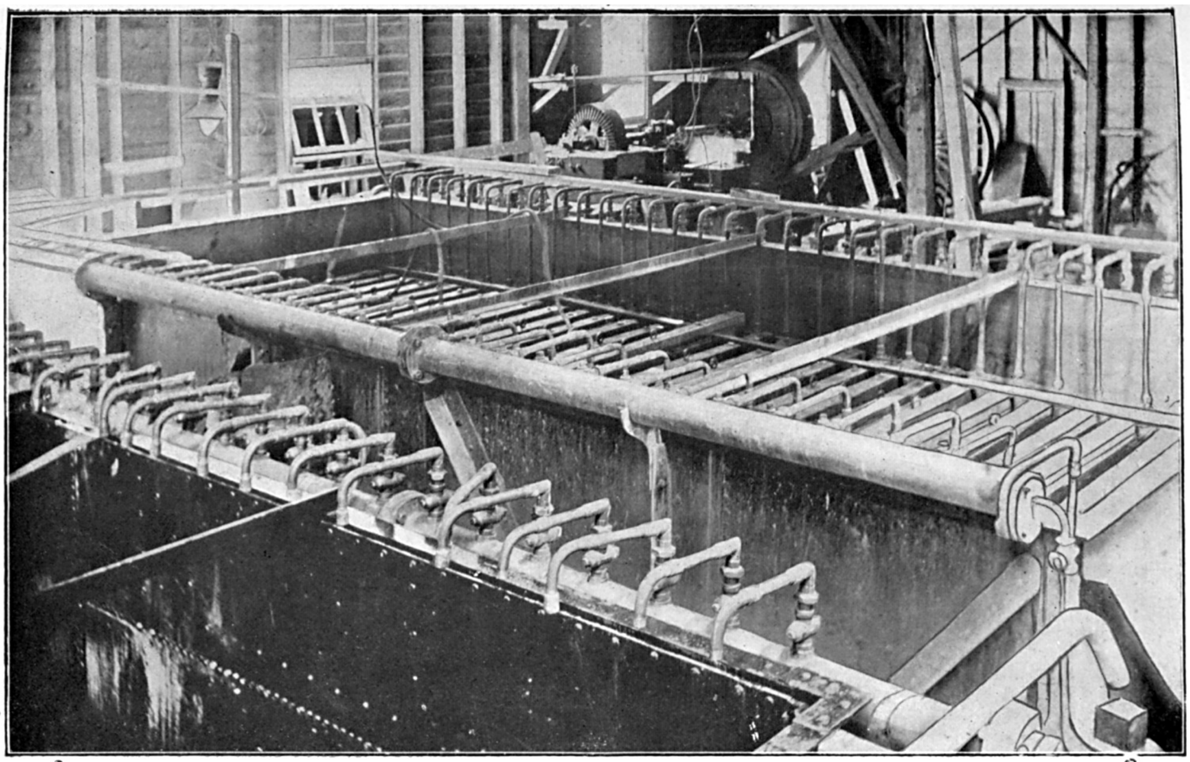
Mine Plan and Projection
Plate V



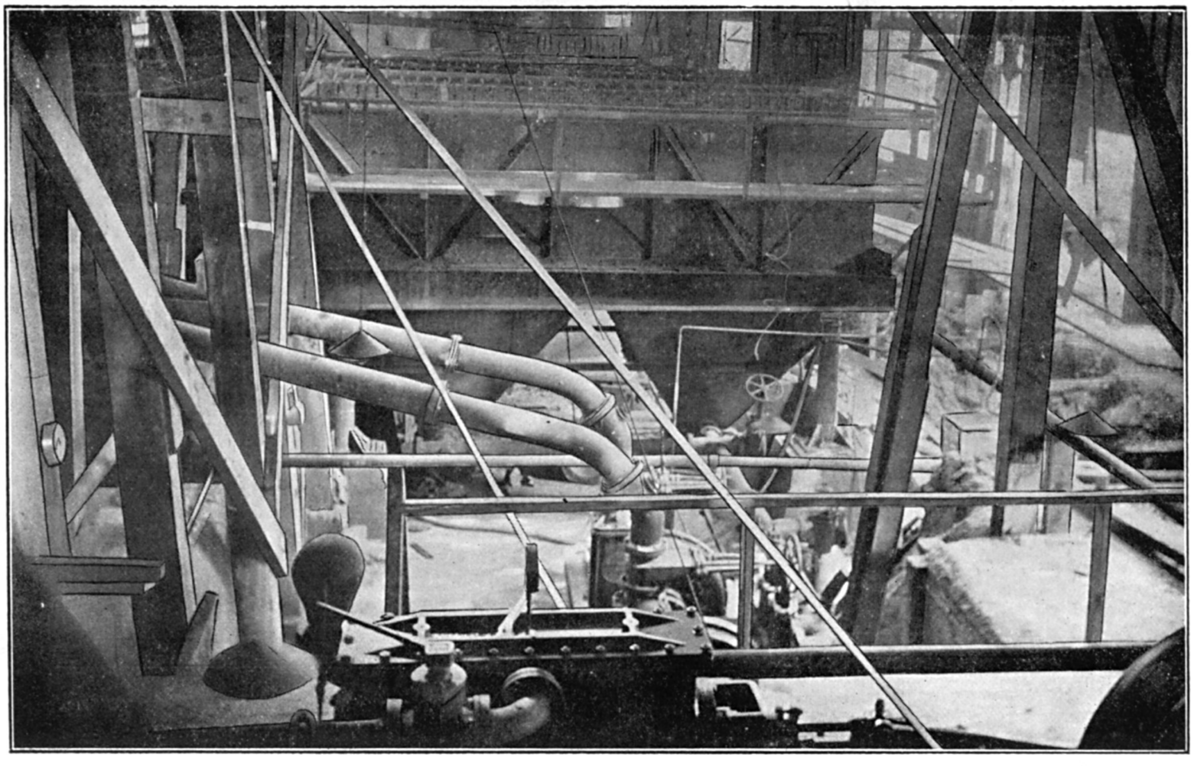
Karangahake.



Classifiers and Tube Mills.
Barry mill in foreground and Brown mill in background,



Fixed-Frame Slime Filter.
One under construction in foreground.



Slime Filter.
Vacuum pump in foreground.