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No - 213

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Dividends:

PAID TO DATE £1,030,000

Summary of Working Costs: Tons 14,537

	Total Cost £	s.	d.	Cost per Ton. s.	d.
Battery Working Expenses ...	2,444	6	7	3	4·355
Fine Grinding	229	3	2		3·783
Pumping	321	3	9		5·303
Sands Treatment	882	9	7	1	2·569
Slimes Treatmeut	2,495	6	0	3	5·197
Furnace Treatment	963	9	11	1	3·907
Stoping	6,628	0	11	9	1·426
Hauling	432	7	6		7·138
Trucking	650	1	1		10·732
General Working Expenses ...	297	16	3		4·917
General Administration Expenses	522	4	1		8·621
Mint and Bank Charges on Bullion	364	7	2		6·016
	16,230	16	0	22	3·964

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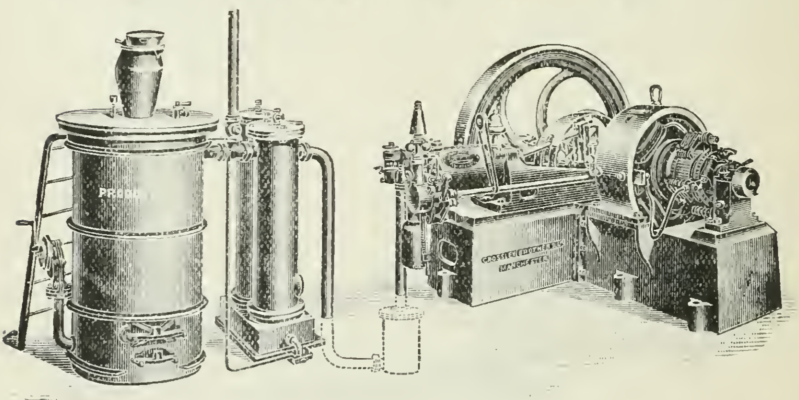
(See Descriptive Article, pages 397-401.)



The Australian Commonwealth Tariff and The "Australian Mining Standard."

Under the Australian tariff as first framed and brought before the House of Representatives, it was proposed to tax the items underwritten at the rates attached:—

Mining machinery	25 per cent.
Engines	25 per cent.
Boilers and pumps...	25 per cent.
Gas engines...	20 per cent.
Electrical machinery	20 per cent.
Electrical appliances	20 per cent.
Beams, girders and joists	20 per cent.

The **Australian Mining Standard** strenuously and steadfastly contended for a lower range of duties in each case, and its articles in this behalf were extensively copied by the goldfields press throughout the six States of the Commonwealth. When the tariff finally passed, the items named bore the subjoined imposts:—

Mining machinery	12½ per cent.
Engines	12½ per cent
Boilers and pumps	12½ per cent.
Gas engines	12½ per cent.
Electrical machinery	12½ per cent.
Electrical appliances	12½ per cent.
Beams, girders, and joists (iron or steel)	12½ per cent.

Herewith are furnished excerpts from a few of the speeches delivered by several of the prominent mining members in the House of Representatives, indicating the extent to which the arguments relied upon and information furnished by the "Mining Standard" influenced the course of the tariff debate.

Extract from the speech delivered by Mr. THOS. BROWN, member for Canoblas (New South Wales), during the tariff debate in the Australian House of Representatives, January 16th, 1902:—

If the mineral wealth is not here, the Government may impose a duty of 100 per cent. on mining machinery, and yet do very little towards developing the manufacture of the latter. The first thing necessary is to develop mining, and make a demand for machinery. I hold that the Government, in the method which they have adopted, are sacrificing the mining interest for the benefit of a few manufacturers, and that their conduct amounts to "killing the goose with the golden egg." I do not wish to labor this matter, which has been very ably dealt with by the leader of the Opposition and other honorable members. There are honorable members of the House who have practical mining knowledge gained in the different States, and who are in a position to lay before us valuable information. But I should like to refer honorable members to an article in the **Australian Mining Standard**, which is a very ably conducted journal, published specially in the interests of mining. The matter which finds expression in the columns of this journal may be regarded as different from that in the daily press, in so far as this particular publication represents the requirements of the mining industry, and is conducted by experts who thoroughly know the subjects with which they deal. The article takes the ground I have taken from the outset, namely, that the industry is natural to this country, and one to which the Commonwealth in no small degree owes its present position amongst the peoples of the world. . . . During the course of this debate a big point has been made of the great advance which has taken place in the manufacture of mining machinery in Victoria. It is claimed that this State is capable of competing with the most up-to-date machinery manufactured in any other part of the world, and that it can produce it as cheaply. If that be so, there can be no objection to the Tariff submitted. But the writer to whom I have already referred holds quite a different view. Speaking of Victoria, he says:—

> Its method of mining, of management, of reduction and extraction, are unfavorably criticised by every visiting expert.

Mr. Isaacs.—Who is the writer?

Mr. Brown.—I am quoting the opinion of the editor of the **Australian Mining Standard**, who says that the opinion of visiting experts is to the effect that Victorian mining, so far as the reduction and extraction of gold from ores is concerned, is not up to the standard which obtains in other parts of the world. He then goes on to show to what extent this Tariff will affect existing mining enterprises which are conducted upon a large scale, particularly in Western Australia. I have already shown to what degree it will affect the industry in New South Wales, and this expert authority sums up the position thus—

> As we have pointed out on more than one previous occasion, however it may apply to some others, the miner is a man who derives no particle of benefit from protection either directly or indirectly. He stands in dread of no competitor. He asks for no assistance at the expense of his neighbor. His product is an article the market price of which protection is powerless to advance, and it is one that must be exported before its value can be realised. If Australasian gold could not be exported it would be a worthless drug. Silver, copper, tin and lead are in the same case; yet protection anathematises export, and concerns itself only about local manufacture. Nevertheless, the miner, though he derives no benefit from protection, must pay its full cost in the advanced price of every dutiable import he uses, and every local manufacture it affects, and he is now called upon to do this at a time when his industry is fighting against a serious depression caused by the market decline in the value of products. Silver, copper, tin, lead, zinc, all are down, and only taxes are up.

Extract from the speech delivered by Mr. HENRY WILLIS, member for Robertson (New South Wales) during the tariff debate in the Australian House of Representatives, January 16, 1902:—

I suppose that honorable members would like to see our mineral resources developed, and that wealth won from the soil; but how can that come about unless we foster the mining industry? Honorable members who claim to be protectionists, and who are always speaking of fostering native industries, have an opportunity here We can foster the mining industry by making it possible for those engaged in it to obtain their machinery and other requirements at as low a price as possible. But in spite of the heavy decline in the selling price of metals, the Government propose to place a duty of 25 per cent. upon mining machinery. The honorable member for Barrier has told us that this duty is equal to increasing the cost of the output of the Barrier district by from 7s. 6d. to 10s. a ton. In Western Australia, where the output of gold has been even a little larger than that of New South Wales, there are mines which have been shut down altogether, because the companies cannot afford to pay for machinery the high prices created by this duty. The **Mining Standard** of 16th January makes a statement of the utmost importance as to the decline in the value of metals. In 1900 tin was selling at £153 per ton; to-day its value is £104 per ton. Copper has come down from £79 per ton to £45 10s.: lead from £18 per ton to £10 8s. 9d.; spelter from £22 per ton to £16 8s. 9d., and silver from 30 3-16d.to 25 11-16d. per oz.

Extract from the speech delivered by Mr. WILLIAM KNOX, member for Kooyong (Victoria), during the tariff debate in the Australian House of Representatives, April 9, 1902: –

Mining interests from one end of the world to the other are suffering very much from the depreciation in the prices of the various metals—copper, lead, and silver having declined so much that a profitable working margin is rapidly vanishing. In the case of Broken Hill, which was particularly referred to by the honorable member for Barrier, the services of large numbers of men have had to be dispensed with in consequence of the fall in the price of lead, and on that ground there is justification for asking the committee to give as favorable a duty as is consistent with the necessities of revenue. But there is another important point which was referred to earlier in the evening. The products of the mines have to compete with products from other parts of the world, all the silver, copper, and lead being sold abroad. That competition has to meet with cheaper labor and more complete appliances, and on that ground alone the mining industry deserves the fullest sympathy and help. It must be remembered that mining has contributed £500,000,000 to the wealth of Australasia. Of the total production of wealth in Australia in 1899, amounting to £137,000,000, mining produced £24,858,000, and manufactures £33,316,000. These figures show that mining produced nearly one-fourth of the primary production of Australasia, or was nearly equal to the production of agriculture, which amounted to £25,247,000. I am indebted for these figures to the **Australian Mining Standard**, the recent articles in which, on the mining industry as it is affected by the duties, are most powerfully written, and claim the attention of any honorable member who desires to study this subject for himself.

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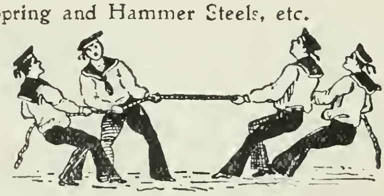

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TONNAGE TREATED TO DATE, 152,389 SHORT TONS
YIELD OF GOLD, ... 129,896·87 OZ. FINE GOLD
DIVIDENDS DISTRIBUTED, ... £135,000

MINING AND TREATMENT COSTS.

1900. 4629 TONS.		1901. 17,453 TONS.		1902. 24,707 TONS.		1903. 50,542 TONS.		1904. 55,056 TONS.	
Total Cost.	Cost Per Ton.	Total Cost.	Cost Per Ton.	Total Cost.	Cost Per Ton.	Total Cost.	Cost Per Ton.	Total Cost.	Cost Per Ton.
£11,378 16 7	£2 9 1·96	£36,447 8 5	£2 1 9·20	£41,019 7 3	£1 13 2·45	£68,133 9 4	£1 6 11·53	£63,755 3 10	£1 3 1·92

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. . . The . . .

Hainault Gold Mine

LIMITED.

HEAD OFFICE:

34 West George Street,
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CAPITAL—Authorised, £130,000.
Issued 121,000 Shares of £1 each,
£121,000.

TONNAGE TREATED, ... 52,652 SHORT TONS.
YIELD OF GOLD, ... 22,736·55 OZ. FINE GOLD.

MINING AND TREATMENT COSTS.

1902. 8818 TONS.		1903. 13,347 TONS.		1904. 29,121 TONS.	
Total Cost.	Cost Per Ton.	Total Cost.	Cost Per Ton.	Total Cost.	Cost Per Ton.
£12,785 11 10	£1 8 11·98	£16,671 3 5	£1 4 11·77	£26,030 18 0	£0 17 10·53

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	Tons.	£
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Output from Sulphide Mill to June 30th, 1904 ...	16,500	78,045
	43,213	695,994

Dividends paid cash £262,500

Dividends paid shares £17,500

Average total revenue expenditure since starting of Sulphide Mill, 1st December, 1903, to 30th June, 1904.

£1 8 9 per ton.

G. M. ROBERTS,
General Manager.

T.E.Mg
C

Australian Mining and Metallurgy

By

DONALD CLARK, B.C.E.

Special Commissioner to the Australian Mining Standard

LONDON
SIR ISAAC PITMAN & SONS, LTD.

AUSTRALIAN MINING STANDARD,
2 NICHOLAS PASSAGE, E.C.

1907

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Queensland.

Victoria.

New South Wales.

Special Features.

PREFATORY.

Does Gold Mining Pay?

Does gold mining pay? The question has often been asked in doubtful tones as a preliminary to answering it in the negative To Australia it is one of supreme importance, because it assails the basis of its great mining industry, for though the winning of its industrial metals is an important phase of the general subject, on comparing the annual values of the respective products it will be seen that the one stands for but an insignificant proportion of the other. If, then, gold mining does not pay, the foundation of the industry is absolutely unstable, and nothing raised upon it can be profitable or permanent.

Those who contend for this view of the proposition declare that gold mining absorbs more capital than it returns; that every ounce of gold won costs more than its value in the winning; that if the cash equivalent of the whole product were divided among the men engaged in the production it would not give them a living wage; therefore that it is a wasteful occupation, diverting into unremunerative channels the capital and energy which might be employed to general profit in other directions.

It is hard to definitely and precisely disprove those asertions, because it is practically impossible to reduce the factors in the general calculation to definite and precise terms. It is easy to arrive at a fairly accurate estimate of the world's gold production year by year, but the very reverse to say what amount of capital has been expended in its production; with what proportion of that expenditure production should be legitimately charged; by what rule the men employed in this production are to be estimated, and how the employment total is to be calculated. The difficulties are so obvious, seeing it is the general and not any particular case which must be judged, that they need not be further detailed.

Furthermore, the allegation as to the unprofitableness of gold mining is undoubtedly a specious one, because it is certain that the dividend-paying enterprises bear but a small proportion to the dead failures and the representatives of hope deferred. But it is by the general results that the general case must be judged, and it may be laid down as an unassailable proposition that no permanent success has ever yet been established, or ever will be established, on the basis of financial failure. To find an answer, therefore, to the question, Does gold mining pay? it is only necessary to ask what has gold mining done? The reply is furnished by the waste places of the world which it has converted into populous provinces, by California, Australasia, South Africa, and Alaska, and it is further emphasised not only by the great gold mining centres, such as Johannesburg, Cripple Creek, and Kalgoorlie, but by all that these reclamations from the desert have meant and still mean in points of development and progressive settlement.

Taking the comprehensive case immediately to hand, it may be claimed without fear of challenge that gold mining has built up the Australasian States; and that with so magnificent an achievement to its credit, it is manifestly absurd to say it does not pay. What Victoria, New Zealand, Queensland, and New South Wales owe to gold mining in the opening up and settlement of these several States need not be dwelt upon to be fully realised. How feeble a course a country can run deprived of this great stimulus was attested in the case of Western Australia. For nearly three-quarters of a century it stood the non-progressive unit in an actively progressive aggregate; until gold mining gave it also the required impetus, and in such measure that in a decade it sprang from a position of insignificance to one of commanding importance. So far from being unprofitable, it may therefore be strongly contended that no other form of industry stands credited with such an accumulation of tangible, realisable, and convertible profit. Moreover, while it injures none, it is the pioneer, the founder and sustainer, of all other industries. Cut-throat competition is the evil against which organised labor in all its branches inveighs, as it affects pretty well everything but gold mining. In this competition has no place. The gold miner, in making room for himself, makes room and provides employment for whole armies of dependents. He is at once the patron and the paymaster of all by whom he is surrounded. His industry is the mainspring which keeps all the wheels of the general mechanism in motion, and as it is certain that gold mining would not be sustained at a continuing loss, the general profit to the investor must be assumed as a point not less clearly proven than the general profit to the community.

Does gold mining pay? Let the enquirer mentally contemplate the Australasia of the opening 19th and of the opening 20th centuries. Let him pass in mental review the great lone lands before they were awakened from their unproductive sleep of countless centuries by ring of pick and roar of stamp mill, and the seven sovereign States, with their noble cities, their vast areas under cultivation, their land and sea-borne trade, and their thriving millions of population. Let him regard these two pictures illustrating the change wrought by gold mining, and then let him answer his question for himself.

THE EDITOR.

WESTERN AUSTRALIA.

MINING AND ORE TREATMENT IN WEST AUSTRALIA

Introductory.

In accumulating information on various subjects connected with mining, one may obtain a cinematographic picture of each and yet actually acquire very little knowledge. It is the explanations given, the data, figures and results which enable one to set forth the experience of other men. Through the courtesy of the Hon. H. Gregory, Minister of Mines, all particulars concerning the public batteries of the State were obtained. To Mr. H. S. King, the Under-Secretary, I am specially indebted, having been afforded every facility for acquiring much of the material necessary for these articles, and also to Mr. Gibb Maitland, the Government Geologist; Messrs. Purdie and Allen, of the Perth Technical School; and Mr. Robt. Allen, of Perth, for their welcome co-operation and assistance. My acknowledgments to workers on the fields and mines will be made in due course.

So far as mining was concerned, Western Australia was certainly the Cinderella of the States. Queensland, with its Charters Towers, Gympie and Mount Morgan; New South Wales, with its extensive coal measures and Broken Hill; South Australia, with its Moonta and Burra Burra; Tasmania, with its Tasmania, Mount Bischoff and Mount Lyell; Victoria, with its £260,000,000 worth of gold; were all apt to look upon their western sister with a tolerant, patronising air. Now she has outstripped all her sister States, and shown how one small field such as Kalgoorlie can turn out as much gold as the whole of the fields in any State. The area of Western Australia is 975,920 square miles, its great length from north to south being 1400 miles, and from east to west 1000 miles. The southern and south-western areas are fringed with coastal limestones from eocene age to the recent calcareous sands, so prevalent around Perth and Fremantle.

Carboniferous and mesozoic rocks occur over large areas; cambrian rocks have been identified in the Kimberley district, while the archæan rocks, comprising gneiss, granite and schist, are said to be more extensive than in any portion of the world. These archæan rocks are important, in that the valuable metallic deposits occur in them. Mr. Woodward considers there are six distinct belts running north and dipping slightly eastward. The first belt runs nearly parallel to the western coast and contains lead, copper and zinc. The second belt contains the Greenbushes tin fields and fine deposits of graphite.

The third belt lies about 100 miles inland from the coast, and is about 100 miles wide. This belt contains bold, bare outcrops of granite, which are flanked round with sand. Rain runs off as from the roof of a house. This water finds its way through the sand to be entrapped in hollows, giving rise to soaks and gnamma holes.

Drains are now made circumscribing these outcrops, and the water led into excavated dams.

The fourth, or first auriferous belt, lies east of the granite belt. It is about 20 miles in width. Starting at Phillip's River, it includes Parker's Range, Southern Cross, Golden Valley, Mount Magnet, Cue, and Nannine. The fifth belt resembles the third; that is, it is made up of granites and other igneous rocks. This divides the Southern Cross auriferous belt from the sixth belt, whose width has not been determined. The sixth belt starts from the Dundas Hills, and includes the famous fields of Coolgardie and Kalgoorlie, and extends right through to Marble Bar and Mulline on the north-west coast. The area of the present goldfields may be conceived when it is remembered that each of these belts is over 800 miles in length.

As early as 1861 a prospector found auriferous stone 30 miles east of Northam, but could never re-discover the place. In 1862 the Government engaged Mr. E. H. Hargraves, the discoverer of gold in New South Wales in 1851, to prospect and inspect the country for gold. Mr. Hargraves' opinion as published, "The Non-Auriferous Character of the Rocks of Western Australia," shows how the practical man will theorise, even to his own undoing. In 1864 Mr. C. C. Hunt headed parties which penetrated the country to the Dundas Hills, Lake Lefroy, and probably passed over the Coolgardie goldfields. They named the "Hampton Plains" after the Governor of the day, but only looked upon the country from a pastoral point of view. In 1869 the Government offered a reward of £5000 for the discovery of a goldfield within 300 miles from any port, to be paid after 5000oz. had been won; a still more progressive step was taken in 1873, when 16 Ballarat miners were imported to prospect and mine for precious metals. A battery was erected at Fremantle. This venture ended in failure. In 1885 the Kimberley goldfield was opened up, and in the following year 2000 men had been attracted to the place This marks the first successful move which has resulted in the modern discoveries. Prospectors again set eastward from Northam, and after many small finds Southern Cross was discovered in 1887. From that time for a number of years new fields were proclaimed with feverish haste—Pilbarra, Ashburton, Marchison, Cue, Siberia, were all discovered before 1891. In April, 1892, Bayley and Ford started eastward over that desolate, dreary strip which lies to the east of Southern Cross. After having been beaten back for want of water they slightly changed their course, and arrived at the native well—Coolgardie—where they camped. During the next three days they picked up 200oz. of alluvial gold. Returning to Southern Cross for provisions they hurried back to the scene of their find, and on the day they reached Coolgardie for the second time discovered the famous Bayley's Reward mine. The first afternoon's work gave them 500oz. in specimens. Bayley returned and showed 554oz. to Warden Finnerty on the 17th September, and obtained a lease of his discovery claim. The gold fever seized the local inhabitants, and in less than a week hundreds were on the way to the latest land of promise.

Kalgoorlie was discovered in June, 1893. During the next three years Bulong (I.O.U.), The Londonderry, the Wealth of Nations.

Menzies, the Norseman, and a host of other goldfields were discovered.

The difficulties the early prospectors had to contend with can never be properly understood by those who have not faced them. The country from Northam to the goldfields is so flat that for hundreds of miles there is neither cutting, embankment, nor bridge. A few culverts are constructed where water might run. Isolated bare granite outcrops, or low rugged ridges capped with ironstone are dignified with the name of mountains. The country, however, is not bare, but sparsely covered with stunted gums about 50 feet in height, with a scaly butt and bare white or red stems; a thin cluster of pendent leaves is borne on the summit of umbrella-shaped branches. Mulga, quandongs, cotton-bush and salt-bush hide the bareness of the parched red soil, while on the sand ridges the globular tufts of the bright-green spinifex shows in marked contrast against the blue of the short scrub and the bilious green of the scraggy gums. Great flat plains, covered in some instances with sand, in others with a stunted scrub, are known as lakes. The water, it is explained, is underneath the surface. The soil appears to be fertile, but is redeemed from being a desert by the scrub, which extends over hundreds of miles. As soon as settlement destroys vegetation the fine red sand which floats with every breeze envelopes and tinges all natural and artificial objects with a color peculiar to itself.

While the Queenslander is never at a loss to give an alligator story, those from the other States have their snake stories, yet every Western Australian can outdo them all with the wonderful tales of the wandering willy-willies. The meandering whirlwinds are ever present. Now and again they gather up tins which so plentifully strew the surface, rattle them together in a manner suggestive of a tin-kettling party, carry dust, paper and rubbish aloft until they vanish into thin air. While these are the bane of a housekeeper's existence, yet on a larger scale further north their destructive violence is vouched for by the Government Astronomer, who states:—"Houses and sheds are usually tied down to the ground by means of strong iron cables, but the wind generally takes no notice of these impediments, and is rarely satisfied with anything short of a complete wreck." For further information as to disappearing tanks and bulky articles I need only refer you to any of the residents.

The climate of the southern and south-western coastal districts is said to be the most salubrious in the world; most people would not say the same about the goldfields towns. The average mean maximum day temperature for January is very nearly 100 degrees for the Coolgardie goldfields, while the Murchison is 105 degrees. The nights are said to be cool, but this is only relative to the day temperature. There is no doubt that a continual sun bath from early in the morning until late in the afternoon is not conducive to the comfort of most white people. The winter climate is said to be delightful. It cannot be said that flies or mosquitoes are troublesome : possibly with the advent of water they will multiply and increase.

The want of fresh water in the centre of the State has proved the chief drawback to its progress, and even now it is a grave menace to the welfare of a city with 25,000 people, nearly 400 miles from a supply centre. Even now the cost of travelling or prospecting must be great when a horse will consume 2s. 6d. worth of water and a camel swallow £1 worth at a draught. One cannot help admiring the courage of the explorers who traversed a desolate waste of spinifex, sand and scrub, or the perseverance and pluck of the prospectors who went out into the wilderness, and by their discoveries sowed the seeds of the State's mining greatness. Some few were personally successful, but the credit belongs to all. Water had to be carried off or condensed from a liquid salter than the sea. Tinned meat and damper, work of the most arduous description, carried on in the driest of atmospheres, all helped to impoverish the blood and weaken the system. Little wonder is it that they went down with fever, and that many were carried off in those days when medical comforts were little known and nursing still less so. Conditions are now so altered that hardships, as compared with those of a few years ago, are almost unknown, yet mining is handicapped to a very great extent owing to the absence of abundant supplies of wood and water.

There is little doubt, had the Eastern goldfields a rainfall like that of the coastal districts, the topographical features of the country would have been greatly different. Deep channels would have been cut, and water-sheds would have been clearly marked. The rainfall is from about six to nine inches per annum; this limited supply has little or no geological effect in removing material and eroding channels. In more ways than one such a meagre rainfall is of advantage in a country where the water lies, and where a downpour of a few inches would mean flooding the country and converting the roads in the red soil into clay quagmires. Semi-tropical conditions would be much more trying than the clear, dry heat which now prevails.

Water is conserved in other places, even in immense quantities, where the rainfall is no greater, but the configuration of the country and the amount of soluble salts in the soil are against this here, except on a small scale. For household purposes, an extensive roof and an abundant tank supply will give enough water if sparingly used. While a few are so favored, the great majority on the goldfields have to purchase all the water they use, while all the large mines have to erect elaborate condensing plants to supply the shortage caused by imperfect condensation of their steam. There are many public condensing works on the fields, and special condensers have been evolved to suit local conditions. Many elaborate methods were tried, but a very simple form is almost universally adopted.

Five ordinary cubical 400 gallon tanks 4 feet by 4 feet by 4 feet are set on edge, so that they form a diamond shaped boiler 20 feet in length. A similar row is set parallel to these with middle edges adjoining. These are built in. A fireplace is placed in front between the series, and flues are so arranged that the heated air circulates round the tanks and returns to a chimney alongside the fireplace. The tanks are connected below, while above they lead to a common pipe through which the steam passes. This pipe leads to a closed jacket which surrounds a galvanised iron tank, the tank itself being full of the salt water, which has to be condensed. A portion of the steam is liquefied and drains away to a reservoir.

The water in the tank becomes raised to near its boiling point by its contact with the steam, and runs continuously into the first tanks. or those on each side of the fireplace. The salts in the water, which soon deposit, are blown out at the opposite end from time to time.

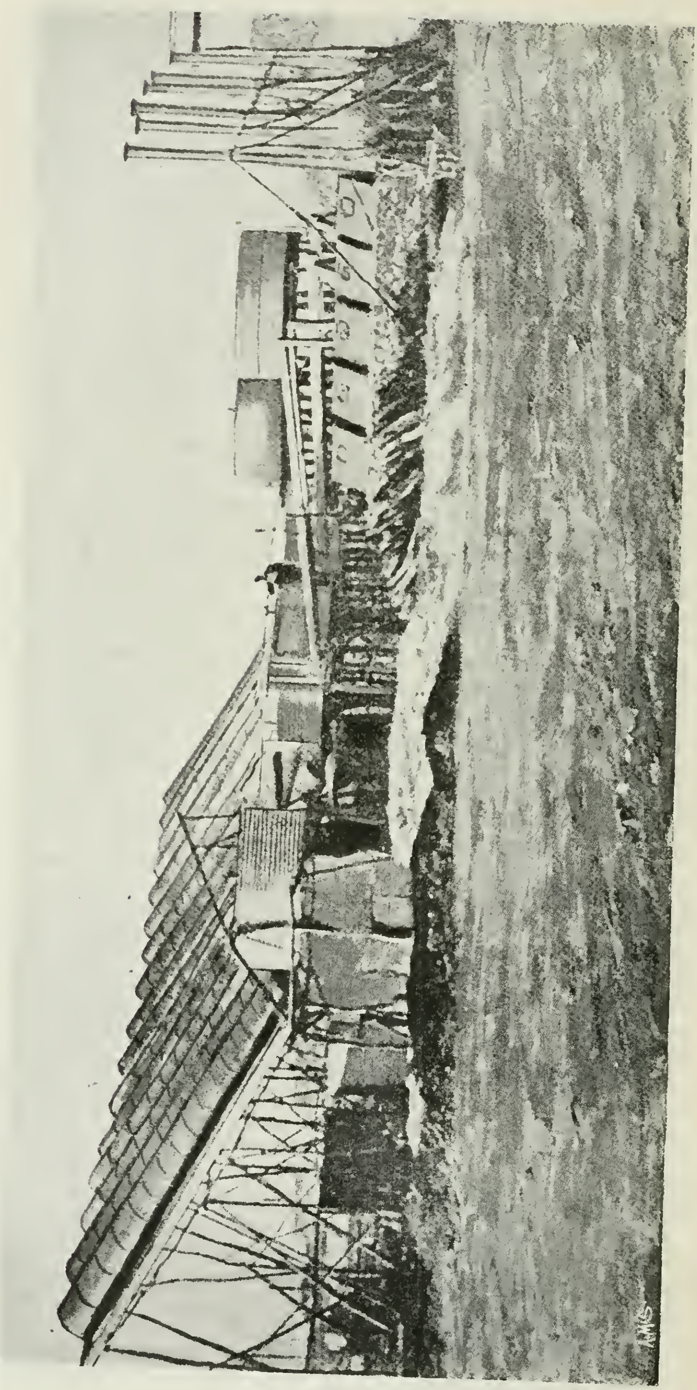
The vapor which passes the first cooler goes through a series of inclined or vertical galvanised iron hollow cylinders about 4 feet in diameter and 20 feet in length. These have open pipes running through them. Each of the cylinders condenses some of the vapor which is drained away, and the number can be so arranged that very little escapes. These rows of columns, so common on the goldfields, are in reality atmospheric condensers. Joints are made tight with a little paste. In some cases, in order to facilitate cleaning out, each tank is divided so that the upper half forms a cover resting in a water seal. The top can thus be lifted bodily with block and tackle, and scale or salt removed from the lower portion.

The cost of distilling with wood at 15s. per ton, or green wood from 24s. to 25s. per cord, is about ½d. per gallon, or 4s. 6d. per 100 gallons. It is sold to the mines in large quantities at from 6s. to 7s. per 100 gallons; while the townspeople pay from 10s. to 12s. 6d. So far as one could see, no effort was made to reduce the atmospheric pressure in order to lower the boiling point; possibly the additional cost of working on a comparatively small scale would have counterbalanced the saving effected. The water condensed still contained appreciable quantities of chlorine, and had the effect of causing iron vessels in which it was stored to rust rapidly. Possibly it still contains appreciable quantities of ammonium salts or volatile organic products; but on this point I was not able to obtain information. Certainly in the interests of the consumers a most careful analysis should be made.

The water from the shafts adjoining the mines, for the gold mines themselves may be said to be dry, contains from 6 per cent. to 19 per cent. solids; that from the Great Boulder mine giving—sodium chloride, 11.164; magnesium chloride, 1.425; calcium sulphate, .462; lime, .422; 13.473. Such water is sold at the pit's mouth for from 5s. to 6s. per 1000 gallons. Mr. George Roberts, the metallurgist at the Great Boulder mine, has kindly supplied me with the cost of water per ton of ore treated: Fresh water, 3s. 4.182d.; salt water, 10.830d.; up keep, etc., 4.686d.; total, 4s. 7.698d. These figures are for 89,521 tons crushed during the year. The total cost of fresh water being £14,921 16s. 5d.; salt, £4021 13s. 1d.; up keep, repairs to pipe, etc., £1745 11s. 9d.; total, £20,689 1s. 3d. On other mines it has cost as much as 6s. 8d. per ton of ore treated.

It will readily be seen the water difficulty is a serious one, not only handicapping the mines, but adding to the cost and discomforts of living. Almost the whole inland population depend on condensed water, while house-owners live in perpetual dread of fire. Insurance rates amount to several per cent. per annum; and there is little interval between the beginning and end of a fire in a canvas house, or those whose timber is dessicated in this almost rainless region.

The salt water at the mines is a highly objectionable material, loaded as it is with solids; it has to be used over and over again, until it is so saturated that even a small amount of evaporation or



A Big Condensing Plant.

change of temperature will cause it to deposit its salts all over the place: every leak or drip of water about the works is indicated by long pencil-like stalactites, while the pipes and launders choked and filled with large and often perfectly formed crystals of gypsum (selenite), and even the grinding pans and settlers become coated with a thick incrustation of the same substance. When the water becomes alkaline through lime being added, a white inflorescence of magnesium hydroxide separates out: this covers the filter presses with a snow-like deposit. The water still left in the press cakes of residues amounts to from 15 to 20 per cent.; this evaporates from the tailings heaps and leaves the finely pulverised sand cemented and crusted with various salts. Fortunately this is so, for were the slimes loose and pulverent, the dust caused by many millions of tons would smother the city. It is of course impossible to use such saline water in the boilers, consequently it must be distilled and elaborate precautions taken to ensure its condensation. It is a rare thing to see a puff of steam escaping into the air. The steam condensed from the cylinders contains oil: this is filtered out with special filters before it passes back to the boilers. The loss owing to imperfect condensation amounts to 25 per cent.: this must be made good with fresh water.

The Government, with foresight almost intuitive, undertook to lighten the load which so handicapped the mining industry, by means of a scheme which is far beyond any venture of the sort ever attempted. A weir has been constructed on the Helena River, 21 miles from Perth, and 330 miles from Coolgardie. Eight intermediate pumping stations are installed: the water is forced into reservoirs from one to the other until raised to a reservoir to a height of 1000 feet in all, whence it gravitates to Kalgoorlie At each pumping station three sets of engines and pumps are provided, two of which will be sufficient to do the work. The pipes which are now laid are made of steel 30 inches in diameter, ¼ inch thick, and 28 feet in length: they are coated with asphaltum. Expansion joints have been provided, and all provision made against serious failure. The late Mr. C. Y. O'Connor, C.M.G., was the engineer for the scheme. Messrs. Mephan Ferguson, G. and C. Hoskins were the successful contractors for the 60,000 pipes required, their contract price being £1,025,124. At the date of my visit to the West, in 1902, the water scheme had not been completed, and salt water was mainly used for milling and cyaniding. The change from the saline to the fresh water introduced many difficulties: the incrustations from pipes and pumps were slowly dissolved off, giving rise to leaky joints, while a considerable quantity of sediment was stopped by the filter cloths of the filter presses, giving rise to an increased consumption. Next gypsum, which is freely soluble in saline solutions, crystallised out from the fresh water. This occasioned much trouble in the zinc boxes, and gave much extra work in the treatment of the zinc slime.

Fuel is also a source of much higher expenditure than in other States. The wood used as a rule is so free burning that it is like feeding a fire with paper. The country has been devastated of large timber for miles around, and though branch timber lines are pushed further and further back, it is likely that the price will reach that of coal conveyed on the railway from the local mines.

The present cost of 15s. per ton means about 20s. when compared with the firewood we produce.

Timber for mining purposes is practically non-existent at the mines, but the other portions of the State send up the magnificent poppet legs and heavy timbers so necessary in these large mines, and the cost of these would probably be very little greater than in our own mines. As a matter of fact, the timber used underground in the mines visited was very much less than one sees in other places. The ground is certainly very good, but great care is taken not to waste timber unnecessarily.

A scheme which would have worked out well, were it not for vested interests now involved, would be to build a heavy rail broad gauge line down to Esperence Bay, the fall from Coolgardie being over 1200 feet in 236 miles. The ore requiring chemical treatment from the mines could be run down to sea-board and treated even by the present methods at a less cost than they now are. It is doubtful if this can be done if the fields are supplied with cheap water.

It cannot be said that the State is behind in providing railways. In 1881 the Eastern line was opened to Guildford; in '84 it reached Chidlow's Well: the following year it had passed through York, while in 1886 Northam was the terminus. Gold brought it to Southern Cross in 1894, to Kalgoorlie in 1896, to Menzies in 1899, and now it is being pushed along to connect with the railway of the Murchison field. Even now there must be a difficulty in supplying fields having a population of about 40,000 with necessary foodstuffs, and mines yielding some £5,000,000 worth of gold with machinery and supplies. The water difficulty again has to be faced. The railway tanks along the line are somewhat precarious sources of supply, and when they give out either condensed water has to be purchased, or the engine has to haul its own requirements in water trucks, which unnecessarily burden it. Fortunately a breakdown has never occurred, though it must cost a large sum to avoid the possibility of it. Marvellous is not a word to apply to the cities on the goldfields; they are phenomenal in their growth and solidity; old methods have been left behind to a great extent. Fine buildings, streets laid out on no niggardly system, well paved footpaths, bicycle tracks and well metalled roadways; electric lights, suburban railway traffic, and electric trams, all show the magic power of the yellow metal.

Gold Deposition - The Great Boulder.

It is the common custom to make comparisons with other States in order to show the latest gold producer off to the greatest advantage, but such comparisons are not fair. Take, for instance, the Victorian case. Victoria in the early days was rushed by a gold-seeking population who had very little experience in searching for alluvial, and none when it came to extracting small particles of gold from hard rock. Reef after reef was left in the early days because the material was looked upon as too difficult to work; the evolution of quartz mining was slow, the work proceeded cautiously, and even now, owing to this original policy, any change of machinery or method is viewed with the greatest suspicion. Contrast this with the West. Thousands of trained miners flocked to the newly discovered fields; money poured in for development purposes; machinery of the most extravagant type was rapidly run up, and mines were opened up at a rate almost unparalleled. Had Victoria been a new field with modern facilities, modern knowledge, and with modern men, then the world would have been astounded with the wealth produced year by year. Western Australia will be a great gold producer for years to come, but it may take another jubilee for her to catch up to the long lead Victoria has given with regard to gold production.

One cannot help contrasting the comparative small amount of alluvial—or so-called alluvial gold—won in W.A. in comparison with that obtained from reefs also with that of Victoria. Nor can one get over the remarkable fact that while the great bulk of the gold obtained from the reefs is as fine as flour that very little of this gold has been won from the alluvial. It seems to be quite feasible that surface solutions may have had solvent action on this gold and carried it to spots where it has been re-precipitated. A certain amount of denudation has gone on all over the fields, and even if this only amounted to a couple of hundred feet during the time the present reefs were exposed, then the amount of gold shed into the brecciated material surrounding these outcrops should, in a field such as Kalgoorlie, have amounted to millions of ounces, and not a paltry few thousands which have been won. The matter is well worthy of investigation, and it may throw some light on the poverty of some outcrops and their richness down below and vice versa.

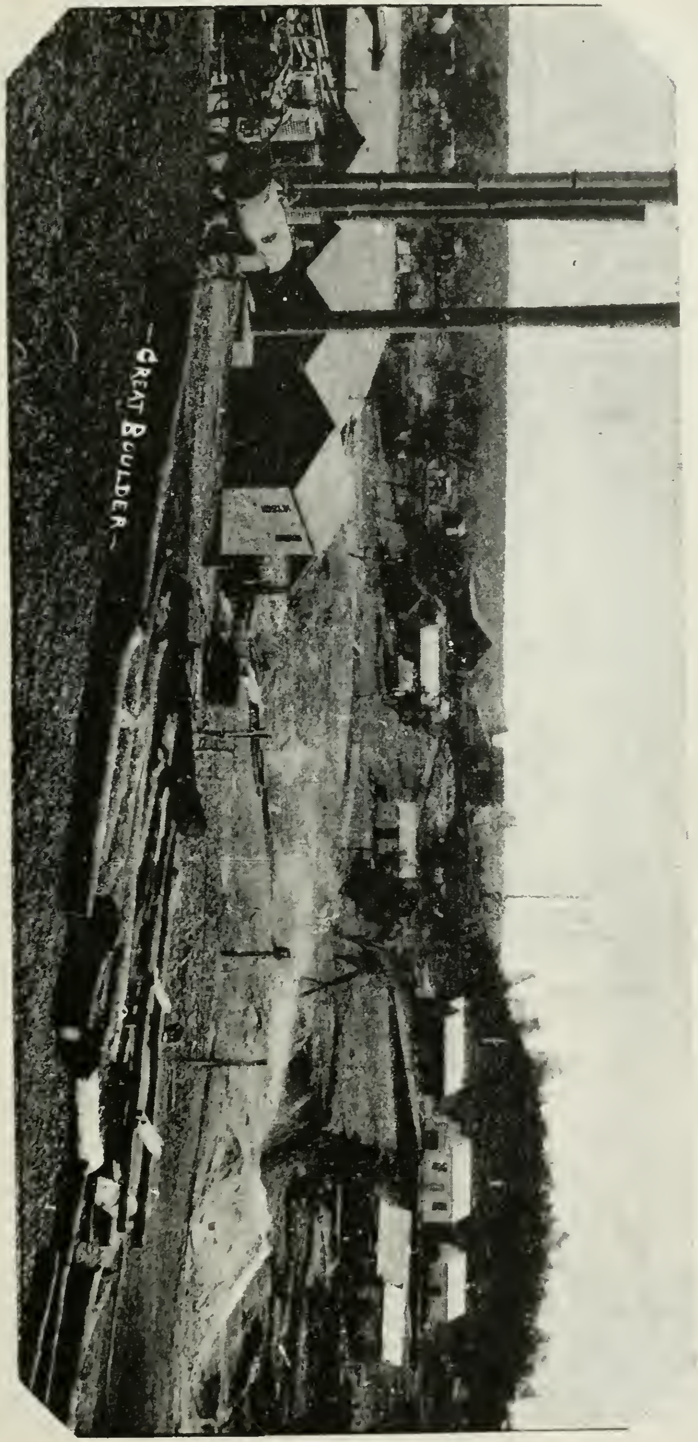
One of the most noticeable features at Kalgoorlie is the enormous plants dumped down on such small areas; coincident with this is the hurry and bustle that goes on. "We are anxious to work the place out and get away from it," jocularly remarked one prominent mining man. "The gold is better in the bank than in the ground," was another excuse given for running out 30,000oz. per month, and causing many sanguine shareholders to fall in. Victoria in the early days sent shafts down a few hundred feet and appointed a Royal Commission to decide whether gold would probably go down to 1000 feet. At Kalgoorlie they sink to 1000 feet almost in one act and start off straight away for the second thousand. There can be no husbanding of resources with leases containing only

24 acres, where the small hoppy-go-hop batteries have been displaced by the hundred head of heavy stamps, with their long and rolling hum. Work progresses at a rapid rate, day and night. Sunday and Monday; it must be confessed that with work proceeding at such a rate, the greatest efforts have to be made to keep development work well ahead, and it is probable that the limit of production has been reached in most of the mines, and perhaps overreached in a few of them.

It has often been stated that Victorian mining men were at a loss when called upon to deal with the lodes of Kalgoorlie. The matrix which contained the gold was not quartz, neither were the lodes, so-called, contained in a channel with defined walls. A hard blue rock, specked with pyrites here and there, which on Bendigo would be called mullock, carried the gold, and this material was called a lode wherever it was payable. Many of the mines contain little or no free gold: in others veinlets of telluride of gold, or blotches of the same mineral, barely distinguishable from pyrite, permeate the rock.

The rock which contains the gold is locally known as a diorite or greenstone, but it has been proved by examination to be a diabase. The lodes themselves are portions which have been fractured and fissured, and through such channels the auriferous solutions have flowed, leaving some of their valuable metals crystallised through the disturbed areas. On some parts of the field vast chambers have been hewn out in removing payable ore: in others, such as the Boulder, the values are sharply defined to within a few inches one way or another of the width of the material taken out. Appearance counts for little or nothing where no gold nor telluride veins are visible. It is only by a continuous system of sampling that the mining manager is guided as to values: not the slip-shod assaying and sampling done on some of our Victorian mines, which have brought the practice into discredit. Hundreds of assays, taken at regular intervals, broken down from wall to wall, are separately pulverised and sieved, and the sieved sample handed to the assayer. "How often do you take your samples?" "Every six inches in this mine," was the reply obtained from a manager who was not even then satisfied that he could speak with absolute certainty as to values. Certainly he erred on the right side, and his mine was as well mapped out as to values as the stock in a merchant's shop. This system, so necessary here, has been of such assistance to managers that it has been adopted almost universally; and it has been stated by most reliable men that even with patchy reefs, having coarse gold, their battery values followed their assay values very closely. At times there are great discrepancies on this field between estimated and actual values, but if the origin of these were sought it would be found either in the incompetence of the men connected with the work or else be the result of market manipulations. It may be a difficult matter to value a mine to within a few per cent., but there is no excuse for values of a mine well opened up being declared to be hundreds of per cent. over actual values.

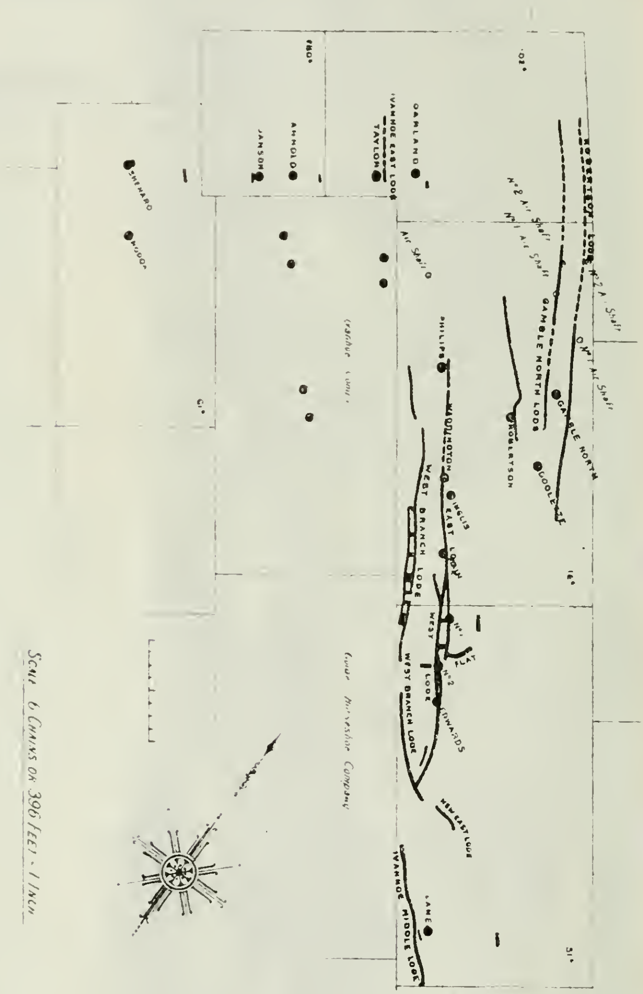
Another indispensable method of development lies in the use of the diamond drill. This work is so important, and can be carried on so cheaply, that the best managed companies have made full use of the system, and have bored their properties from boundary to



The Great Boulder in 1895.

boundary, thereby gaining valuable information, and in many cases making important discoveries. The work at Kalgoorlie is largely done by the Sullivan Diamond Drilling Co., and I have no doubt that the system on which they work will be extended to other States in the near future.

The Great Boulder property consists of four leases, the first two having an area of 24 acres each, the third 21, and the fourth 16



Plan of the Great Boulder Leases.

acres. The main workings are carried on in the first two leases, whose length in a north-westerly direction is nearly 3000 feet, and whose width is one-fourth of this, or 750 feet. The lode runs tantalisingly close to the western boundary, and dips towards the Golden Horseshoe lease on the south-eastern end, and towards the Ivanhoe property on the northern end. These last two properties, each having a lease of 24 acres, together with the Boulder, produce nearly half a million ounces of gold per annum. In the Boulder property a belt of rocks classed as felsitic schists and slates, varying in width from 200 to 500 feet, runs along the whole length of the leases. On the western side is the famous Boulder, Horseshoe and Ivanhoe; on the eastern the Lake View, Perseverance, and other properties. The lodes on the Boulder mine are known as the east lode, the west lode, and a lode which forms a loop with the west lode, known as the west branch lode. The last outcropped close up to the western boundary, and dipped into the Golden Horseshoe property, thereby helping to make that mine one of the richest in the world.

The west lode, after a westerly dip, which brought it up to within 50 feet of the Horseshoe boundary, commenced to dip the other way, so that at the 1400-feet level it is further away from the boundary than it has been for some time. Down to the 900-feet level the general trend of the dip is westerly.

In going down a mine such as the Boulder a casual observer cannot form any opinion as to values, other than that apparent from the quantity of stone removed, the absence of a mullock tip and the presence of barren material from other sources filling the stopes depleted of their ore. The ore removed must have been of wonderful value, for up to the end of 1900, 198,248 tons were treated for 449,944.7 ounces and £175,000 paid in dividends. For the year 1900, 54,887 tons were treated for a yield of 115,908oz., and at the end of 1900 it was estimated that down to the 1000 feet level there were 143,800 tons, containing 211,324oz. of gold, an estimate which has since proved to be considerably under the mark.

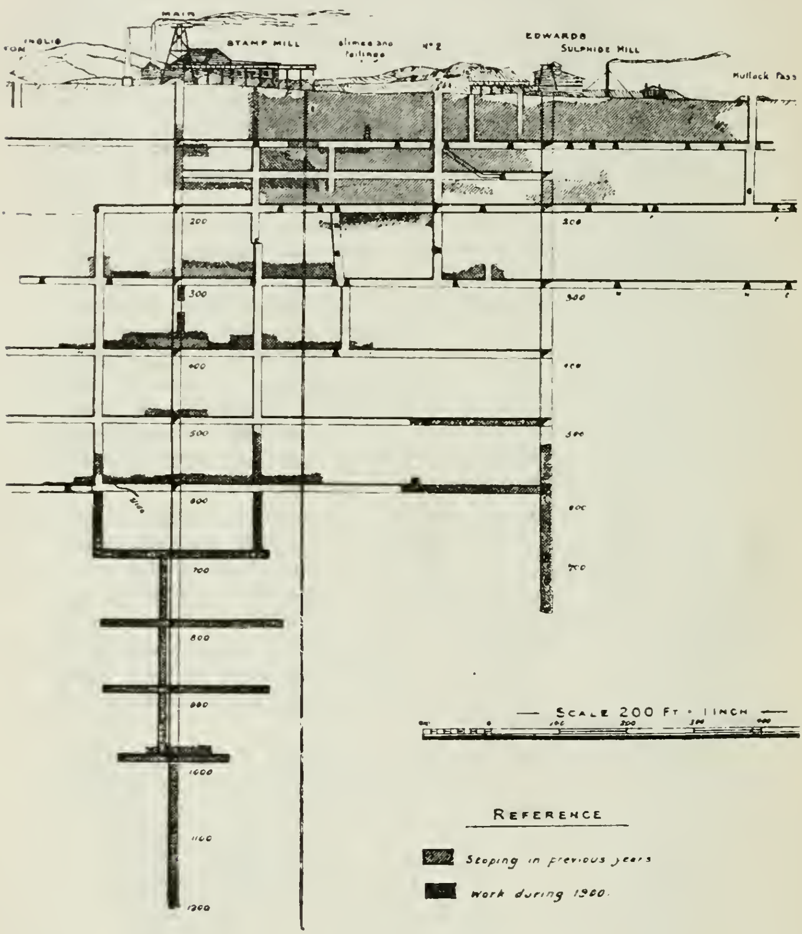
At the 1200 feet level the lode is about 8 feet in width, and had been driven on at the date of my visit for over 500 feet, the values averaging at least 2oz. per ton for this distance—none of that stone had been broken out. Although this level does not make the mine any more than one swallow makes summer, yet it has raised the hopes of the local people, who anticipated better yields from the lower than from the upper levels in other mines as well. This may prove so, but, taken as a whole, it would prove contrary to all other Australian experience.

As may be seen from the plans and sections the Great Boulder mine is one of the best developed on the field. The work done for a single year is suggestive—

	Feet.
Shafts	940
Levels	2,872
Crosscutting	1,339
Rises	789
Winzes	677
Air passes	180
Total	6,797
Diamond drilling	6,977

The mine is now well opened up, and should have stone for at least three years ahead, while the dividends on the present price of the shares amount to about 10 per cent. The savings effected in the cost of treatment have amounted to as much as 7s. per ton for the past twelve months, so that if the ore maintains its value a very much larger profit should be made. As a further guarantee of careful management, it need only be mentioned that Mr. R. Hamilton is in charge, and that Mr. Geo. Roberts is the metallurgist of the mine.

Since this description was written Mr. Hamilton's estimates have been abundantly justified. The main shaft has been pushed down



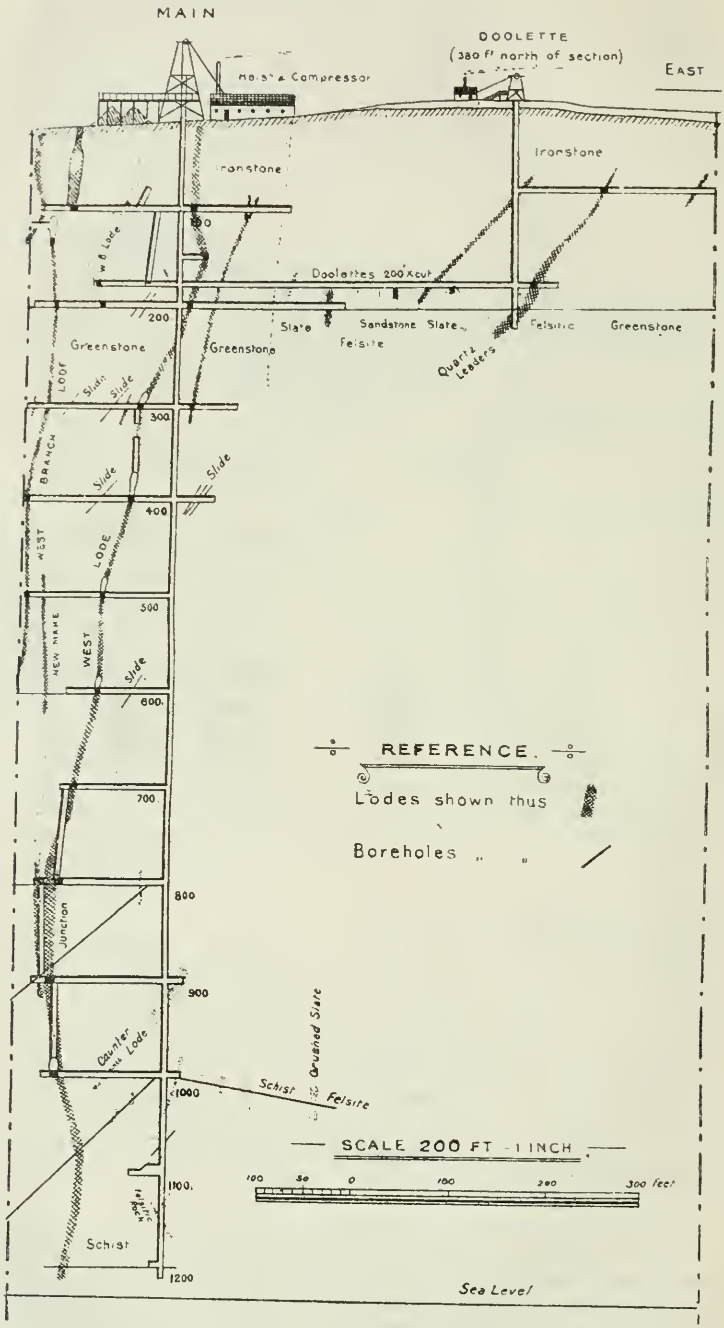
Great Boulder Workings.

to 1600 feet, and the lode has been proved to be much nearer the shaft than it has been for a thousand feet. The probability of it dipping into the Golden Horseshoe Estates, which seemed almost a certainty at the 900-feet level, has not been borne out by the results of work at deeper levels. The present ore reserves are even now ahead of what they were in 1902, while the cost of production has been lowered.

When ore was first treated at the Boulder it was assumed that the old-time methods would be good enough: the gold was got, the tailings were waste products. Battery work, followed by pan amalgamation, was looked upon as perfect. Fortunately there were no running rivers in which they could hide evidences of bad work. Tailings heaps remained to be sampled, and the army of assayers and scientists from other States, Germany, England, and America showed that altered methods were necessary. The tailings heaps were cyanided, and the difficulty of extraction was declared to be overcome: roasting and cyaniding were introduced, and yet the recovery of gold was not satisfactory. Rich tailings were hidden and buried in the stopes of some mines. A new word was necessary, since the tailings from such operations were rich enough to be re-treated. Residues was applied to such payable sand, and tailings to the final material going over the dump. Then commenced a series of rival methods of treatment. Claims were always put forward for over 90 per cent. extractions. Yet even after this tailings often became residues, from which as much as 16dwt. per ton were extracted. Each man did what was right in his own eyes, and though in many instances the process has been fearfully extravagant so far as companies are concerned, as silently testified by the "hospitals" or masses of unused or discarded machinery; yet out of the chaos has emerged a few interesting and original processes, which will serve as stepping stones to future advancement. The end is not yet come. Even now the rivalry does not appear to have ended: tailings heaps are said to be guarded in some instances against would-be intruders, who are anxious to know the value of the tails, "for fear tellurides might be stolen," as one manager facetiously remarked. The knowledge of bad work being done, and the efforts made to improve upon the present extractions, is a very hopeful sign. On many fields, so long as good yields are forthcoming, the managers have adopted the ostrich policy with regard to their losses. Great blunders have been made in the past, but the competition between rival companies has been too keen for the authors of such blunders to live in happy ignorance.

From a treatment point of view, the Boulder ores cannot be looked upon as specially refractory. For instance, they cannot be compared with those at Bethanga or Cassilis in Victoria, nor with many ores in Central and Northern Queensland. There are two difficulties to contend with, the former being due to the presence of tellurides, the second that the gold is so fine and so encased that the finest grinding is necessary to free it. Treatment with special solvents, such as cyanide of potassium, with bromo-cyanide, or roasting, is necessary to overcome the first difficulty, while the second necessitates sliming the whole of the ore and the attendant difficulties in treating it.

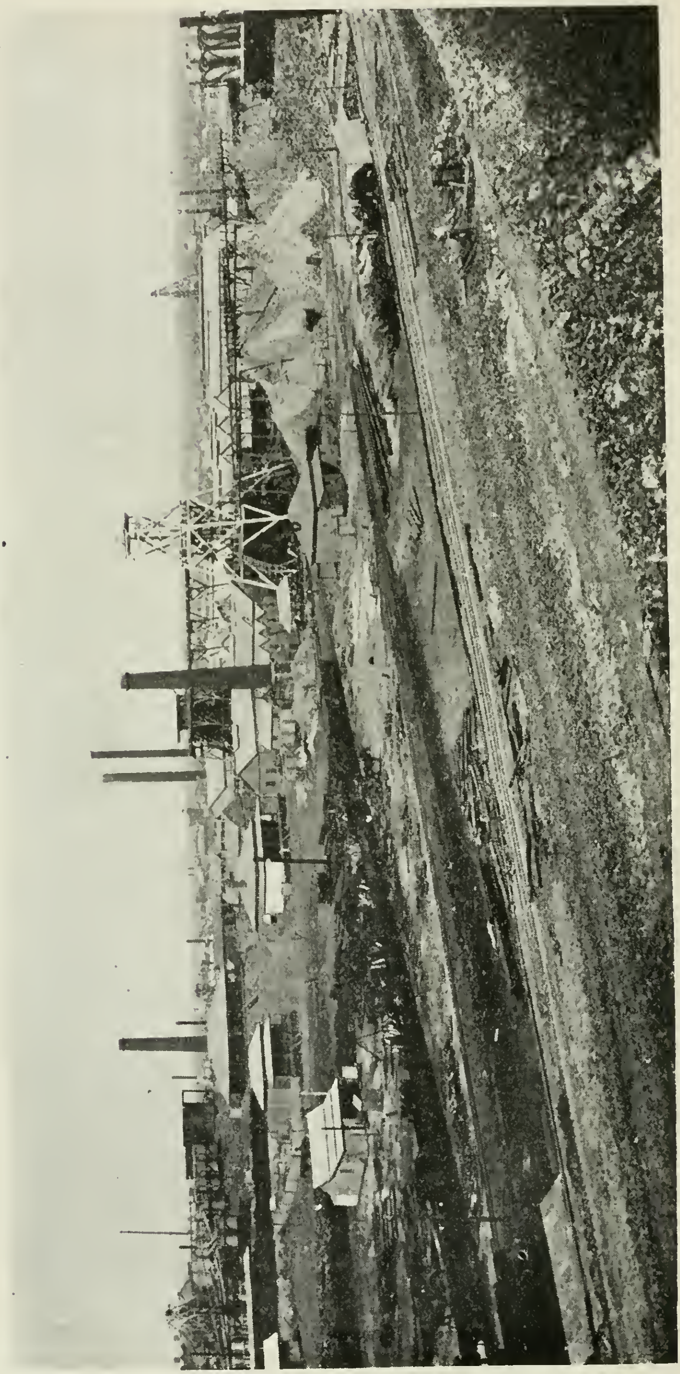
The Great Boulder Company started with the battery and pan amalgamation system, but afterwards, with the knowledge gained from the nature of the ore, erected a special plant to deal with the same; while the tailings which had accumulated were dealt with specially also. Analyses of the oxidised and sulphide ores are given:—



Great Boulder Workings.—Transverse Section.

Oxidised.		Sulphide.	
SiO_2	66.80	SiO_2	74.95
Al_2O_3	8.44	Al_2O_3	1.75
Fe_2O_3	17.39	FeS_2	5.40
Fe_3O_4	0.46	$FeCO_3$	3.22
CaO	0.37	$CaCO_3$	7.03
MgO	0.13	$MgCO_3$	4.76
NaCl	0.50	K_2O Na_2O	1.40
H_2O (combined)	5.15	H_2O (combined)	.25
	99.24		98.76

As previously mentioned, there are two distinct classes of machinery for the treatment of the oxidised and sulphide ores. The oxidised ore and a portion of the sulphide free from telluride is put through the usual stamp battery, followed by amalgamation and cyaniding. The machinery for treating the sulphides is of an altogether different type, the ore being dry crushed, roasted, pan amalgamated, and cyanided. The ore is brought from the main shaft in box cars, having a capacity of 14cwt. each. These are pushed into a kick-up, and discharge their contents into a No. 5 Gates breaker D type. This breaker is fitted with a manganese steel mantle instead of the ordinary chilled solid head. One mantle in use crushed 80,000 tons of rock from pieces up to a hundredweight—weight down to a gauge of 2 inches before it was replaced. This breaker is driven by a 14-inch rubber belt, from a 5 feet 6 inches friction clutch pulley placed 60 feet from the breaker. The breaker wheel runs 400 revolutions per minute, and absorbs about 50 horse-power. Its capacity is 170 tons in 8 hours. The broken material is delivered into a revolving trommel, having inch screens. The fines when dry drop on to a 14-inch Robin's belt conveyor, and are carried direct to the Griffin mill bin; if they contain more than 3 per cent. of water they are passed through a Howell White pattern revolving drier, and then elevated to the bins. The coarse lumps from the trommel fall on to a grasshopper conveyor, having discharge doors six feet apart: these doors are regulated so that a 500-ton bin may be evenly filled. The ore is taken from this bin by three automatic feeders, and dropped on to shaking troughs, which deliver it into three Gate's breakers H type, each having a capacity of 100 tons per 24 hours, crushing the rock down to 1-inch gauge. These are driven at 600 revolutions per minute. The crushed ore falls on to a 14-inch Robin's conveyor, thence into a truck, which is tipped into the Griffin mill bin. The Griffin mill, as used on some of the largest mines on the field, somewhat resembles a Huntingdon mill, but in the former case only one disc is hung on the shaft. It consists of a pan lined with a steel ring, 2 feet 6 inches in diameter. A horizontal disc of less than half this diameter is attached to the lower end of a vertical shaft 6 feet 6 inches long. The upper end of this shaft is hung in the axis of a horizontal pulley by means of a ball and socket joint. On rotating the pulley, the shaft, with the disc attached, also rotates at the same rate, while the disc swings out like a conical pendulum, and presses against the steel ring. The ore is broken, crushed, and pulverised by the action between the two. Above the ring is a screen. By means of a fan fixed on the shaft, the pulverised ore is blown through the screen, and falls through a series

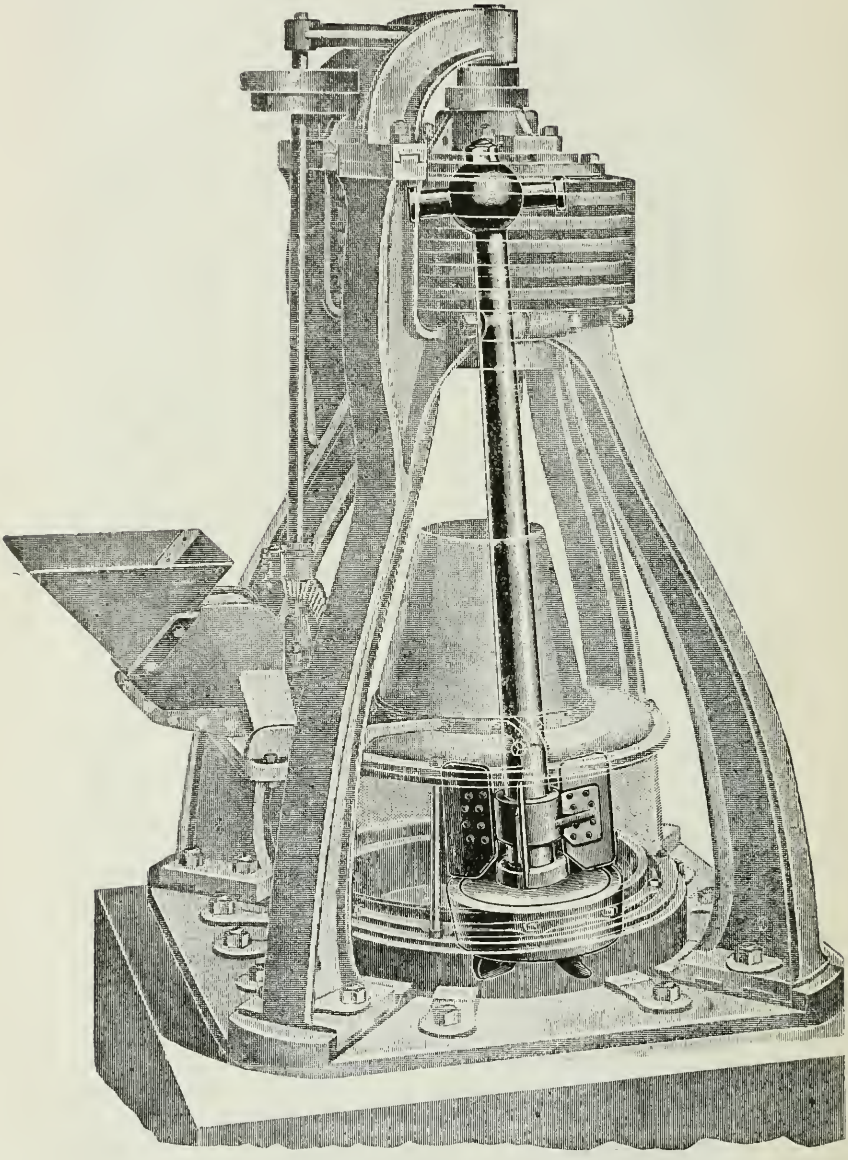


The Great Boulder—1902.

of slots into a receptacle below, whence it is continuously removed by means of a spiral conveyor. The pans and screens are encased in an iron cover, while on top is fastened a conical shield, through which the shaft works. At the Boulder, the annular space between the shaft and the top of the shield is usually covered by a piece of bagging to prevent dusting. The mills are fed automatically, and on the under side of the roll or disc are ploughs or shoes, which prevent the ore from packing below. They scoop up the uncrushed material, and throw it between the ring and disc. The disc, which is capable of rotation on the shaft, travels round the ring, revolving in an opposite direction to the swinging circle described by the shaft. The pressure between the roll and ring is said to amount to about 6000lb.

A remarkable feature about the machine is the fineness of the product from a comparatively coarse screen. For instance, at the Boulder, where a 15 woven wire mesh is used, it is found that 80 per cent. will pass through a 120-mesh screen. The total weight of the machine is from 6 to 7 tons, the ring or die being the heaviest portion—about 400lbs. The pulley is driven at a speed of 135 to 150 revolutions per minute. The cost of the mill in Melbourne is between £500 and £600. Altogether for fine crushing or sliming it is the most effective mill known, and the experience at the Boulder shows that each mill requires 25 horse-power, and that the crushing capacity is 26 tons per day. At the Perseverance each is said to crush from 30 to 40 tons per day; in this case the ore is ground afterwards. At the South Kalgurli 20 tons per day is treated, so that the average of one ton per hour may be looked upon as fair work on Boulder ore. The cost per ton for wages and repairs amounts to about 1s. 10d. per ton, power not included. The mill requires to be fed with an even product; otherwise a breakage of the shaft may occur, and owing to the feed being in one place, a greater wear takes place on the ring here, thus necessitating its replacement or moving it round. Another curious and yet somewhat serious drawback to it is the number of explosions which sometimes occur through portions of unexploded charges of dynamite being fed in with the ore. At one mine, as many as 13 took place in two shifts, resulting in either blowing out the screen or the bottom of the pan. The same material will go through a battery or even ball mill without explosion, but in the Griffin mill will never fail to go off. This in itself serves to indicate the heat and enormous pressure produced within the pan.

The Great Boulder has ten Griffin mills. The dust is drawn away through a zig-zag pipe by a fan and settled in a large chamber. The pulverised ore is conveyed by a spiral conveyor to a double push conveyor 180 feet long. From this the ore is distributed to 12 shoots, each leading to an Edwards' furnace. The furnaces are automatically fed with 16½ tons of ore per day. The furnaces have been described in connection with the article on Messrs. Edwards and Co.'s works. They are of the standard type, 64 feet long, swung on trunnions, and set for the local ore with an elevation of 15 inches. The upper rabbles are plain cast-iron; the lower ones are water-jacketed. When wood is used for fuelling, 15 per cent. of the weight of the ore is found to be necessary. It was discovered in roasting this ore that the furnace was unnecessarily long for the operation. If any roasting went on in the upper portion the



30-inch Griffin Mill arranged for Dry Pulverising.

lower part became too hot and led to fusion of the ground material; the ore in itself, containing such a small quantity of sulphur, supplied such a small amount of heat to the furnace that the conditions which held with concentrates did not apply. In order to supply sufficient heat, so that the temperature should be high enough to induce the necessary chemical changes, recourse was had to gas firing. This was supplied through the crown of the furnace at various points, and has overcome the difficulty. There are three producers of the Dowson type. They are fed with coal, saw-dust chips, and like materials. The gas so produced is led away to any part of the furnace which requires it. The roasted ore contains from 2 per cent. to 3½ per cent. of sulphur as sulphates, and .18 per cent. to .2 per cent. of sulphur as sulphide. The Edwards furnace is as good as any on the field, but it has all the faults common to roasting furnaces. There is no method for the conservation of heat, almost the whole of which is wasted. Reducing gases from the fuel are allowed to come in contact with the ore when they should be rigorously excluded; the amount of air passed in bears no proportion to the work it has to do. The metallurgist of the Great Boulder, not satisfied if a better furnace could be introduced, supervised the roasting of a parcel of ore at Merton's metallurgical works, Spottiswoode, Victoria, and was so well pleased with the result that the management decided to instal six others.

The result of the roasting by Merton's furnace on a parcel of 50 tons from the Great Boulder mine led the metallurgist of the mine, Mr. Geo. M. Roberts, to recommend the adoption of six of the furnaces. These are now installed at the mine. Merton's furnace consists of three hearths, superimposed above each other, each hearth being 23 feet long, 8 feet wide, and about 12 inches from the floor of one hearth to the bottom of the next, the hearths themselves being about 12 inches thick. The ore is delivered automatically into a hopper on top of the last hearth near the flue end; from the hopper it is fed on to the first hearth. It is worked automatically by rabbles until it comes to the end; from this it may be dropped through a discharge chute on to the second hearth, along which another set of rabbles cause it to travel towards the flue end; from this it drops on to the third floor, and it travels along this towards the fire-box end. It drops through a chute on to a short fourth floor, where it either can be in the hottest part of the furnace, or be worked with a single rabble, after which it may be discharged.

The modification of this furnace for silver roasting or chloridising is arranged by having a chute from the last hearth mentioned discharging into a revolving cylinder, which has a movable fire-box at one end.

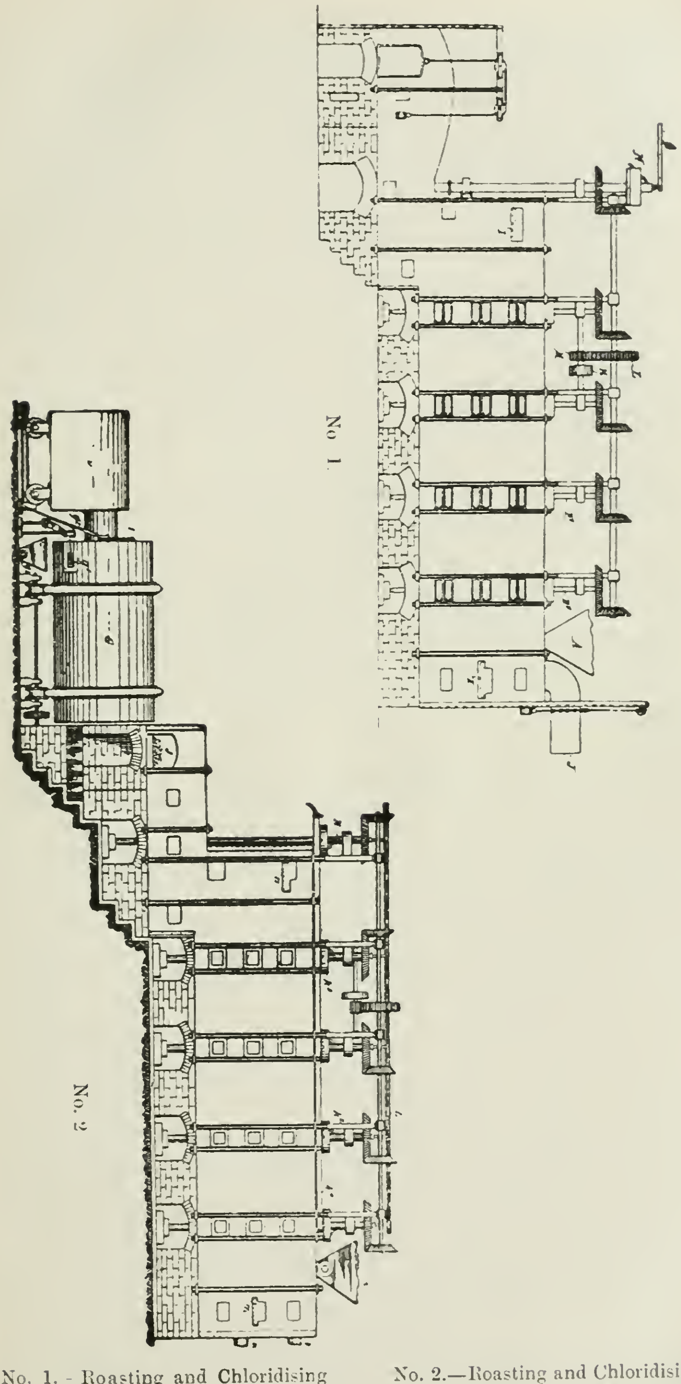
There are four rabbles working on each hearth, each being 4 feet in length. Three revolve in one direction, and are always in the same or in parallel lines; the fourth revolves in the opposite direction, and is at right angles to the others. The rabbles are attached to vertical shafts, of which there are four, through the three-hearth furnace, and one through the single hearth. Those through the three hearths are 5 feet apart, and pass through the bottom floor to an archway, where each is supported on a footstep. They are all water-jacketed, and are driven at a uniform rate by bevelled cogwheels attached to a horizontal shaft working into corresponding wheels on the upper end of the vertical shaft. Provision has been made for removing or replacing rabbles without having to cool down the furnace. Inspection doors are provided on each floor opposite the rabbles. The furnace is stayed by means of railway rails placed as girders, and connected together above and below the hearths.

The cylinder on the chloridising furnace is 10 feet long and 6 feet in diameter. It is lined with firebrick, arranged so as to taper from the feed to the discharge end. The movable fire-box is a brick-lined furnace set on wheels. The flue passes into the centre of the end of the cylinder. This is simply a modification of the well-known Bruckner cylinder.

The air is allowed to pass into the ordinary furnace through the fire-box, where it is partly consumed; the excess of oxygen in air, the nitrogen, and products of combustion pass over the ore lying on the last hearth, thence over a short bridge along the third hearth. At this point, if desired, the gases may be sent through the flue without traversing the upper hearths at all. Otherwise, as is usually the case, the air passes back over the second floor, and escapes through a flue capable of being closed by a damper on to the upper floor, from which it finally passes away.

In the chloridising furnace the fire-box of the ordinary furnace is used as an auxiliary source of heat. The ore, after leaving the fourth hearth, slides into the cylinder, where salt is added, and the gases from the movable fire-box pass through and over it. Air is admitted also at the end of the cylinder.

The advantages claimed for this furnace are its compactness, simplicity of construction and working, its large output, and heat economy. The space occupied is only 34 feet by 10 feet, the top of the first hearth only being about 9 feet above the ground. The products of combustion passing, as they do, over and under the hearths, tend to keep them at an even temperature. There is no great amount of heat radiated from the furnace. By carefully regulating the supply of air roasting can be so arranged that the heat from the waste gases can be taken up by the raw ore fed in. The difficulty with regard to material becoming too hot in the upper floor, through the access of a large quantity of hot air combining with the pyrites, is got over by allowing the bulk of the heated air in such a case to pass away into the flue. Should an ore require to be roasted in stages this furnace offers facilities for doing so, such as many continuous process furnaces do not possess. In Edwards' furnace, for instance, the rabbled material moves down, and there is a possibility of part of the ore from No. 1 rabble passing on to No. 2 rabble, and if it passes to No. 2 the chances are that it may pass to No. 3, and so on be transferred from one end of the furnace to the other in as many revolutions as the furnace has rabbles. Though in practice this is not found to take place to any appreciable extent, any such possibility may be dismissed from consideration in dealing with Merton's furnace. The ore may be kept on No. 1 hearth until certain desirable changes occur, such as the transformation of the pyrites into the magnetic sulphide and, in part, magnetic oxide. On the next floor the sulphur may be almost wholly removed and part of the magnetic oxide transformed into ferric oxide. On the last floor this change may be com-



No. 1.—Roasting and Chloridising Furnace for Gold.

No. 2.—Roasting and Chloridising Furnace for Silver.

Merton Ore Roasting Furnace.

pleted, while in the finishing floor any sulphates of the weaker bases may be decomposed, should this be desirable.

An ore like that from the Boulder does not roast in the same way as the ordinary pyritic one. The finely-powdered ore clings in dusty clots; it is light and fluffy, and like flour will not flow freely. Its penetration by air is slow, while any attempt to stir it vigorously or to cause it to flow in heated air leads to an inordinate amount of dusting. From a few experiments made with similar material, it was found that as soon as the calcium carbonate was decomposed the resulting oxide was acted upon by sulphur dioxide, giving calcium sulphite. Calcium sulphide also formed from the inter-reaction of pyrites or ferrous sulphide and calcium oxide. These in their turn slowly oxidised to calcium sulphate. Certain it is that the Boulder ores give on a bad roast a considerable quantity of alkaline sulphides. This may be due to the reactions given above, or more probably to the alkalies and alkaline earths in the ore reacting with ferrous sulphide direct. The product from all the furnaces on the field has a slight alkaline reaction.

The method of testing the roasted product is done, as I was informed by Dr. Earp, B.Sc., F.I.C., and Mr. Wright, of the Perseverance mine, by adding lead carbonate to the aqueous extract from the ore; this, of course, shows soluble sulphides if the black or brown lead sulphide is produced. A further extension of this is to boil the ore with caustic potash and then filtering; lead carbonate added to this solution will show the presence of sulphides, which may not be soluble in water. I have confirmed this test, which is a very useful one, so far as ferrous sulphide is concerned, although finely ground di-sulphide or pyrites only gave a faint sulphide reaction.

The quantitative method of ascertaining the state of the roast depends upon the termination of sulphur present as sulphide. This is done by determining the total amount of sulphur present by fusing the material with sodium carbonate and some oxidising agent, thereby transforming the whole of the sulphur to the form of sulphate of sodium. The fused mass is disintegrated and filtered; the filtrate evaporated to dryness with HCl to render silica insoluble. The solution is taken, treated with barium chloride, and the sulphur determined from the weight of the barium sulphate produced. The sulphates are determined by boiling some of the ore with sodium carbonate; this serves to transform the sulphates into sulphate of sodium. The ore is filtered, the filtrate acidified, evaporated to dryness, and the solution treated as before, and the sulphur present as sulphate determined. The difference between the two amounts of sulphur is attributed to sulphides. The process can be shortened materially from that given in this description, but it cannot be said that such a mode of determination is altogether satisfactory. The difference in the two amounts is but small, and this small amount has to carry all the errors of both determinations. Further, the presence of a small quantity of barium sulphate in the ore would not be decomposed by boiling with sodium carbonate, and the sulphur in it would consequently be counted as sulphide.

It must be confessed the estimation of the sulphur as insoluble sulphide is not an easy matter on such an ore as this. The method of fusing with a kown weight alkaline carbonate and some oxygen-supplying compound, and estimating the alkalinity afterwards, fails in the presence of easily decomposable sulphates, e.g., $Ca\ SO_4 + Na_2\ CO_3 = Ca\ CO_3 + Na_2\ SO_4$; each molecule of calcium sulphate neutralising one of carbonate of sodium.

As the sulphur as sulphide has not only all the errors of determination, but also all the sins of a bad extraction laid on its shoulders, it is necessary to state that this is not the case, but that all other reducing agents must be looked upon as equally as bad; and even the oxidation of such agents may not suffice to correct the evil. The sulphur as sulphide, then, can only be looked upon as an accompanying evil, and a simpler test might be devised to show the state of the roast relative to the consumption of cyanide. For example, the KOH extract might be acidified when diluted, and its reducing power estimated with a standard solution of $KMn\ O_4$.

The same samples might be tested with KCy shaking test to determine the consumption of cyanide. From the two on ores of similar nature a table could be readily constructed showing the relationship between the reducing power of the ore and the consumption of cyanide. The most direct test as to the state of the roast is tried now at most mines, and also at the Great Boulder. This is the shaking test or the actual test with cyanide on a sample of the ore.

Although such tests are useful for the same classes of ore, and while they may give good results so far as showing the state of the roast is concerned, they may be of little or no use in guiding the metallurgist as to his probable extraction. Very often a roast that will respond to all tabulated tests will fail to give a good extraction, and one reeking with basic and other sulphates, which the textbook authorities are so strong on, may be made to give a first-rate extraction. The cause of the non-extraction of fine gold by solutions is either due to the destruction of the solvent by some other agent in contact with the gold or by want of contact of the solvent with the gold. The latter includes cases of bad percolation where certain patches of ore are not acted on at all, and the more common cases of the gold being locked up in some other material, either physically or chemically. It may exist in fused oxide of iron, or in some easily fusible silicates; it may exist in a telluride naturally, or may be locked up in some metal or metallic sulphide formed in a roasting furnace. In the case of some solvents, such as chlorine, the outer coating of many of the reducing agents will be destroyed, and the gold, whether locked up physically or chemically, will be dissolved, the exceptions to this being the case of an insoluble compound forming and enveloping the gold, such as chloride of silver. Even this material does not prevent the entire solution of the gold, provided the latter is present in the alloy to the extent of 60 per cent. Should the particles enclosing the gold be too coarse, then the solvent action may be too slow, even if in course of time a perfect extraction might be obtained. While the solvent is dissolving the particle it is being destroyed as such, and consequently can not attack the gold, so that there must be an ever replenishing supply to all such particles. It is very often forgotten by the metallurgical engineer that he is dealing with masses which may only be very imperfectly represented by molecular formulæ. He forgets that every change that takes place is between molecule and molecule, and that in the smallest particle visible there are some myriads of molecules, and in the time given for action a comparatively small number may be acted upon. Chlorine is able to penetrate and dissolve gold out of material impervious to cyanide; so that when the latter solvent is used on rich ore which is likely to frit in any way, it is necessary to grind the product very finely, and thereby expose as much of the metal as possible. This is an accep ed axiom on the Kalgoorlie field.

After leaving this furnace the roasted material is sent along by means of push conveyors, two for every six furnaces, to a Krupp bucket elevator. There are two of these elevators, each delivering ore continuously into a mixer. Each mixer is a small steel vat, 4 feet in diameter, and 2 feet deep, fitted with a movable cover, through which a shaft fitted with paddles works. There are six discharge outlets near the top of the mixer, through which the pulp flows into the amalgamating pans. The hot sand is churned up with the stream of solution supplied from the wash water and excess cyanide solution from the filter presses. The object of the mixers is to damp the ore without creating much dust; in this they are partly successful. The pulp flows from the mixers into twelve grinding and amalgamating pans. These are of the usual Wheeler type, and are arranged to have a continuous overflow from sets of three into a large settling pan. As mentioned in a previous article, the contact of the dry hot sand with water causes a solution of many soluble materials present in the roasted ore. The two which have the most influence on subsequent operations are calcium and magnesium sulphates. The former exists in the mine even at low levels, in the form of gypsum $CaSO_4$ $2H_2O$, while it is also produced from the interaction of calcium carbonate, sulphur dioxide, and oxygen of the roatsing furnace. The magnesium sulphate is produced in a similar manner.

The anhydrous calcium sulphate is very slowly ac ed upon by water. It is practically dead burnt plaster of paris, but in course of time it becomes $2CaSO_4$ H_2O, or the ordinary plaster of paris, which, under ordinary circumstances, sets very rapidly, becoming $CaSO_4$ $2H_2O$, or it goes back to gypsum again. This action is more or less retarded by the presence of other salts, but appears to be going on at almost all stages of operations at the Boulder. The settling takes place to such an extent in the pans that all parts become coated with a crust of material which has to be chipped off from time to time. For the same reason, it is useless hanging amalgamated copper plates in the pans, the amalgamation being done by adding large quantities of mercury, somewhat after the fashion of silver amalgamation, and periodically replacing that amalgamated. Whether the crystals of gypsum, as they form, are able to lock up gold and prevent it from further attack I was not able to discover, but should this be the case it would be preferable to allow sufficient time for the crystals to form and then grind them, it being obvious that when once the compound formed regrinding would reduce it to a slime, which would not re-set. The chippings from the pan bottoms have a high value, but this is due to the amalgam entangled. The pans are cleaned up twice per month. By starting at one end of the series and doing one pan per day, then on the thirteenth day the first pan would be again cleaned up, and so on.

The amalgamation which takes place is very good, while the loss of mercury is small. This is no doubt due in a great measure to the fact that hot alkaline cyanide solutions are used in place of water in the pans, thereby keeping the mercury and gold bright, and also to the fineness of the product fed into the pans. A pulp which contains from 2½ to 3 parts by weight of water to 1 of dried slime has been found to be fluid enough for the operation. Under ordinary conditions a pan is not a suitable amalgamating contrivance if worked with a continuous supply and discharge. In the first place the finer particles, including the fine gold, are washed out without ever getting to the bottom or coming in contact with the mercury. Secondly, the fine gold present only amalgamates with great difficulty, and even if amalgamated plates are hung on the baffles or sides of the pan the bulk of the fine gold will pass out. Thirdly, a reducing action appears to be always going on, and hydrogen is always evolved in appreciable amount, while with arsenical ores arseniuretted hydrogen is given off freely. Fourthly, the pulp has to be so diluted that practically all the mercury fed in periodically immediately falls to the bottom, where it becomes covered with all the heavy metallic minerals present. It should not be necessary to point these simple facts out, but in parts of Australia the system of pan amalgamation is still believed in as the most perfect of all methods. It is a different proposition altogether to take a thickened pulp and have globules of mercury desseminated evenly through it, to add chemicals to assist amalgamation, and then to thin down so that mercury and amalgam sink to the bottom and the thin fluid pulp overflows. The pan amalgamation process at the Boulder serves to eliminate the coarse specks of gold from the pulp, while only the finest or that capable of being dissolved by cyanide solutions in a limited time is carried over into the settlers. Solution of gold is, in fact, going on all the time. The amount of gold recovered as amalgam varies from 30 to 60 per cent. The amalgam caught by the pan is treated in an amalgamacing barrel and thereby cleaned. The cleaned, well squeezed amalgam contains from 25 to 33 per cent. of gold. The pulp overflows from three pans into a settler. This is of the usual type, having suspended paddles working from a vertical shaft. The duty of the settler is to allow the fine globules of mercury carried over to subside. The pulp is so fine when it leaves the pan that 98 per cent. of it will pass through a 120-mesh screen, the balance through a 100-mesh. From the settlers the pulp passes into a sump, from which it is pumped up 48 feet to two distributing tanks, each having four discharge holes leading to four small conical spitzlutte, from which it overflows into a series of spitzkasten. The former serves to eliminate any coarse or heavy particles which may have come over with the slime. These are led back again into the pans; the latter are for the purpose of thickening the pulp and clarifying the solution. So effectively do the spitzkasten do their work that clear water is drawn away from the top of the final one, while a pulp having a consistency of 1 of water to 1 of slime by weight is drawn off below. A detailed description of these will be given in a later article. The clear water from the spitzkasten is led back to the mixers, while the thickened pulp is led into the agitation vats. The strength of the solution is now made up to from .01 per cent. to .08 per cent. KCy, and the paddles are started and as a rule kept going from 16 to 17 hours.

Tests are frequently made to see if the gold is passing into solution. Since the weight of pulp and solution is nearly equal, by taking samples and filtering and determining the amount of gold in the solution, the decrease in the slimes may be readily inferred. This result is checked from time to time by assaying the slimes themselves. The ordinary method of determining gold in cyanide solutions by evaporating to dryness with litharge, and then running down the lead button and cupelling, is not looked upon with favor by some of the metallurgists on the field. It is held that large losses take place on account of the solutions being saturated with chlorides and other salts. Mr. Goldstein, of the Great Boulder No. 1, is positive that this method of assay is misleading, and seeks to lessen the error by adding charcoal as well as litharge to the solution to be evaporated. He also fuses at a low temperature.

The gold dissolves readily at first, and then solution takes place more and more slowly, until the commercial limit is reached. The state of the roast is the essential factor governing the success or failure of the process. It is found that if alkaline sulphides are present not only does the solvent action of the cyanide cease, but that a partial precipitation of gold takes place. For instance, at one of the prominent mines, the following results were obtained:—

The solutions flowing into the settlers contained 40 grains of gold per ton. The solution was then made up to the desired strength with cyanide. One hour after adding KCy solutions assayed 50 grains per ton; 4½ hours, 42; 8¼ hours, 40; 12¼ hours, 37; 16¼ hours, 25; 28¼ hours, 25. Had the solution gone on in the regular way, the value would have been about 170gr. per ton. This sample was found when tested with the lead carbonate method already mentioned to give a sulphide reaction. The method adopted for the quantitative determination of sulphur is based upon the qualitative test already described. About 5 grm. of ore are boiled with about 50c.c. of 5 per cent. NaOH for about 5 minutes. This is filtered, washed, and the filtrate diluted to 200c.c. with cold water. It is then made faintly acid with acetic acid and titrated with decinormal iodine solution. The reaction $H_2S + I_2 = 2HI + S$ is quantitative, and the end reaction with starch as an indicator is sharp. This method will indicate not only sulphur, but any other compound which can be raised to a higher state of oxidation by iodine. Further, it assumes the decomposition of all sulphides by the caustic alkalies. By this method of indicating sulphur, only 0.03 per cent. was shown to be present in the case of the ore, which refused to yield its gold to the KCy solution. Assuming that the sulphur is present in the state of soluble sulphide, and that the following equation is a quantitative one—$K_2S + KCy + H_2O = KCyS + 2KOH$—then each atom of S destroys a molecule of KCy, or double its weight, so that 0.1 per cent. of S means 2.24lb. of S and a destruction of 4½lb. of cyanide per ton. The addition of soluble lead salts to the solution, in order to precipitate the sulphur of the soluble sulphides as sulphide of lead, does not appear to be carried out at any of the mines. A test which has to be carried out regularly is the determination of the alkalinity of the solutions from the ore. Now, as water is not used for moistening the sand, but dilute KCy solution, the problem is to determine that of the liquid, independent of the KCy present. This is done by running in $AgNO_3$ solution, using KI as an indicator, and taking it to opalescence. This fixes the KCy; a few drops of alcoholic phenolphthalein solution is added, and the alkalinity indicated. This is titrated with decinormal HCl. The protective action of an alkaline on KCy is now very well known, and care is always taken to see that there is always free alkali present. If the solution from an ore is neutral a small amount of lime is added.

What the effect of moistening finely divided hot ore with KCy solutions is cannot be definitely stated, but it would seem from some instructive experiments carried out by Mr. S. Radcliff, the metallurgist to the Chance mine, Cassilis (V.), that a great deal of cyanide is destroyed.

Mr. Radcliff was struck with the amount of KCy decomposed on running a solution through dry sand, and found the finer the sand the higher was the consumption. In order to find the relationship between the size of the particles and the consumption of KCy he powdered some clean glass and sieved it through various sized sieves. He determined the amount of cyanide in his original sample, the amount retained on the glass and the amount oxidised to cyanate. 25c.c. was used in each case; this was washed with 25c.c. distilled water.

Size of Particles.	KCN originally present. grm.	KCN after passing through. grm.	KCN oxidised to KCNO. grm.	KCN left on glass. grm.	KCN not accounted for. grm.	Per cent. converted to cyanate.	Per cent. left on glass.
Through 40 sieve and on 60	.2075	.1880	.0160	.0026	.0019	7.7	1.25
,, 60 ,, 80	.2075	.1833	.0193	.0034	.0015	9.3	1.6
,, 80 ,, 100	.2075	.1720	.0305	.0038	.0013	14.62	1.87
,, 100 ,, 120	.2075	.1710	.0318	.0045	.0003	15.3	2.10

It would therefore seem that there is a condensation of oxygen on finely-divided material, which is able to transform as much as 15 per cent. of cyanide of potassium into cyanate, which in its turn is transformed into potassium bi-carbonate and ammonia.

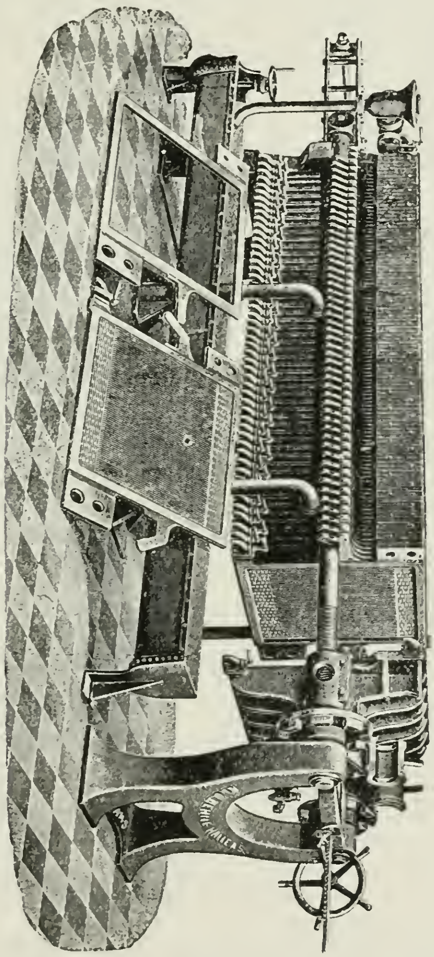
$$KCNO + 2H\ O = KHCO_3 + NH$$

The method of treatment at the Great Boulder of sending back the water-washes from the filter-presses, in which there is a minimum amount of cyanide, is one to be recommended.

The treatment of the pulp after its gold has been dissolved in the agitators is an adaptation from modern chemical practice. First of all the montejus, of which there are four, into which the slimes from the agitators pass, are modifications of the acid egg, so well known in the manufacture of sulphuric acid. Each monteju has a capacity of 500 cubic feet, and is simply a steel pressure tank, fitted with a pipe through which air may be delivered, and another pipe reaching nearly to the bottom and leading to the filter presses above. By admitting air under pressure the pulp is forced up the other tube as water is forced out of a wash bottle. The pulp is forced into a filter press, taking but a few minutes to fill it; as soon as the press has been filled, water is turned on to wash the cakes, and finally a current of air to dry them.

A filter press is a mechanical contrivance for separating finely divided solids from liquids; in its simplest form, it might consist of three square or rectangular frames, like the frames of a schoolboy's slate, tightly clamped together, the press being solely for the purpose of drawing the frames together, and not, as might be imagined, of forcing solutions through. The middle frame is open —that is, a frame without a slate in it, this is indicated by C; on one side of this open frame is another (A), which contains a corrugated plate instead of a slate; over each side of the corrugated plate lies a sheet of punched iron, or a screen; on the other side of the open frame lies another frame (B), similar in construction to A. A sheet of filtering cloth is doubled over the open frame. If water and slime is forced through a lateral opening into the open frame C, then the liquid will escape through the filter cloth, then through the screen, and find its way between the corrugated plate and screen. By providing a lateral tap on this frame, water may be drawn off, and the open frame filled with a cake of slime. By closing the lateral opening through which the slime entered, and forcing water through a lateral opening between the corrugated plate and the screen in plate A, the liquid will pass through the screen, then through the filter cloth, then through the slab of slime in the open frame C, through the filter cloth on the other side, then through the screen, and find its way down the corrugated plate B, when it may be drawn off. Similarly air may be forced through to displace water or solutions. Instead of three frames, a great many are clamped up together. All of these have projecting lugs on their corners. Through these lugs holes are bored, which form a continuous pipe when the frames are in position. Lateral openings are made into each frame from one or other of these pipes, in accordance with its purpose. For instance, the slimes run only into the open frames. Solutions or wash water passes into the A frame, and air escapes from it; while from the B frame only solution escapes. Both A and B frames are provided with taps for the escape of liquids. Each frame must be planed up accurately to minimise the danger of leakage; also, strips of tarred blanket are laid along the faces, so as to serve as packing. The filter cloths which lie over the open frames are stitched together in two or three places, where they project, so as to prevent them giving when the pressure is applied. The whole of the frames, which are vertical, are carried on horizontal bearers by means of projecting arms; they are brought together by means of powerful screws, or by hydraulic pressure.

When a press is empty, slimes are forced in, either by pumps or by the compressed air of the monteju. The mud flows along the slime passage pipe, and rushes into the open frames; air is displaced and the solution starts to run through. Taps are turned off, and the liquid finds its way through the filter cloths, thence through the filter frames; in from 15 to 20 minutes, a press is full of hard caked slime. The cakes are next washed. Connection with the pressure tanks is cut off, wash water is forced into a frame on one side of the cake, this passes through the cake, and escapes on the other side, the pressure applied being about 80lb. per square inch, the time taken for washing being about 20 minutes. Air is then passed through in the same direction as the water. This serves to displace nearly 50 per cent. of the water that would otherwise remain in the cake. Even after air washing, they contain from 15 to 20 per cent. of moisture. The press is then opened by unscrewing or releasing the hydraulic pressure. The frames are slid along the bearers; the slabs of semi-solid slime are knocked out; these fall into trucks below, the filter cloths scraped wherever necessary, the frames brought together, and tightened up again, the press then being ready for another charge; the time taken for emptying and closing being about 20 minutes, or an hour in all. As an actual example, a solution, containing 0.17 per cent KCy before agitation, contained 0.09 per cent. after treatment, and 0.05 per cent. after washing the cakes. The slimes assayed 5dwt. before



Filter-Press—Open.

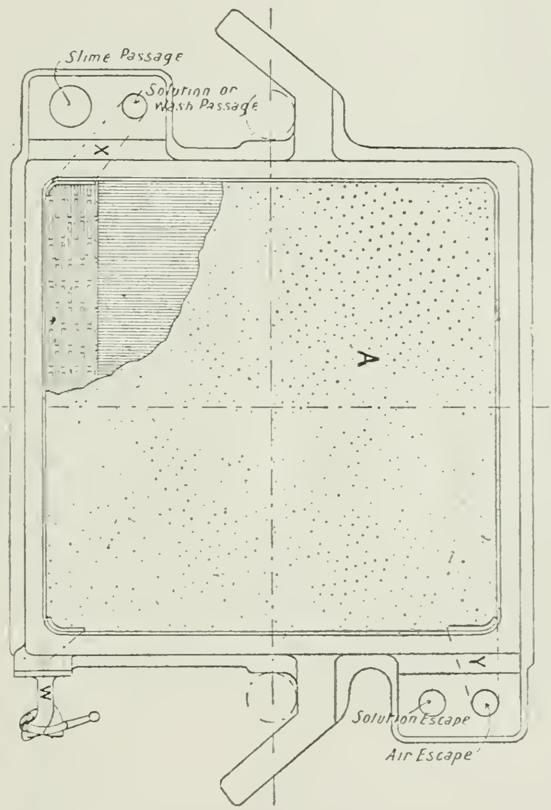
treatment, and 1dwt. after treatment: the consumption of cyanide being 47 per cent, the displacement of KCy solution by water 44 per cent. The actual cost of labor in filter pressing is not high. Two men, paid at the rate of 11s. 9d. per shift, deal with 10 presses of four tons capacity each in that time, and if they had the hydraulic closing apparatus, as supplied by Martin and Co., Adelaide, they would do two more, so that 48 tons can be dealt with at a cost of 23s. 6d., or less than 6d. per ton. This, however, is only the approximate cost of one part of the operation.

The subject of filter pressing is a very important one. Mr. J. W. Sutherland, general manager of the Golden Horseshoe Estates, deserves great credit for having been the first to introduce them on this field, and for having shown that they were suitable for slime treatment on an extensive scale. The result of this is that they have been adopted by all the large companies on the field, and they are used extensively all over Australia. The success is all the more noteworthy, since, I believe, they were tried in South Africa, but deemed unsuitable. The original method of using them was to pump slimes direct into the presses, force the cyanide solutions through in order to dissolve the gold, and then to force wash water through to displace the gold solution. The introduction of the agitators for carrying on the solvent action in was an apparent improvement on the original method, the presses only being used for filtering purposes. The introduction of the montejus came later: but Mr. Sutherland stuck to the three-throw pumps, and these have displaced the montejus at the Great Boulder also. It was held by many that the presses were not so evenly filled by the pump as with the montejus; but this is not so. Mr. R. Hamilton kindly supplied the additional information as to the working of the pumps: "The pump for the filling of the filter presses acts very well now that provision is made for safety appliances to prevent extra pressure of the pump opening the hydraulically-closed presses. If we exceeded a pressure of 85lb., the press would begin to open. In order to avoid this, we have a lever safety valve, after the style of the old-fashioned steam boiler valve, but with sheet rubber face on the valve instead of being metal to metal. We have also an arrangement whereby an attendant can open a by-pass in order to reduce the pressure. We have also a float valve to control the charging pipe from the vats to the sump; this valve works automatically by means of a float, and is also controlled by a lever from the second floor, so that if necessary the attendant can shut off the product into the pump sump at will. The method is much more economical than filling by means of compressed air." The pump most favored by Mr. Sutherland, and which is being generally adopted, is a three-throw plunger, with 9 inches to 12 inches diameter cast iron plungers, twelve to twenty-four inch stroke, the whole erected in a cast iron frame, and belt driven with fast and loose pulleys. The valves most serviceable being the ordinary clack, with hard rubber, or piece of old belting for wearing face against an iron seat, the whole being simple and wearing parts interchangeable.

The cakes from the presses have an even structure throughout, and it is remarkable to see how they split along a central plane parallel to the frames. It would appear as if this central portion was the last filled with slime. Some naturally occurring argillaceous ores cannot be treated successfully with the ordinary filter press, while, if the ordinary slimes contain much gritty or coarse material, a great deal of trouble is caused by the excessive wear and tear and imperfect washing.

The filter presses first adopted were small, having cakes from 1½ inches to 2 inches in thickness. The cakes now are 3 inches thick, and 40 inches square: while as many as 50 frames occur in one press. It is hardly likely that larger presses on the same principle will be used, for there is a limit to the size. Dehne's and Martin's are the only two in use on the field.

One of the greatest faults with the press is that continuous automatic work stops as soon as the slime enters the press. Men have to be in attendance to shut off the cocks, to open the press and knock out the cakes, to scrape the filter cloths and close the press up again. Much trouble is occasioned by the screws for tightening up the frames not being so nicely adjusted that the men can tell



Solution Plate for Filter-Press.

when each is perfectly closed. Each set of men must know their own press and stick to it. At the Great Boulder, this trouble is for the most part got rid of by using hydraulic pressure. The press is closed evenly and tightly in a few seconds. If the filter frames were not properly set, or each joint in the frames not properly cleaned out, a leaky press would be the consequence, and muddy instead of clear solutions would run out. All the adjuncts of the process, such as agitation vats, pressure tanks, air and hydraulic pressure vessels, are expensive to instal and costly to maintain, and they are necessary for a small, as well as for a large, plant. The presses themselves are costly, and. for a mine, somewhat fragile

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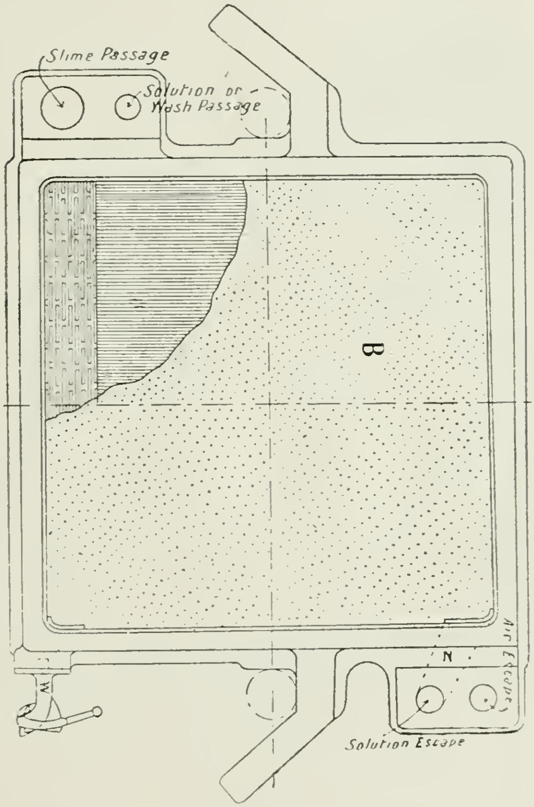
pieces of mechanism. It is doubtful if perfect washing can be given to the cakes of slime in the time allowed; from the mode of washing the tendency of solutions will be to escape in the easiest way through the cake, so that if sandy or gritty materials are present, and not distributed in even layers parallel to the filter frames, the wash water will pass through these channels in preference to more impervious portions in the cake. Air pressure will only remove the water from the most penetrable portion. On the other hand, the necessity for sliming Boulder ores in order to obtain a good extraction had to be met by adopting some new process. The usual decantation method was not suitable on account of the enormous volume of water required and the want of a suitable dumping ground. The filter press gives a cake which is coherent enough to be handled and stacked up on the tailings dumps, while the cakes when water washed contain only 40 per cent. of moisture, and when compressed air is forced through them less than half this amount. A considerable saving of KCy is also effected by using a minimum of water, and work can be so arranged that no solutions are run to waste. A further great advantage is that they are clean in their work. Instead of the slimy, sticky mass of mud in the decantation process, clean cakes are turned out, and the amount of slime each press can treat per day is great compared with its capacity. With ten presses, having a capacity of about 4 tons each, 350 tons per day may be easily dealt with. On seeing the way in which slimes are settled in the series of spitzkasten, and solutions almost clear are drawn away, one is forcibly impressed with the possibilities of some modification of this system coming into use for slime treatment. The decanted solutions could be run through a clarifying press (which would take a long time to become filled with slime), the clear solution sent through the zinc boxes and returned to wash the thickened pulp in a lower series of spitzkasten. With some such simple system the present elaborate filter press method could not compete on fields having different conditions to Kalgoorlie. It is hardly likely that the sliming of all the ore, as practised on the field, will be adopted anywhere else, except by those who slavishly copy without understanding why they do.

The cost of treatment at the Great Boulder mine is now as follows:—

	s.	d.	s.	d.
Crushing and delivery of material	4	7.28		
Roasting (Merton's and Edwards')	3	9.83		
Amalgamating	1	6.88		
Supervision (including salaries)	0	3.81		
Assaying, electric light, clerical, etc.	0	4.51		
			10	8.31
Cyanide Mill—				
Mixing, agitating, and filter pressing	4	6.56		
Disposal of residues	0	11.44		
Precipitating	0	2.92		
Supervision (including salaries)	0	2.23		
Assaying, electric light, clerical, etc.	0	4.01		
			6	3.13
Total			16	11.44

The clear solutions from the filter presses pass through the zinc boxes, thence to the grinding pans; the second wash water from the presses goes direct to the pans. Many details have to be attended to in order to get such an elaborate plant as this to work smoothly from one end to the other. The "montes," or pressure tanks, where used, should be on a lower level than the presses, so that the slimes may be forced up a sloping pipe to the presses. By this means, when the press is full the mud filling the pipe runs back into the monte instead of choking it up, as has been the case with some

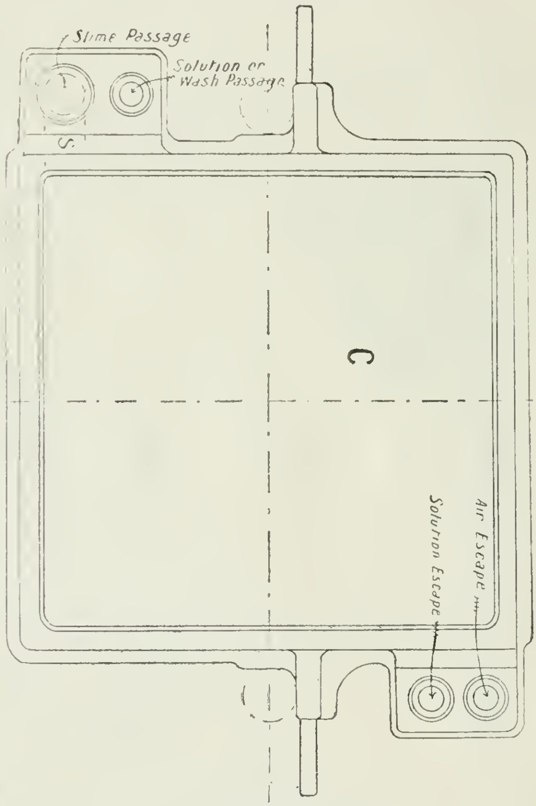


Air Plate for Filter-Press.

plants. In addition to the dry crushing plant, a battery of 30 head of stampers was used for milling lower grade ore, or that free from tellurides. The stone as crushed by the large breaker was fed into a truck and tipped into a hopper behind the battery; an automatic feeder delivered the stone into each box. Mercury was fed in at regular intervals, and the crushed pulp passed through a 24-mesh screen and thence into a mercury well, then over amalgamated copper plates into a Wheeler pan. There were six pans, working continuously. Each two pans discharged into a settler, thence over canvas strakes, the tailings going to a settling dam.

This material was cyanided without roasting, the slimes being filter pressed as before.

The term slimes is not an easy one to define. The common meaning was that portion of ore crushed in a battery which flowed away with the water. The amount of slimes was usually determined by allowing the sand from a sample to settle for a few moments and decanting the discolored liquor into a larger vessel, repeating the operation until the latter was filled. The finely-divided suspended material could then be precipitated with alum.



Open Plate or Slimes Plate for Filter-Press.

This method gave a slimy levigated product, which was free from grit and heavy coarse particles. The material also set in leathery cakes, which were impervious to water; any attempt to wash, or sand solutions through the slimed cake under pressure only led to the material becoming more closely packed and more impervious. The slimes formed in this way were usually argillaceous or clayey products, and owed their plasticity to the water chemically combined with each particle. When this water had been driven off the material at a low red heat, the plasticity of the material is destroyed, and its slimy nature with it. It is gritty instead of greasy, and even when in lumps solutions will percolate it, and gold may be extracted. In fact, there is the same relation between the material before and after heating as between clay and brick dust. The more usual method of estimating slimes is by means of sieves. Formerly, 100 meshes to the linear inch was deemed a sufficient test; then 120, and now 150; yet this method of classification is not satisfactory. Particles of gold which would readily go through the screen would fall through water rapidly and not be counted in the slimes by the old method of decantation, while the same would hold for many other rapidly falling equal sized heavy grains. The fine grains, whether plastic or gritty, when wet, would all be classed in the same category if sieving is to be the only guide. This is not at all satisfactory, for no accurate comparison is possible when dealing with treatment on various fields. For instance, at Mount Morgan (Q.), where the gold is as fine as it is at Kalgoorlie, and where tellurides are said to be present in the lower level ores, the whole of the ore after dry crushing in ball mills—and when the stuff is very rich they crush very finely—is sent through a roasting furnace, and treated in large shallow vats by percolation. The percentage extraction, according to published returns, is very much better than at Kalgoorlie, while the cost for mining and treatment amounts to 25s. per ton for sulphide ores. There is no more fine material in the Kalgoorlie crushed product than in the Mount Morgan, but the former is a tough rock comparatively soft, which gives a clay-like product when finely ground. The latter is hard, brittle and gritty, and the angular fragments do not pack in such a way as to be impenetrable to solutions.

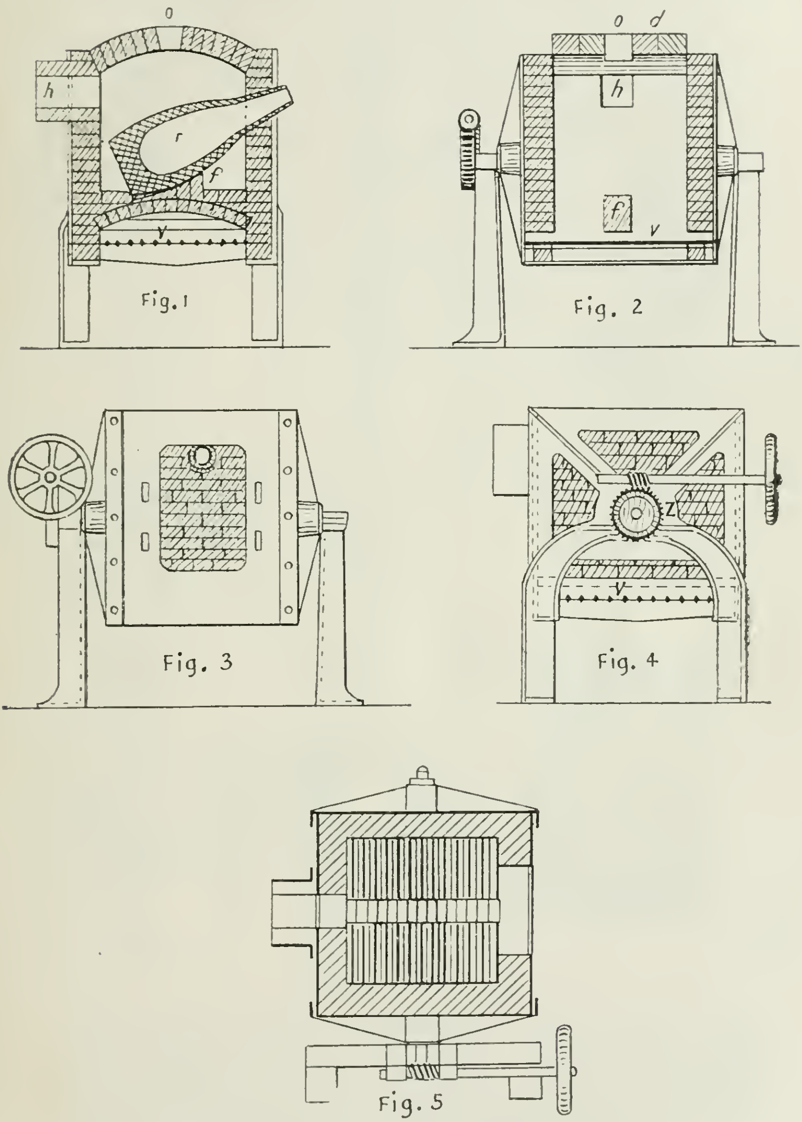
The whole of the Kalgoorlie rock does not become slime even after its passage through the pans—after which the bulk of it will go through a 100-mesh sieve—for it is found this material will admit of percolation of solution, and a considerable quantity of this finely-ground, but still gritty and angular fragments, is treated in ordinary cyanide vats. An extension of the simple system for such material, and the use of the filter presses only for true slimes, would appear to be a more natural course for people who are not in too much of a hurry.

The zinc boxes used on the Boulder field generally are the usual type of rectangular wooden box, with the ordinary partitions. The solutions passing through are very dilute, yet they readily part with most of their gold, those from the sulphide plant running from 10 to 12 grains per ton of solution, and from the oxidised ore 5 grains. It may be that the warm liquors in circulation help to bring about this desirable result. The extraction at the Great Boulder from the battery and its accessory plant was about 80 per cent. Before the telluride came in, the results were 10 per cent. higher. With the sulphide plant, an extraction of over 90 per cent. is obtained, the tailings running from 2 to 3.2dwt. per ton. With fresh water the residues are reduced to from 30 to 40 grains per ton. The tailings themselves are trucked away and stacked in a gigantic heap over 100 feet in height by means of a Lidgerwood flying fox, the cost of dumping amounting to 1s. per ton. The clean-up of the zinc boxes is effected by treating the zinc precipitate with sulphuric acid in a large lead-lined cast-iron pot. A hood is provided to carry away hydrogen and noxious gases. After the whole of the zinc is dissolved, the sludge is put through a small filter press having plates of a lead-antimony alloy to resist the attack of acids. The cakes are washed within the press, and after drying are ready for smelting. This is done in a Faber du Faur zinc distilling tilting furnace, which offers great advantages in smelting large quantities of bullion. These furnaces are cubical in shape, and are supported on trunnions, after the manner of the Bessemer converter. Figs. 1 to 5 show the construction of the furnace. The exterior consists of iron plate, covering a sheet of fireclay or brick-work. The opening (o) on the top serves for the introduction of coke or charcoal; the opening (h) is connected with the flue. The crucible, somewhat resembling a skittle-pot, lies at an angle with the furnace, and is supported on a narrow arch of brickwork (f). The mouth of the pear-shaped pot, which is in reality a zinc distillation retort, projects slightly beyond the furnace. The fire-bars (v) consist of wrought-iron bars set on edge. The furnace can be tilted by a worm gearing into a toothed wheel. The advantage of such a furnace for the smelting of large charges will be obvious to all who have to labor with even the best-designed ordinary smelting plants. The size of those on the Boulder is much too large for most Australian mines, but of course they may be constructed to hold from 1000oz. to half a ton.

The pots themselves are made of plumbago, and the charging of them through the open mouth is a very simple matter compared with that of feeding crucibles, even in smelting furnaces sunk below floor level. The flux generally used for smelting cyanide slimes after these have undergone acid treatment is borax, and the melted bullion runs from 98 to 99 per cent. of gold and silver. The purity of the bullion produced shows that the reproach attached to the zinc method of precipitation should be shifted on to the shoulders of the complainants. It is proposed to instal a plant for refining the bullion by Miller's process, but as I have repeatedly pointed out, this method is slow for low-grade bullion, and needs careful attention. A much simpler plan would be to take the gold sludge, purified from most of the zinc and other baser metals by the sulphuric acid treatment, and dissolve the gold out of this with chlorine in the wet way. The gold solution could be decanted and the gold precipitated in an almost pure state. The residual sludge of chloride of silver could be washed and reduced in the ordinary way.

The method adopted by Mr. McIntyre at Charters Towers is also a simple and economical one; this is described in detail in my article on methods of ore treatment in Queensland. Briefly it consists in melting the bullion as obtained from the zinc boxes so as to form a zinc-silver-gold alloy, the proportion of gold being kept at about 45 to 47 per cent., or a value, say, of £2 per oz. This alloy is granulated and the silver, lead, zinc, and copper dissolved out with nitric acid. The porous gold sponge left is washed, dried and smelted, when it assays 99.6 per cent. to 99.9 per cent. gold. The silver is precipitated with salt, washed, and the chloride reduced with scrap zinc left from the turning lathes. Many thousands of ounces were purified by this method, and the saving effected amounted to a very large sum annually.

An even more attractive method would be to take the precipitate after treatment with sulphuric acid, to wash it thoroughly free from chlorine compounds, and then treat it with dilute nitric acid; the whole of the lead may be removed by this method, but whether the whole of the silver would be in all cases, I cannot at present say.



Gold Smelting Furnace used at Kalgoorlie.

In the precipitation of bullion from cyanide solutions, it would appear as if silver were precipitated at the same time as the gold, the two forming an alloy on the surface of the zinc. I have before shown that from the result of many clean-ups with new plants the first bullion produced, or that precipitated in the first boxes, was richer in gold than that obtained subsequently. The gold is precipitated in the upper boxes, and the silver in the lower ones. After the first clean up, when the zinc is

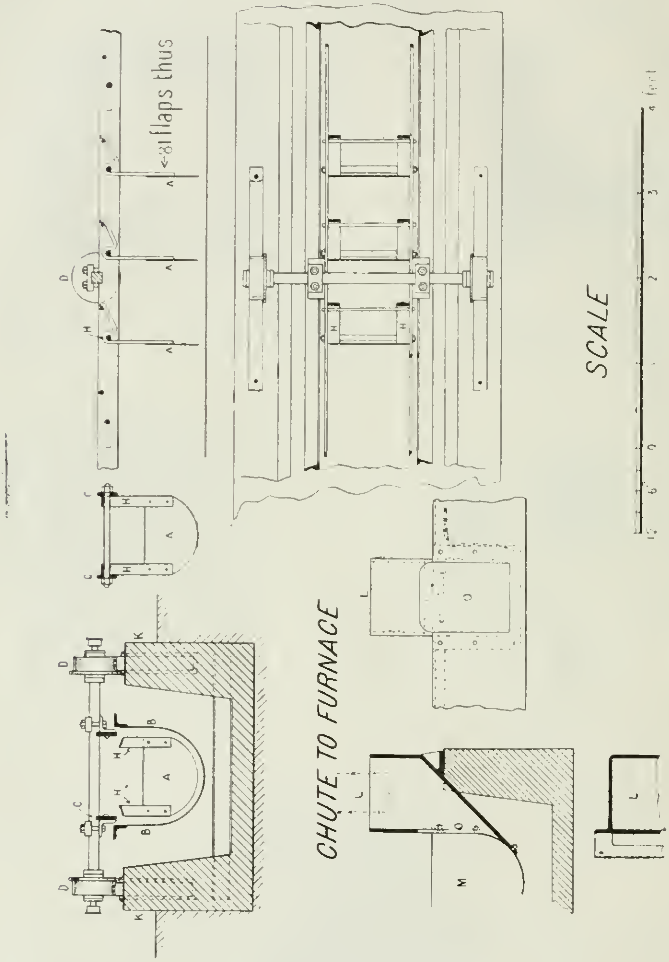
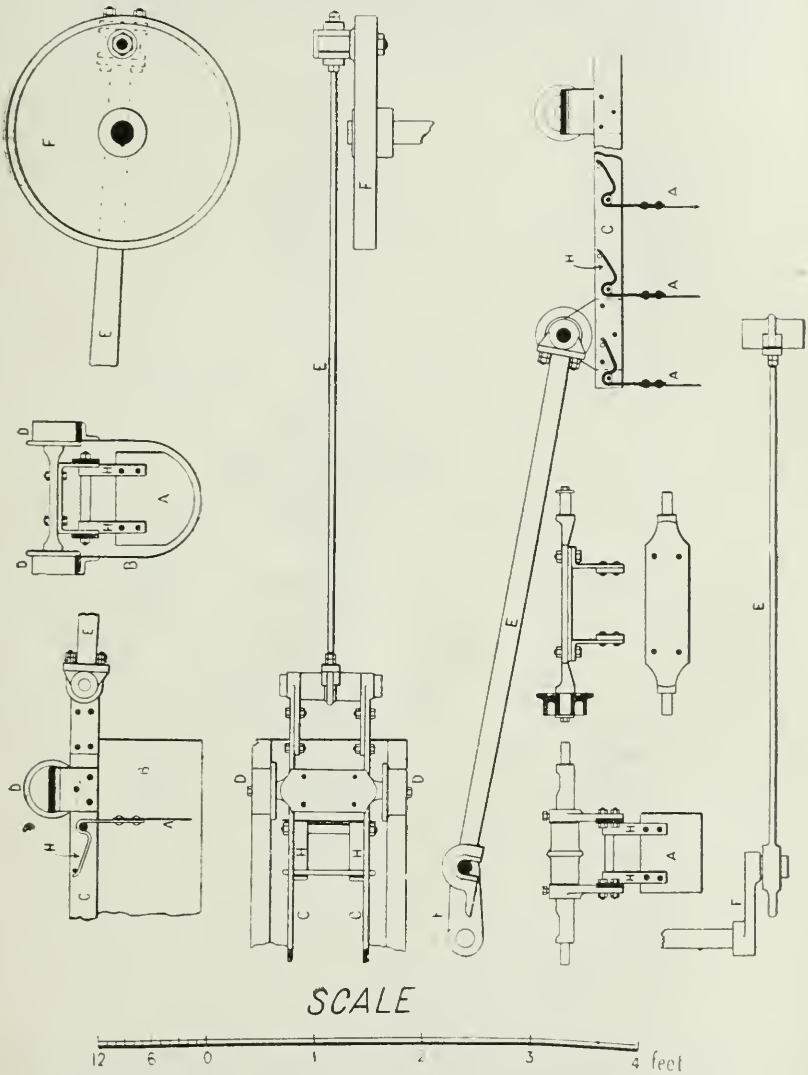


Fig. 1.—Push Conveyor.

moved upwards, more silver appears in the bullion until it subsequently reaches the same ratio as it is in solution. With charcoal precipitation this is even more marked. If the gold and silver are deposited as an alloy rich in gold, it would not be possible to part the gold-silver slime with nitric acid. The crudeness of much of the cyanide bullion produced, and the losses entailed in producing it and in subsequent refining charges, should disappear in a land producing so much gold.

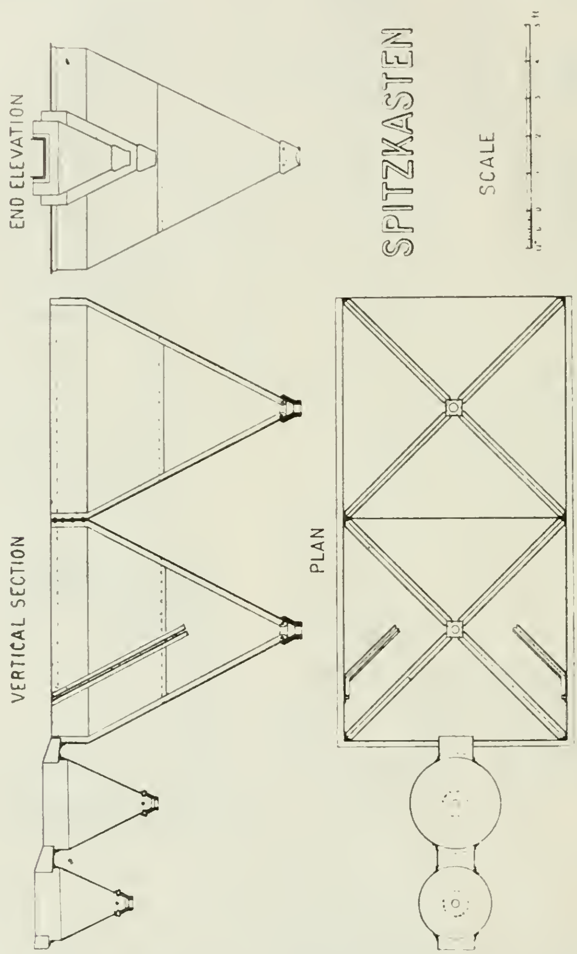
In addition to the plants already described, the Great Boulder Company has a customs works on the Great Boulder No. 1 lease. This is under the charge of Mr. S. Goldstein, who has the responsibility of dealing with a large amount of stone and sand per annum. The system adopted is crushing in a 10-head battery automatically fed, inside and outside amalgamated copper plates being used. The sand is passed over Wilfley tables, and the heavier con-



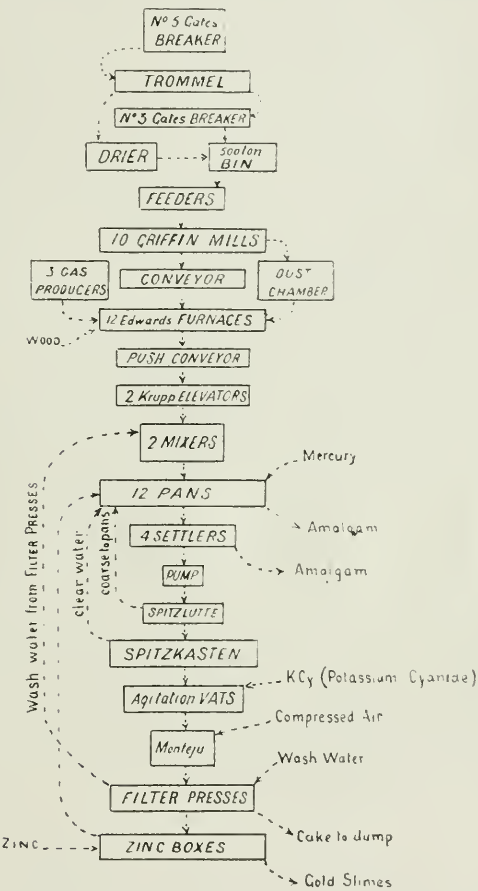
Figs. 2 and 3.—Push Conveyors.

centrates extracted. The sands escaping are ground in pans and then cyanided in double treatment vats, one being placed directly over the other. A Riecken plant for the treatment of a large quantity of low-grade slimes was erected. This will be described in a future article. An interesting feature of Mr. Goldstein's plant is that the cyanide solutions are pumped to and from the Great Boulder plant, about half a mile away.

The shifting of a large amount of hot material along level surfaces has led to the adoption of push conveyors in preference to those of other type. Spiral conveyors were found to distort in hot sand, but the push conveyors are simple to construct, easy to



maintain, and will work on wet or dry sand, and will move hot sand along without dusting. The conveyors figured were designed specially for use at the Great Boulder. Figs. 1, 2 and 3 show the construction of the conveyors, No. 1 being over 80 feet in length. They all consist of a channel (B) of thin steel plate, either of trough-shaped or square section. A series of suspended flaps (A) or plates of $\frac{1}{8}$-inch steel are arranged at equal distances, and fit within about one inch of the inner cross-section of the channels. The straps (H) suspending the flaps are riveted on to the outside edges, and are about $1\frac{1}{2}$ inches by $\frac{3}{8}$-inch thick. These rise vertically from the flap, and are turned over a horizontal rod, thence backward and slightly upwards for a few inches, the end being unattached, but resting against another horizontal rod. The rods are made of gas pipe, which act as spacing pieces between two horizontal longitudinal bearers (C), made of 3-inch by $\frac{1}{2}$-inch bar iron. An iron rod runs through the gas pipe, and is tightened up by screwing on nuts on the outside of the bearers. The bearers themselves are



Diagramatic Plan of Great Boulder Works.

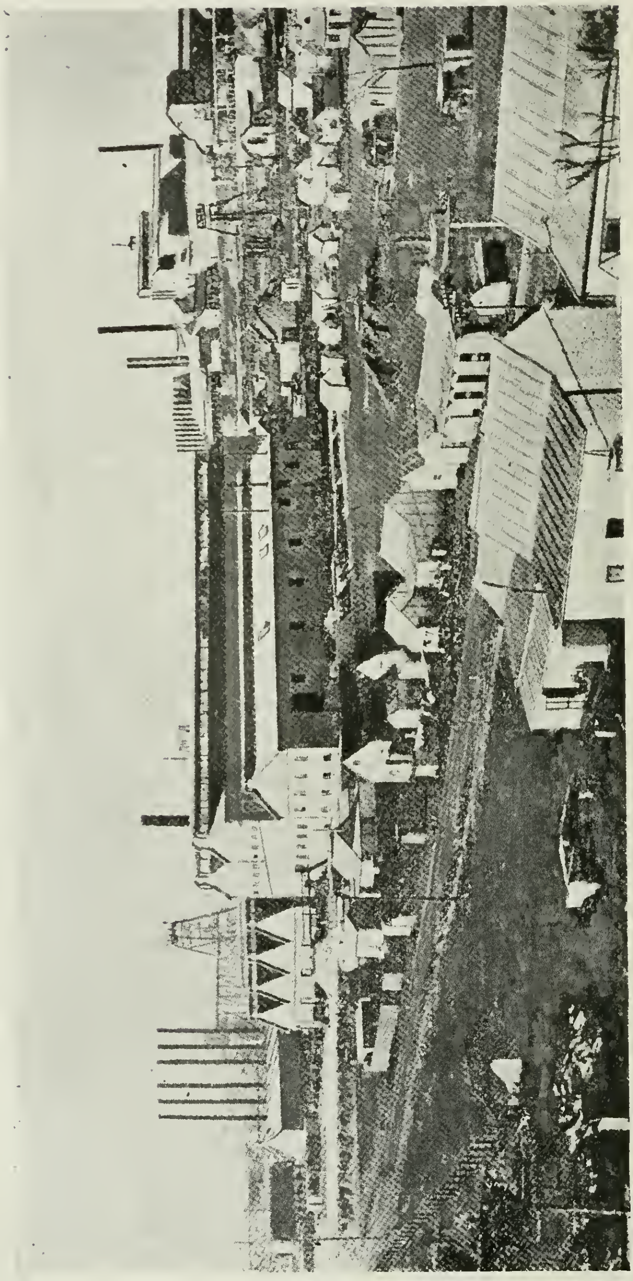
attached at intervals to an axle which spans the trough, and on the ends of which are two small flanged wheels (D), which run on a short track on a concrete foundation (K). A bar (E), acting as a connecting rod is attached to the horizontal bearers at one end, and to a crank (F) at the other. On the revolution of the crank the bearers run forward; the sand in the trough causes the lower end of the flap to be lightly dragged over it. On the backward revolution the flap assumes a vertical position, and a certain quantity of sand is pushed along by each flap for the length of the stroke.

Perhaps one of the most instructive innovations on the field is the spitzkasten used for thickening slimes and clarifying solutions. The old rules with regard to their dimensions are ignored, and though there is great diversity as to the number employed for a given quantity of slime, yet at all the mines they appear to work well. Practically one size is adopted, and for increased capacity the number is added to. The spitzkasten is, as the name implies, a pointed box, the dimensions being 6 feet square, and 6 feet 9 inches deep, the slope starting 1 foot from the top. The inverted pyramid is really 6 x 6 x 6 feet. These are constructed of $\frac{1}{8}$-inch mild steel, riveted on to angle iron, 2 x 2 x $\frac{1}{4}$-inch at the edges; $\frac{3}{8}$-inch rivets, spaced $1\frac{1}{2}$-inch centres at the sloping edges, and 3-inch centres on the upper edge, are used. The plates forming the sides, if more than one, are riveted with $\frac{3}{8}$-inch rivets, $1\frac{1}{2}$-inch pitch, the lower plate to overlap, so that no slime will collect on the ledge. The bottom of each box terminates in a cast iron cap, fastened on with $\frac{1}{2}$-inch cup head bolts, and packed watertight. A $2\frac{1}{2}$-inch gas tap opening is provided; a pipe with valve attached screws into this, and so allows of regulation of downward flow. The boxes are arranged in series, connected into a square or rectangle. Each series has one or two smaller conical boxes preceding it. These are about 2 feet and 2 feet 6 inches in diameter, and 2 feet $8\frac{1}{4}$ inches and 3 feet $2\frac{1}{4}$ inches deep respectively. A cast iron cap, with screw for a 2-inch gas pipe, is provided below. The distances between the centres of the boxes are 2 feet 8 inches, 4 feet 8 inches, and 6 feet respectively. The slimy water flows from the sand separators, from the bottom of which any sand or heavy particles of amalgam are led back to the pans. The lighter material from one spitzkasten to another is settled and drawn off from the bottom of each at about a 1 to 1 consistency, while the clearer and clearer water overflows from each until it finally passes into a launder almost clear. How strange it is that the terrible bug-bear of the old-time amalgamators and concentrators should be looked upon as a blessing by the modern metallurgists. Avoid making slimes was the old, and slime everything the newest, doctrine. But since slime production and treatment is much more expensive than the older methods, the latter is much to be preferred when there is no advantage in fine grinding. The general scheme of ore treatment and gold extraction employed at the Great Boulder may be followed from the diagramatic plan shown.

The Great Boulder Perseverance.

The plant on this mine is one of the most perfect examples of engineering skill on the field. The buildings, though covering so much of the 24-acre lease as to leave no stacking room for tailings, are compact and proportioned to the machinery they have to surround, support and cover. The ore itself, from the time it leaves the mine until it receives its check at the filter presses, flows through the works in a steady stream, and after leaving the presses is carried mechanically to the tailings dump. The stone as broken below comes to the surface in large blocks admixed with the fines. This is dropped on to a grizzly, the fines passing through and the blocks being fed into a No. 5 Gates breaker; thence the coarse part drops into two No. 3 Gates breakers, the fines passing through a grizzly. The whole of the crushed product falls into a 1200 ton bin. An automatic feed from this bin is given by a Challenge feeder, which delivers a regular stream of rock on to a Robins belt conveyor. These conveyors are used wherever possible throughout the building and have done such excellent work that a short description will not be out of place. At the Perseverance one of the conveyors has transported over 100,000 tons of rough angular stone without showing any appreciable wear. The conveyor consists of a belt made of canvas, and covered with rubber. The edges of the belt are turned up, so that the material carried lies in a trough while the belt is moving. The belt is driven by passing over pulleys, and is guided and supported by a number of fixed pulleys. The only part of the surface subjected to wear is the point where the ore is delivered. After the surface of the belt is reached it is carried noiselessly and without any motion on the belt itself to its point of discharge. The guide pulleys are known as idlers, those above being called the troughing idlers. The two topmost ones turned inwards are only used occasionally to prevent the belt running off at the edges and thereby inducing chafing. This can only happen when the belt runs sideways from the setting of the framework, or the end pulleys not being parallel; similarly the vertical bottom idlers.

The belt is of an even thickness throughout, but as the centre wears more rapidly than the sides an additional thickness of rubber has been supplied there. This also allows of greater flexibility at the sides. When it is desired to discharge material at any point along the belt this is provided for by a tripper, which is simply an arrangement consisting of a pair of pulleys which turn the upper conveying belt into curves resembling the letter S. The ore falls over the upper curve and drops into a chute, which delivers it sideways clear of the belt. The tripper may be moved along to any part of the belt. The conveyors may be used as sorting belts, conveyors on horizontal or inclined planes; they may be used for conveying grain, ore, sand, and are now in use for elevating tailings from bucket and other dredges. It is needless to say that they will not convey material beyond the limiting angle of friction or the angle of slope for various materials. Coal has been conveyed up to an angle of 26 degrees. The drawing shows the tail end of the belt, and pro-



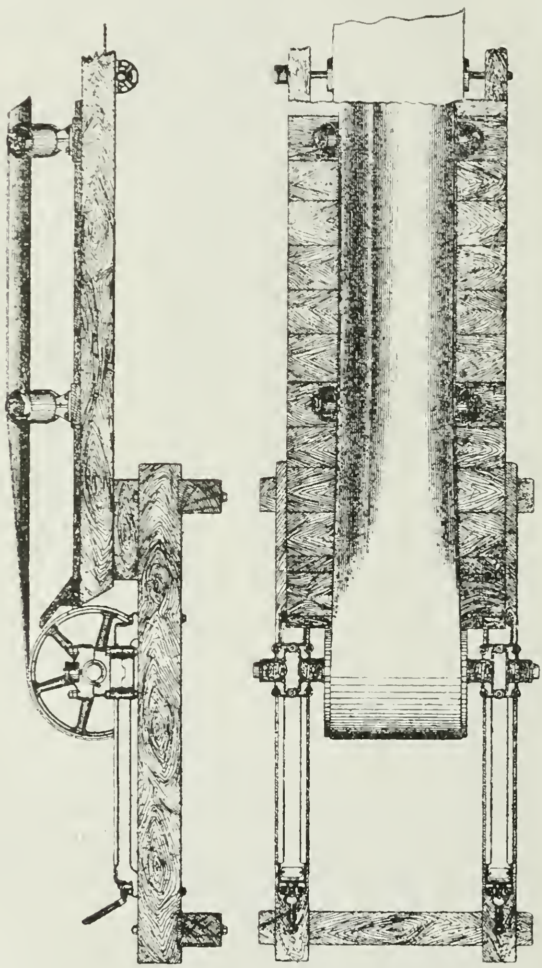
Great Boulder Perseverance.

vision made, i.e., a wooden platform, to prevent ore tumbling over to the lower belt. The sole agents for this simple and useful conveyor are Fraser and Chalmers.

The ore after leaving the conveyor drops into a hopper, which supplies the Griffin mills, of which there are 10. These mills were not fully employed at the date of my visit, seven of them putting through 300 tons per day. Each mill takes about 25 horse-power to drive it, and crushes from 30 to 40 tons per day. Since the amount is much larger than that treated in other works, a comparison of the grade as regards size is interesting:—Passing through a 120-mesh screen, 72.6; remaining on a 120-mesh screen, 3.5; on 100, 2.3; on 80, 6.5; on 60, 7.1; on 40, 4.1; on 30, 3.2; on 20, .7. At the Boulder the capacity is about 26 tons per day, 75 to 85 per cent. going through 120-mesh screens, and at the South Kalgurli 20 tons. It is evident, assuming these figures to be correct over a long period, and there seems to be no doubt about this, that power amounting from 50 to 100 per cent. is required to perform less than 25 per cent. more work. These figures show the advisability of removing the fines and grinding the coarse sand in a pan.

A Sturtevant fan is placed below the Griffin mill, which serves to withdraw the dust. This is then settled by cyclone arresters, and dropped down cones into the furnace. A large chamber for settling fine dust is provided. The crushed product from the Griffins is carried away below by means of a spiral conveyor. It is then elevated by a bucket elevator, and delivered on to a Robins belt conveyor, which carries it to hoppers of 6 Holthoff-Wethey furnaces. Each furnace is 120 feet long and 12 feet wide, and will roast 60 tons (averaging 50) in 24 hours. The type is a favorite American one. The rabbles consist of a number of miniature mould-boards carried on a carriage which spans the hearth. As the carriage is dragged along, these blades cut into the ore, and turn it over to a slight extent, and also move it along slightly. Another carriage is just behind, whose rabbles bisect all the cuts made by the former one, and so on for a series of eight of them. A slot runs along the side of the arch, and the ends of the carriage project through this and are attached to endless cables, after the manner of the Brown furnaces. The ore is discharged underneath the furnace, which becomes the cooling floor. The rabbles, after passing through the furnace, pass round and through the ore on the floor below. A number of suspended small iron doors fall when the rabbles pass in or out, thus preventing a large excess of cold air. Similarly, as the carriages pass along the suspended iron covers are raised, and drop into their places as soon as it passes. The fire places are arranged along the side.

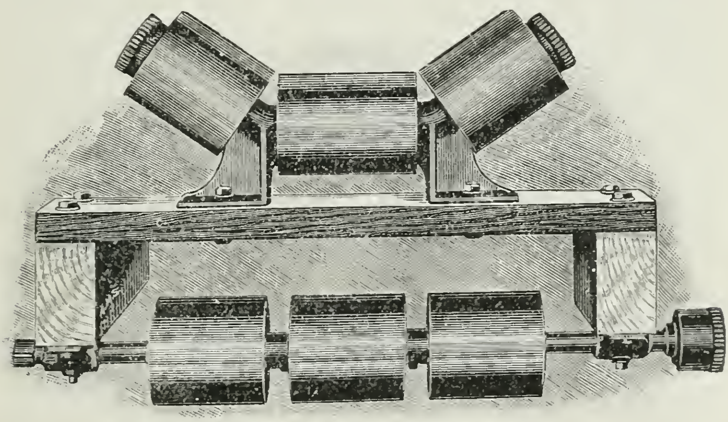
The cooled ore is sent along by a spiral conveyor to an elevator, which lifts it on to a Robins conveyor, which delivers it into a mixer. This is a closed in vessel 5 feet in diameter, with propeller-shaped blades. Dilute cyanide solutions are used to moisten the ore, the amount used being from 50 to 60 per cent. by weight. The slimy material from the mixer is led into amalgamating pans, of which there are 11. Smaller pans of the Wheeler type were first used; these are being replaced by larger pans of the manager's design. The smaller pans are of the Wheeler type, 4 feet in diameter, and run at a speed of 50 revolutions. The larger ones are 8 feet in diameter, and are run at half that speed. In the larger pans the shoes and dies are flat, not radially ribbed, as in the Fulton and Meekison pan, nor circumferentially, as in the Watson and Denny. Provision is made for a large quantity of mercury in the rim, which surrounds the dies. A vertical baffle plate in front of a sloping overflow discharge prevents mercury from being swilled out by centrifugal action. The sides of the larger pans are constructed of mild steel or wrought iron, the bottom being cast. A large proportion of gold is



Robins Belt Conveyor.—Tail End, arranged to prevent material from dropping on lower belt.

caught in these pans, the dilute cyanide solution assisting in promoting amalgamation. As much as 75 per cent. of the total gold won is got from the pans. The loss of mercury is very small. The amalgam obtained from the pans is cleaned up in an amalgamating barrel and retorted in the usual way.

The slimes overflowing from the pans are led into agitating vats, of which there are 24. Each vat is 20 feet in diameter, and 4 feet 6 inches deep; about 16 or 17 tons of slime are run in, the water being present in the proportion of $2\frac{1}{2}$ to 3 by weight to 1 of dried slimes. The vats used on the field for agitating are very large, and since the time of agitation is about 16 hours, their capacity must be two-thirds of the daily output. As a rule, the arms may be raised or lowered on a central spindle, to which they are keyed, but the arms have to be so light and long that there is a great tendency to break or bend them. It cannot be said that they agitate the material to any great extent, for they travel round with almost the same velocity as the slimes; nor can it be said that the material is well mixed by them, for the heavier sand is continually dropping, and ultimately forms a cake on the bottom, which has to be dug out. A simpler, more easily worked, and better type of agitator should be possible. The solutions are made up so as to contain from .03 per cent. to .05 per cent. of cyanide; these are then agitated for about 17 hours. The consumption of cyanide is about 1lb. per ton of ore. Provision is made for raising or lowering the agitators, also for supplying air through hollow arms. The latter device was for the



Robins Belt Conveyor.—Arrangement of Idlers.

purpose of oxidising any soluble reducing compounds, such as sulphides, from imperfectly roasted ores. The addition of chloride of lime was also tried for the same purpose, but it is doubtful if any subsequent treatment will compensate for a bad roast. A certain amount of heavy sand containing mercury and amalgam settles in the bottom of the agitators; this is dug out every few weeks and returned to the pans. The pulp from the agitators is run into one of four pressure tanks, whence it is forced into filter presses. There are 10 presses provided for the purpose, each press holding 50 three-inch cakes, or $4\frac{1}{2}$ tons in all. The solutions running from the presses are clarified by passing them through a large Dehne press, and are then sent through one of seven zinc boxes. The solutions after the gold has been extracted are sent back to the mixers or pans. The residues or cakes discharged from the press are dropped down a hopper and are carried by a Robins conveyor to the tailings dump.

The zinc boxes are cleaned up in the usual way: the slimes being roasted, treated with acid, filter pressed, fluxed and smelted in ordinary pots. The magnitude of the last operation will be grasped by the few who know what 15,000oz. per month takes to clean up and smelt.

The bullion produced by amalgamation is worth £4 per oz., and that from the cyanide works £3 to £3 10s. per oz. [The last report issued states that there were 250,000 tons, averaging 1½oz. per ton, ready for raising. This would amount to about a two-years' supply, the plant erected treating 300 tons per day without difficulty. The cost of mining oxidised ore was stated to be 7s. 10d. per ton, and of sulphide ore 13s. 11d. per ton. The treatment costs amounted to 38s. 9d. in 1900 to 28s. 5d. in 1901, and at present less than 21s. per ton.] The manager, Mr. Ralph Nicholls, C.E., estimated the cost of treatment at 25s. per ton, so that actual expenditure proved to be well within the mark.

The motive power for supplying the plant consists of six Heine water-tube boilers, 200 maximum horse-power, the working pressure being 125lb. per square inch. Originally there was a prejudice against water-tube boilers, even when condensed water was used. It was found that the trouble with them was occasioned by the oil passing in with the condensed water. This is now wholly overcome by the use of oil filters, such as Edmiston's. The mill engine is of the Corliss type, 20 x 36 x 48 inches, supplied with an independent jet condenser, developing 375 horse-power. It is claimed that not more than 60 per cent. of the quantity of fresh water that is being used on other mines for the same horse-power is required. Motors are freely distributed throughout the building, and each one is supplied with a centrifugal fan giving a blast of cold air. The motors have occasioned no trouble, and have served admirably to drive various appliances which would have been awkward to reach with other forms of transmitted power. Another plant, known as the Lakeside mill, was made use of for dealing with oxidised ores. This consisted of a 20-head battery, with copper plates and canvas tables. The tailings from this passed into spitzkasten, the sand flowing into large rectangular vats, of which there were four, capable of holding 100 tons each. The slimes were agitated and treated by four Dehne filter presses of the same size as those used in the sulphide mill. The concentrates were sold.

The plan of operations at the sulphide mill is perhaps the simplest of the mechanical and chemical processes on the field. The plant, buildings and machinery cost £125,000, and the whole of the work, which would take a volume to describe in detail, goes on so smoothly that it reflects the highest credit on Mr. Nicholls, the manager, and Mr. Blakeslee, the engineer. The simple system adopted, starting with dry crushing, then amalgamating and finishing with cyaniding and filter pressing of the whole of the material, is carried out as cheaply as processes which depend on differentiation of sands and solutions, so that if extractions are as good nothing is to be gained by complicating the process. It is evident, even to a casual observer, that a definite scheme was designed and rigorously carried out, and that the lowering of costs is due in a great measure to the engineering skill displayed in the construction of the whole works.

The Golden Horseshoe Estates Company, Limited.

The small lease owned by this company contains some of the richest stone on the field. Mr. J. W. Sutherland is the manager, and through his courtesy I was enabled to see the whole of his well-equipped plant. The method adopted is wet crushing, concentration, grinding of the sands, agitation with cyanide of the slimes and ground sand, and filter pressing. The rich ore and concentrates are smelted in a blast furnace. The ore is tipped out on a grizzly, with bars spaced two inches apart. The fines pass into a bin, while the coarse goes through two No. 3 Gates' breakers, thence into the same bin, which has a capacity of about 200 tons. The ore is then trucked to the battery, being fed in through Challenge feeders. The mill contained 50 head of stamps, 30 by Fraser and Chalmers, and 20 by Martin. An additional 50 stamps, each weighing 1250lb., was in course of erection.

Inside amalgamation is practised, the pulp passing from the mortar-box through a woven screen having 800 holes per square inch. Amalgamated copper plates are used outside. The concentrates are eliminated by Wilfley tables, or on tables somewhat similar, made by Martin and Co., Gawler, which are said to be superior to the Wilfleys. The coarse sand is ground in amalgamating pans of the Wheeler type, the fine material eliminated running through the usual settlers, one being used for two pans. The fine sands are separated from the slimes by means of spitzkasten, and care is taken to eliminate the concentrates from all classes of fine material by running them over canvas strakes having a fall of ⅛-inch per foot. The sands are sent to vats through a Butters' distributor. These vats are 20 feet in diameter, and 7 feet deep, and hold about 70 tons. The size of the sand may be inferred from the fact that about one-quarter will remain on a 50-mesh sieve, one-third extra on a 100-mesh, the balance passing through the 100-mesh sieve. This is treated in the upper vats, then discharged into corresponding vats below, the treatment taking about 10 days. The slimes were treated in filter presses direct, without agitators, there being nine presses, having 50 frames, with 3-inch cake. Three of the presses are filling while three others are under way, and the last three being discharged. The gold is dissolved out of the cake by forcing solutions through it. The solution of the gold is stated to be almost complete in 20 minutes: since this is only 1-60th of the time required in the agitating vats, it is a matter which deserves careful investigation. The speedy solution is generally attributed to pressure alone, but the extra quantity forced through, the more perfect contact, and the larger quantity of oxygen dissolved are more probable causes. After the gold has been dissolved the cakes are water-washed by forcing water through them, after which they are discharged.

The number of stamps in the new sulphide plant is 50. A most interesting series of particulars was courteously supplied by the general manager concerning the output of this plant. Taking a month's run, the ore milled was 6240 tons, or per 24 hours 201.29 tons; the actual time being only 27.54 days, gives an actual output per 24 hours working time of 226.57 tons, so that the actual duty per stamp is 4.53 tons per 24 hours. Each stamp delivers 96 blows per minute, with a 8½-inch drop. The average height of the discharge is 4½ inches. The screens at present in use contain 200 punched holes per square inch, equal to about 700, with woven screens. The average life of the woven wire screens is 4.5 days, of the punched screens 10 days; the dies last 300 days, and the shoes 225. The pulp from the battery averages from 48 to 50 per cent., and will pass through a 200-mesh sieve. The whole of the sand, with the exception of the concentrates, is ground in grit mills, so as to pass through a 200-mesh sieve, or one having 40,000 holes per square inch. The grit mills, described in a later article, are used in the new sulphide mill instead of pans for grinding, put through 28.7 tons in 24 hours, the quantity of flints in each mill being 5 tons, and the consumption of flints in each mill being 1 ton per month, the average life of a set of steel liners in the mill being 240 days.

In the filter press department, the number of press charges during the month was 678; the number of presses treated was double this, or 1356, and the number of tons put through 6102, or 203.4 per 24 hours, the number of presses treated per shift being 15.06.

The number of agitating vats filled was 151, each therefore holding about 40 tons. The cyanide used was 15,904lb., or 105.32lb. per vat, or 2.60lb. per ton. The lime amounted to 14,170lb., or 2.32lb. per ton. The consumption of zinc amounted to 448lb., or only 0.7lb. per ton. The number of filter cloths used was 330, or at the rate of .05 cloths per ton of ore. The salt water absorbed in the residues was 256,284 gallons.

The gist of the treatment is elimination of free gold and concentrates as far as possible, cyaniding the ground sands and filter pressing the slimes. The ore treated by these means gives of the gold recovered about 60 per cent. by amalgamation, 20 per cent. from the sands, and 20 per cent. from the slimes, the total recovery being about 90 per cent. The concentrates and ore richer in sulphides than that being treated are entrained to Fremantle to be smelted. The rich sulphides obtained from the mine are smelted in a water jacketed blast furnace on the works. This has a capacity of about 30 tons per day, from 8 to 10 tons of ore passing through. There is no roasting furnace attached, so that the sulphur is mainly eliminated by feeding scrap iron into it. The limestone for fluxing comes from Southern Cross, the ironstone is procured locally, while the lead ore comes from Broken Hill; metallic lead, matte, and slag are produced. The lead is introduced into a cupel of the English type, and reduced to a size and raised to a value suitable for shipment. The litharge is used again in the furnace, and, when reduced, collects a large proportion of the gold again, the difference being made up with raw galena concentrates.

The cyanide solutions from the vats and filter presses are clarified, and then sent through zinc extractor boxes of the usual type; the zinc sludge is treated with sulphuric acid, then filter pressed and washed, and the cakes are smelted in a Faber du Faur tilting furnace.

For refining the bullion produced a Miller chlorination furnace has been built. This method of parting has been in use at the

Australian mints since 1867. The bullion mainly received, in those days consisted of alluvial gold, or bars derived from melting very pure bullion. The composition of this was gold and silver, with base metals, such as iron and copper, present in quantities, as a rule, not exceeding 1.5 parts per 1000. The silver, as a rule, did not amount to more than 10 per cent. The method, as originally introduced by Miller, has only been modified to a slight extent. About 600 or 700 ounces of gold bullion are placed in a French clay pot, in which there are from 2 to 3 ounces of fused borax. The furnace, a modified assay furnace, is filled with fuel and the gold melted. A perforated lid is placed on the crucible, and a pipe stem, or a cylindrical earthenware pipe, connected at one end with chlorine under pressure, is made red hot and dipped through the hole in the cover of the pot into the molten gold, the chlorine being turned on to prevent the pipe from being choked up with gold. The chlorine attacks the base metals first, and the volatile chlorides escape in dense fumes through the cover. Afterwards the silver is attacked, and the silver chloride formed floats on top of the molten gold. Near the end of the operation the stream of chlorine is lessened to prevent the gas spirting and carrying away gold and silver chloride. As soon as a brown vapor escapes, which will give a brown stain on white earthenware, the operation is finished. The silver chloride is ladled out periodically, the last dippings always containing some gold dipped up; these are kept separate. The gold produced has a fineness of from 996 to 998. The cost at Melbourne averages about a farthing per ounce, but the gold is fairly pure, running considerably over 93 per cent. In Sydney, for bullion about 85 per cent. gold, 13 silver, and 2 to 3 per cent. of base metals the cost is about one penny per ounce, and the loss in gold from 0.11 to 0.19 per 1000. The time taken for gold containing only 2 per cent. of silver and 0.5 per cent. of base metal is an hour and a half, while if the silver is 3.5 and the base metal 1.5 per cent. the time taken is two hours.

This process is eminently adapted for the elimination of small quantities of silver from gold, yet if the silver rises to about 30 per cent. and the gold is contaminated by base metals, it can scarcely compete with other methods. It needs also to be stated that the silver chloride, which floats on top of the gold, carries from 5 to 10 per cent. of gold. This is recovered by fusing the chloride and adding bi-carbonate of sodium fractionally. Part of the chloride is reduced to silver, and the shower of silver buttons carries the gold down. In some instances granulated zinc is added for the first fractional precipitation. Silver is afterwards recovered by hanging the chloride in flannel bags in an iron bath. Plates of iron are placed between the bags, a little ferric chloride is added, and steam is passed in. The silver is reduced to metal, and is washed with boiling water and smelted without fluxes. It was found at the Melbourne Mint that when pure gold was alloyed with 4½ per cent. of copper the cost of extracting the copper was 4.5d. per ounce Troy. For the same percentage of lead, it was 1.4d., for 4 per cent. of iron 3.9d., and for 4½ per cent. of tin 2.5d. per ounce Troy. The melting of cyanide slimes to a base bullion and the subsequent refining by such a process as Miller's does not seem to be a rational proceeding when, as already pointed out, the slimes are capable as such ready attack by gold or silver solvents, as the case may be.

The Great Boulder Main Reef.

The Great Boulder Main Reef lies near one extremity of the Golden Mile. The method of treatment adopted does not differ in any material respect from that on some of the mines already described. The ore is broken in a Gates' crusher No. 3, and is reduced to powder in ball mills. The product from the mills passes either into three Edwards' furnaces, each roasting 15 tons per day, or one Richard furnace, roasting about 35 tons per day. The latter furnace is after the design of that of Captain Richard, the metallurgical engineer at Mount Morgan. The furnace will be described in an article dealing with Mount Morgan. It consists of a shaft rectangular in section, 30 feet long, 12 feet wide, and 65 feet in height. It is divided horizontally by 11 arches, each of which constitutes a floor. The ore is fed in at the top by means of a screw conveyor, and slides down from arch to arch through openings either in the side or top until it reaches the discharging floor which is a hand reverberatory furnace. At Mount Morgan the motion of the ore could be assisted by jets of compressed air when necessary. The floors were so arranged that the heated gases from the fireplace passed over the floors from end to end until they escaped by the outlet on top. The principle was the same as in the Stetefeldt furnace, only in this case the ore was exposed much longer to the oxidising gases. It also has a strong likeness to the Hasenlever and Helbig furnace, which has sloping shelves alternately placed instead of arches. The physical properties of the crushed ore at the Boulder led to different results being obtained from those at Mount Morgan, and it would appear as if Edwards' furnaces were considered more suitable, since three of these have been installed. Curiously enough, the effect of the trunnion has been neutralised by the manager bricking them up underneath, evidently for the purpose of conserving some of the heat. The shaft furnace is said to consume 3.5 tons of firewood per day, and each Edwards' 2.25 tons.

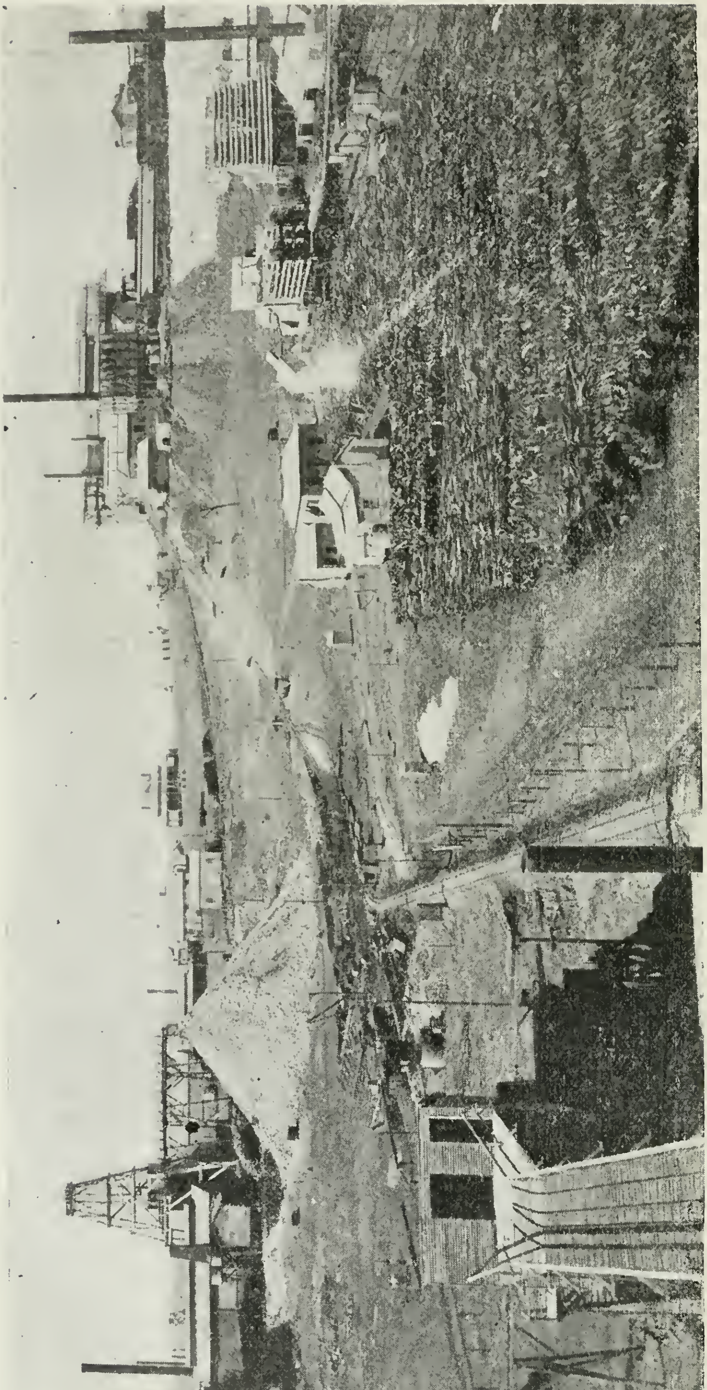
The roasted ore is moistened with dilute cyanide solutions. The slimes are separated and passed into agitators, the sands passing to Wheeler pans, where they are ground and amalgamated. The overflow from the pans is passed through a spitzkasten, any coarse material being sent back to the pans, while the fine slimes are agitated in vats. The material thus reduced to slime passes into one or two montejus and is forced into Dehne filter presses. The clear solutions pass into the zinc extractor boxes, while the washed cakes of slimes are sent to the dump. The costs of treatment were given as £1 6s. 10d. per ton, and the total costs of mining and milling as £2 5s. 8d. The average value of the ore, including tailings treated, appears to be about £3 per ton. Mr. J. T. Marriner was manager at the time.

The Kalgurli Gold Mines, Limited.

In order to obtain an idea of the difficulty of prospecting a mine on the Boulder, a visit to the Kalgurli mine is of importance. At the date of my visit the main shaft had been sunk to a depth of 1078 feet, while crosscutting for the lode had been started at 1050 feet and 1000 feet levels. At 920 feet the lode was proved to be 30 feet in width, having a value of 25dwt. per ton. The ore had been driven on for a distance of 45 feet north and south of the crosscut, the width of the drive being 8 feet. For this distance the average value was 34dwt. per ton. At 850 feet only small lodes were met with, the ore taken from the shaft at this level running about 8dwt. per ton, and gradually lowering in value from the shaft as crosscutting proceeded. From about 660 feet the lode matter was carried down with the shaft. At 700 feet the lode was cut by a western crosscut 36 feet from the shaft. It averaged 30dwt. for a width of 7 feet. At the 640 feet level a western crosscut had been driven for about the same distance as that in the lower level, and the lode matter here was rich enough to be taken out for a width of 50 feet, and has been proved for a length of 160 feet. At the 540 feet level there are two lodes—the eastern, which is from 10 to 35 feet in width, and the western, from 18 to 27 feet wide, with a proved length of 75 feet, averaging 25dwt. per ton. At this level both free gold and telluride occur, and it is noteworthy that these lodes are more siliceous than in other parts of the mine. From 540 feet downwards the lodes are opposite and south of the shaft: above the 420 feet the lodes are 50 feet north of the shaft. It would appear as if a large fault lode running north and underlying east, having a pitch to the south, had caused the lodes above it to shift northwards. Where the main lodes come in contact with the fault at 300 feet and above this level they are payable for a width of over 80 feet. From the 200 feet to the 100 feet level the lodes are 80 feet in width, and averaged 28dwt. per ton.

In going down the mine all that one sees is a bluish-grey rock, which in general appearance looks the same whether one is on the lode or off it. On closer inspection the payable rock carries pyrites evenly diffused through it, and at times veinlets and blotches of tellurides are apparent. At the 640 feet, 700 feet, 850 feet and 920-feet levels no free gold is visible. No doubt those thoroughly familiar with the rock and its characteristics are able to judge of its value to within certain limits, but even by the most experienced it is not deemed safe to rely on any estimate but those given by a multitude of assays. The possibilities of this mine are great when it is considered that between a couple of levels at least 50,000 tons of payable ore exists. Yet this is only looked upon as one of the smaller mines!

The mine is well equipped with a fine winding engine and an air compressor made by James Martin and Co., of Gawler (S.A.). The ore is tipped from the brace of the main shaft over grizzlys into two No. 3 Gates crushers, and crushed down to 2-inch gauge: it then falls into bins having a capacity of 100 tons. From these it is delivered into 11cwt. skips, weighed and conveyed to the ball



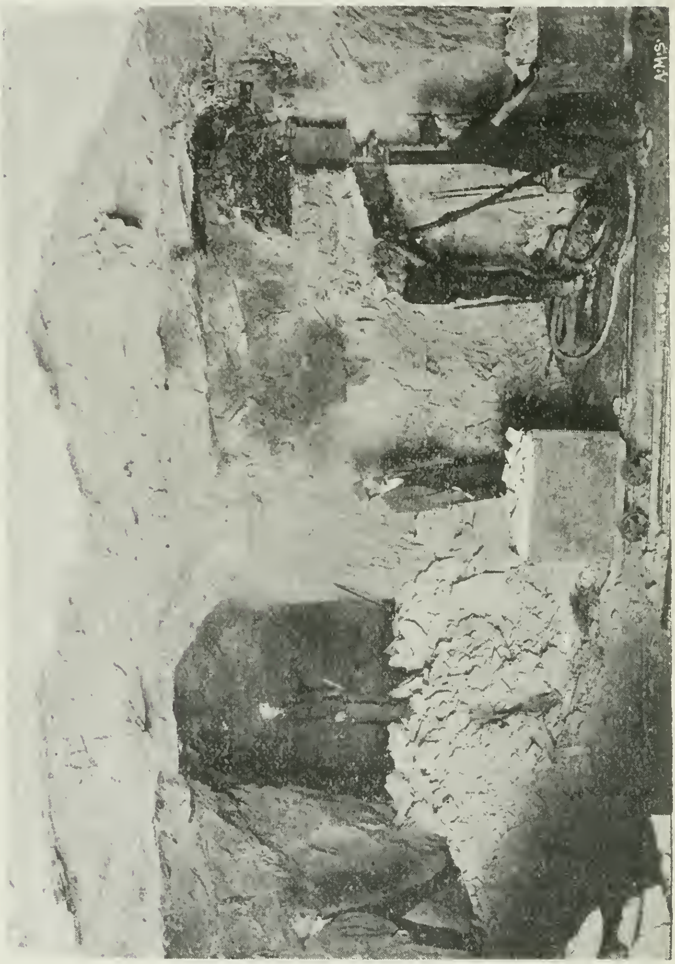
The Kalgurli Gold Mine

mill bins by means of an aerial tram over a distance of 574 feet, and 97 feet above the ground. The cost of conveying, including labor, repairs and renewals, was 2.78d. per ton, and the I.H.P. required was 4.25. The ball mill bins are placed directly over the mills, and have a capacity of 300 tons. Six Krupp mills, No. 5, are arranged in pairs, and are driven by belts. The experience of the field shows that belt driving is far preferable to spur wheel gearing. The ore is automatically fed into each mill and has to pass through a 40-mesh screen before it is discharged. Dusting is prevented by employing a large fan, which exhausts the very fine dust from the mills and drives it into a large dust chamber, which is a long room having hessian sides; the air is forced through the permeable hessian walls, while the dust falls into the chamber, from which it is removed periodically. This ingenious dry filter is very efficient, keeping the mill room free from dust.

The crushed material from the ball mill passes into shoots, from which a screw conveyor distributes it into the roaster bin, which is 185 feet long, having a capacity of 400 tons. Nine Edwards' furnaces are arranged with their longer axis at right angles to this bin, so that each is fed independently by means of a fluted roll worked with bell crank and ratchet, having adjustable pawls. The temperature of the roaster at the feed end is approximately 420deg. Fahr., and the discharge end 1200deg. Fahr. The roasted ore is discharged into a push conveyor 4 feet wide and 200 feet long, and is pushed along to a chain and bucket elevator and raised to a height of 21 feet; it is then discharged into a mixer, from which it issues in a thin pulp; this passes into the hydraulic classifiers, where the sand is separated from the slimes. The coarse and heavy particles escape through the bottom of the classifier, while the fine slimes flow over the top. These slimes are fine enough to pass through a 90-mesh screen. The overflowing material, carrying about one of slime to five of water, is run into five sets of slime settlers of the usual type, the clear water being again elevated to the head water tank by means of a 6-inch air lift. The thickened slime passes out through valves at the bottom of a consistency of one of slime to one of water; this flows in a continuous stream to the combined agitation and pressure tank 5 feet 6 inches in diameter and 13 feet 6 inches deep. The cyanide solution is here added to the pulp and agitated by compressed air for three hours; it is then forced up into the filter presses, of which there are two, with compressed air at a pressure of about 30lb. per square inch; it is washed with spent sump solutions, dried with compressed air and then sent to the residue dumps. One defect of the filter press as used for auriferous slimes lies in the construction of the cock as ordinarily supplied. On almost every mine one may see wooden plugs replacing broken or damaged taps. A special form was devised at the Kalgurli mine which is simple, effective, and not likely to go out of order. The accompanying drawing gives all details.

The coarser and heavier material flowing from the bottom of the classifiers is run over copper plates, thence on to Halley tables, where all the heavy and coarse particles are removed; the sand is then elevated into 100-ton cyanide vats, and treated for approximately 20 days. The solutions are kept constantly circulated and ærated by means of small air lifts introduced by Mr. Moss; these take the solution from the bottom of the vat, and deliver it again on top of the sand. The concentrates are ground to slime in a Wheeler pan, agitated in the pressure tanks and filter pressed in a similar manner to the ordinary slimes.

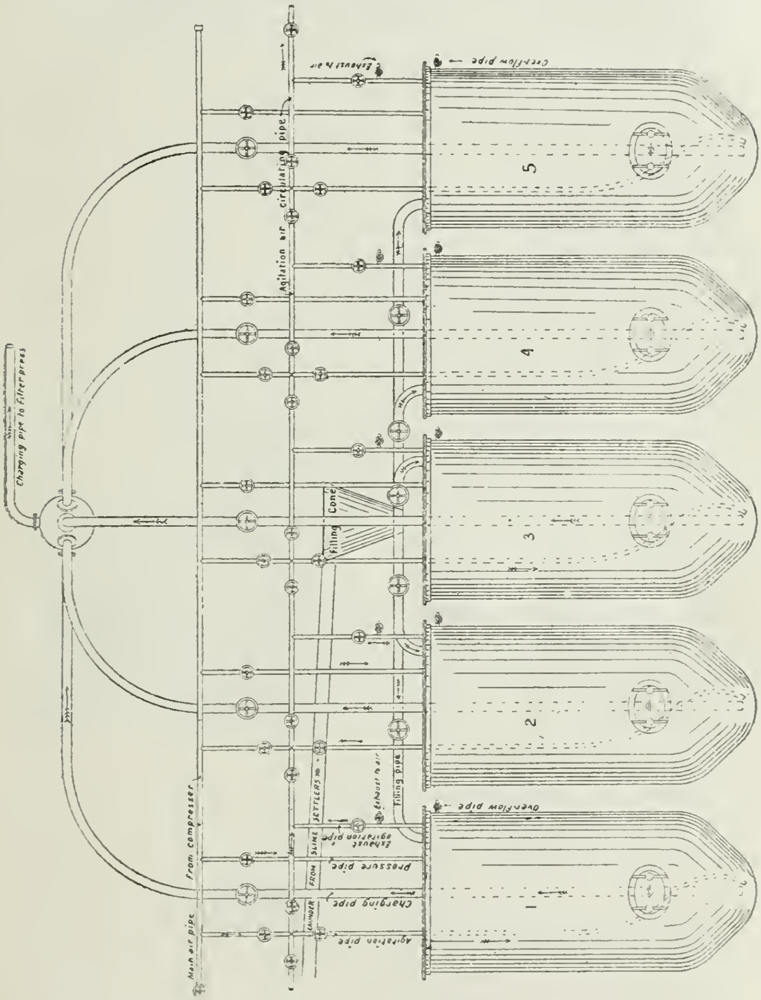
As showing the economical arrangement of the plant, one man on each shift attends to the ball mills, the cost for labor being only 5½d. per ton. Two men on each shift feed the ore into the furnaces, roast the ore and deliver it into the elevator at a cost for



Underground in the Kalgurli.

labor of only 8½d. per ton. The cost of fuel is 3s. per ton. One man attends to the classifiers, slime settlers, elevators; one man takes the slime from the settlers, fills the agitators, agitates the pulp, and charges the filter presses. The total cost of treatment, including repairs and renewals, amounted with this practically new plant, working on methods differing from the others on the field, to 23s. per ton.

It is highly interesting to note that the principles underlaying the method of treatment are the same as adopted by Mr. Moss in 1899, and it is the highest testimonial to his skill as a metallurgist that the tailings from this mine—probably the only one in Kalgoorlie—were never rich enough to be re-treated. The residues now pro-



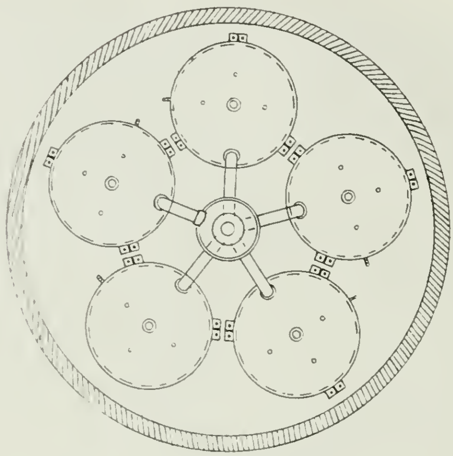
The Kalgurli Gold Mines.

Diagramatic Elevation of Five Tanks. Pneumatic Agitation and Filter-press Pressure Tanks.

C. H. Barnes, Constructing Engineer

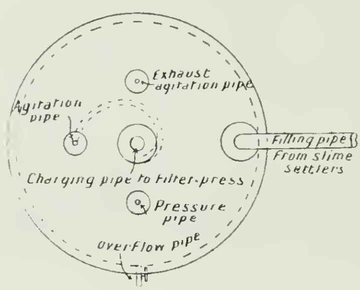
duced are officially stated to be under 2dwt. per ton. The official returns are also given according to the method continuously urged by the "Mining Standard," in standard ounces, and not as crude cyanide bullion or argentiferous gold yields, which only serves to mislead the public.

The arrangements for smelting, assaying and other necessary operations are similar to those described in other mines. The most novel feature in this plant consists in the use of the pressure tanks instead of the ordinary agitators, which have already been characterised as the crudest appliances on the field. The pressure tank is a modified montejus, in which the gold is dissolved, the agitation being supplied by means of compressed air, which keeps the pulp



Suggested Arrangement of Five Tanks.

in constant agitation. Curiously enough, a method based on the same principle, known as the White-Simpson process, was introduced at Stawell (V.) some years ago. The pulp is run in at the top of the tank; a current of compressed air is turned on through a pipe reaching almost to the conical bottom of the tank. Agitation is thus kept up for about three hours, when solution of gold is

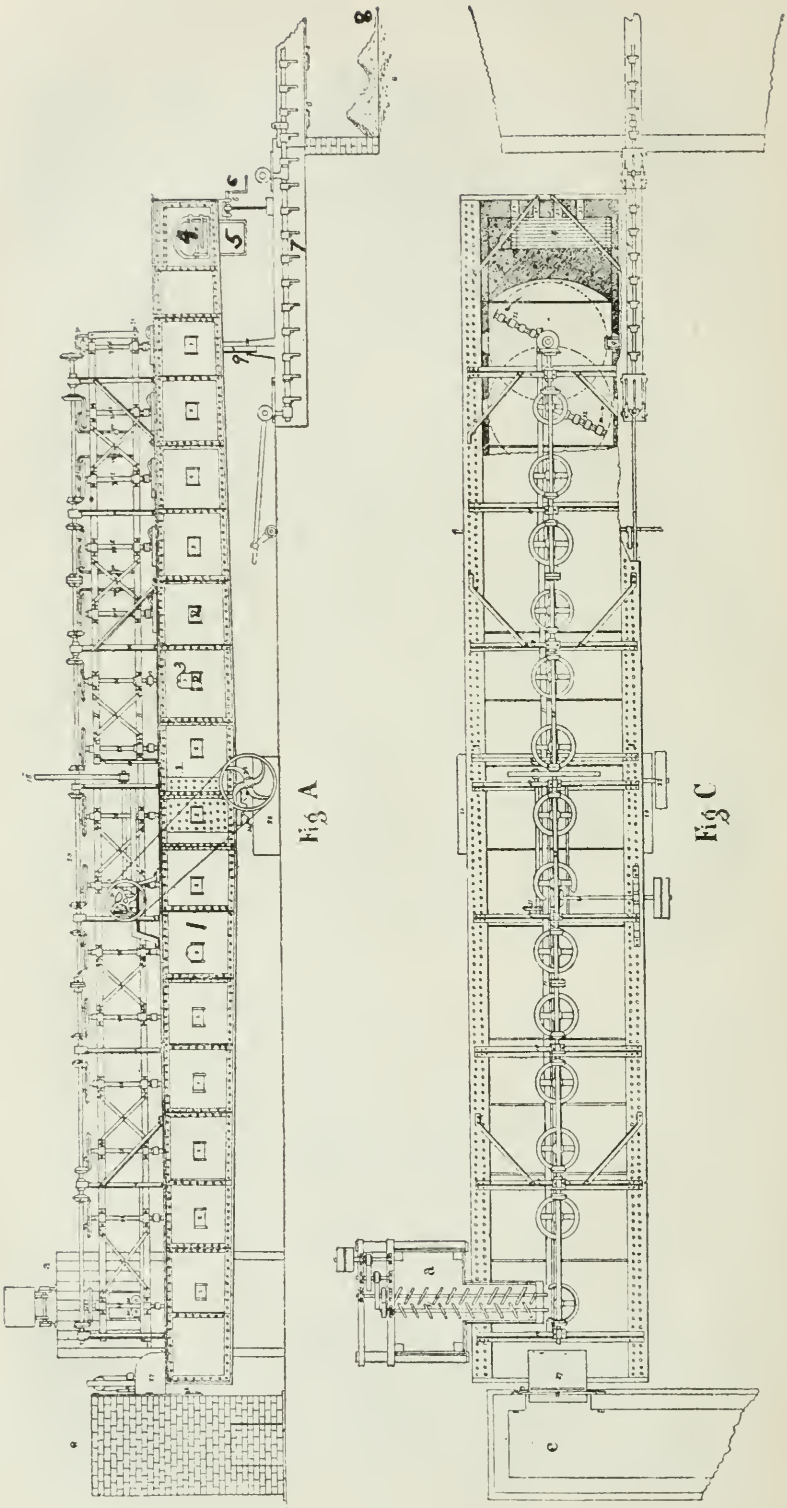


Plan of Top of Tank.

said to be practically complete. This in itself is a great stride forward in point of time beyond that taken at other works. A pipe is provided for air escape, also a discharge pipe through which the slimes are sent to the filter presses.

The Edwards' furnace, which forms so conspicuous a feature of the Kalgurli plant, is designed and manufactured by the well-known Ballarat maker. This furnace is of the McDougall type. The form of McDougall furnace generally adopted for roasting has a series of horizontal circular floors, ranged one over the other. The ore falls from one to the other, until it is discharged at the lowest. Rabbling is effected by attaching rabbling arms to a vertical shaft. McDougall found that this type of furnace caused too much dust, and was therefore not suitable for roasting fine pyrites in the manufacture of sulphuric acid. To avoid this dusting, such as is caused when hot sand drops from shelf to shelf, he designed a long hearth reverberatory furnace, slightly inclined, with revolving rabbles, which stirred the sand, and caused a downward flow of ore from the feed to the discharge. Mr. Edwards' first furnace was practically McDougall's. The ordinary reverberatory, with the inclined hearth, was built. A series of rabbles were driven from vertical shafts passing through the crown of the furnace. From this it was seen that the discharge of the sand mainly depended upon the angle of inclination of the hearth. This naturally led to the construction of a furnace whose inclination could be adjusted at will. This was arranged by boxing the furnace in and suspending it on trunnions, thus making a cantilever of it. Details of the furnace may be obtained from the drawings given. The casing consists of riveted iron or steel plate stiffened with angle irons. The bottom is made of corrugated (No. 14) iron. The interior is lined with ordinary bricks. At the Kalgurli mine Mr. Frank Moss dispensed with the flooring of brick, and used sand. In Fig. A 1 represents the side plate, 2 and 3 the small doors of sheet iron; these are used for inspection only; 4 is the door of the fireplace, 5 the ash pit, 6 the gear for raising and lowering the furnace, 7 the push conveyor for moving the hot sand after its discharge, 8 the cooling floor, 9 the discharge shoot from the furnace; a is the hopper and feed gear. The driving gear, rabble, discharge shoot, and cross section of the furnace are shown in Fig. B. The plan in Fig. C. The fireplace end, showing air holes, in Fig. E.

The sand and concentrates are fed into the hopper A; the feed gear, which consists of two spiral conveyors placed side by side, works the ore, even when wet, into the furnace with a speed proportionate to that of the rabbles. This feed arrangement of Mr. Edwards' is not always followed. After entering the furnace, the rabbles, of which there are usually thirteen, cut under it, and allow it to flow over the edge. The upper rabbles have a wedge-shaped section, the bottom being flat. The heel of the blade almost touches the floor, while the toe describes a circle about this as centre, and almost touches the walls as it revolves. The next rabble is of the same dimensions as the last, but describes a circle in the opposite direction, and in revolving almost touches the heel of the upper one; the two circles so intersect as to leave only a very small fraction of dead space not cut by a rabble. The rabbles thus moving in opposite directions mainly tend to turn the ore over, moving it backwards and forwards. If the hearth is horizontal the ore should not move along the hearth at all, since the rabbles move in opposite directions alternately, and they are all moving at the same rate. In practice, this is not strictly the case, although the movement of sand is almost entirely governed



Edwards' Mechanical Roasting Furnace.

by the angle of inclination of the hearth. When mixed ores are being treated the workmen can always tell by the sparks in the falling ore whether it is coming down too fast, and can consequently regulate the grade by raising the end. The rabbles at the end are made hollow, and

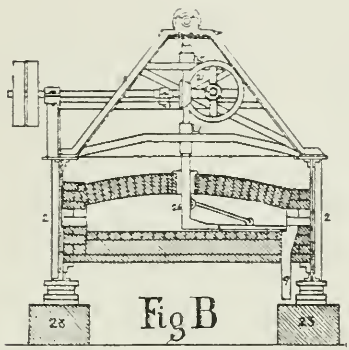


Fig B

water is circulated in them. A cast-iron shoe may be readily slipped on or off. The iron used in a medium-sized furnace amounts to about 20 tons, a ton being the heaviest part. About 1000 red bricks are used in the construction. The wood used is 5cwt. per ton of ore roasted, when an ordinary reverberatory would take 12cwt. This type of furnace is deservedly popular. Its cost is not as great

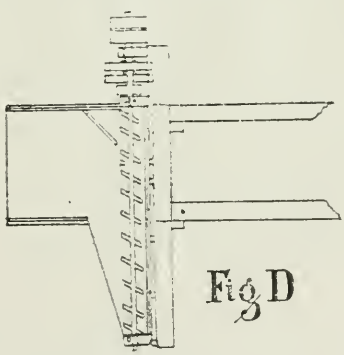
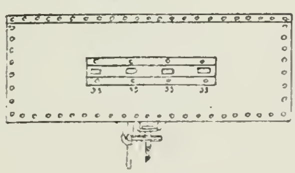


Fig D

Fig E



as many long hearth American furnaces, and it will do better work.

The Edwards furnaces were adopted by Mr. Moss, not from any bias in favor of a Victorian invention, but from bald testimony from comparative figures. At the Lake View and South Kalgurli the costs with the Brown furnace amounted to from 7s. to 9d. per ton.

The Ropp costs almost as much. The Richard shaft furnace is a solitary structure, Edwards' furnaces having been added to deal with the excess of ore produced at the Great Boulder Main Reef. The cost of roasting, as given by Mr. Moss, amounts to only 4s. per ton. At the Great Boulder the Merton furnaces now roast for 2s. 8d. per ton.

The Lake View Consols.—The Diehl Process.

The Lake View is one of those meteoric mines, whose occasional flashes dazzle, but whose re-appearance, like the comet, is a matter of conjecture. It certainly has been a wonderful mine. There are 250,000 shares in the company, and between October 28th, 1897, October 29th, 1901, it paid in dividends £1,312,500, or £5 5s. per share. The present market value of shares indicates the prevailing opinion that its brightest days are done: whether this is so or not I can offer no opinion. The policy of the management of many of the mines on the field is to keep all information confined to the corners of their board rooms. Incorrect newspaper reports may have done some harm to shareholders, but never one-hundredth the harm this policy of silence has occasioned. Judging from the plans of the mines, the lode at 100-feet level was worked for 2000 feet, at the 200-feet level from 1500 to 1600 feet, at the 300-feet for 600 feet, at the 400 for 300 feet, at the 500 for 200 feet, at the 700 for 100 feet. From the shape of the leases it is evident that 800 feet of the lode at the south eastern end will dip into the Golden Link Consolidated lease at a depth of between 700 and 800 feet. Aout the same length of lode will remain within the Lake View's property so far as it can be worked. It may be said that any erratic gold producing mine soon gets into bad odor: shareholders expect a manager to give even returns, not a feast one day and a famine the next. Whatever may be the feeling with regard to the mining and production of gold, it cannot be said that the company has not done good and useful experimental metallurgical work. There are two systems of treatment in use at the mine, one on the ordinary lines, as a dry crushing, roasting, and sliming method: the other, wet crushing, concentration, and sliming, known as the Diehl process. For many details on the latter I am indebted to Mr. G. A. Roberts, on whose advice the system was introduced, and to Mr. Brown, who was metallurgist at the mine, as well as to Mr. W. Frood, metallurgist at the Hannan's Star Gold Mines Limited.

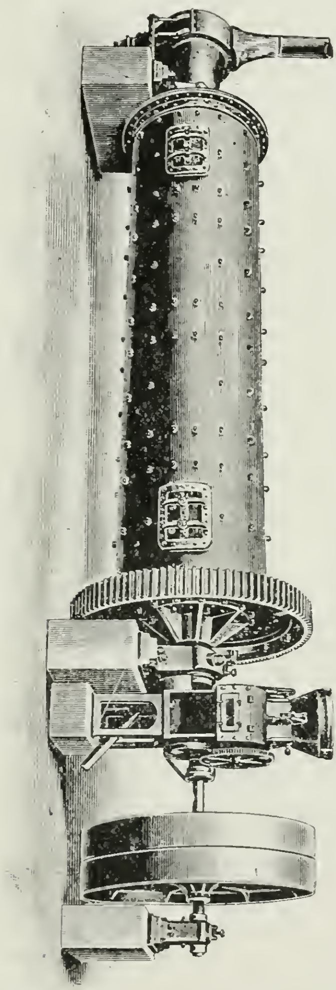
The ore is broken by two No. 5 Gates rock breakers down to 1½-inch gauge, the whole passing into a 400-ton bin. The material from the bin is conveyed by an ærial tram to a bin which supplies the ball mills. There are four of these, two No. 5 and two No. 8; the former are driven at 25 revolutions per minute, the latter at 21 revolutions. The No. 5 mills crush 25 tons, and the No. 8 40 tons, down to a size that over 60 per cent. of it will pass through a 150-mesh screen, and the balance through an 80-mesh. The dust from the ball mills is moved along by a screw conveyor to push conveyors, thence to elevators, from which it is conveyed to bins over the furnaces. The fine dust drawn off from the mills by a fan is settled in a large chamber; it is removed from this from time to time and sent to the furnaces with the other sand. The type of furnace used is the Brown straight line. There are four of these, each 180 feet long, with a 10-feet wide hearth, each capable of roasting about 30 tons per day. This furnace, instead of having a central slot and tunnel like the Ropp, has a pair of low longitudinal walls, which bound the width of the true hearth on which the ore is roasted. Between these interior walls—which do not reach the arch spanning the furnace—and the true walls is a track or rail, on which the wheels of the rabbling carriage spanning the hearth travel. There are two of these rabbling carriages, fitted with a number of stirrers of plough-share shape, which turn the ore over in their passage towards the fire-box end of the furnace. The carriages are attached to two endless steel link chains, which pass over sprocket wheels fixed on both ends of the furnace outside the roasting hearth. After passing through the ore the carriage returns over the top of the arch, and passes into the furnace again at the other end. There are sheet-iron flap doors at both ends of the furnace, which are raised by the carriages passing through, but which fall and exclude the air as soon as the rabbles have passed. Each carriage completes one complete cycle in six minutes. The fireplaces, of which there are four for each furnace, are arranged in pairs on opposite sides of the hearth.

The protecting wall for the carriage track is made by projecting fireclay tiles from the hearth below and the arch above, the two being separated by a slot through which the arm of the carriage passes. The skew backs of the arch are steel channels supported on short columns. The space between the columns from the spring of the arch to the hearth floor is closed by sheet iron doors lined with asbestos, so that the interior of the furnace may be easily reached without having to pull down the brickwork. The whole furnace is bound together externally by steel H beams. This furnace is of simple construction, but it is noteworthy that Edwards' furnaces have been more generally adopted than any other, and even on this mine the Edwards is used for roasting concentrates. Wood is used as a fuel in the Brown straight line, and it is said the ore is roasted down to 0.2 per cent. of sulphur as sulphide.

Push conveyors take the roasted sand to a Krupp chain and bucket elevator, which discharges the pulp into two collecting agitator vats holding about 50 tons each; a cyanide solution is run on, the agitator is kept going and the slimes overflow while the sand settles; the agitator is raised to keep it clear of the settled sand. When full the sands are discharged and trucked to leaching vats, of which there are ten, each holding 60 tons. Since the sand is in contact with cyanide all the time this may be looked upon as a rough double treatment system. The slimes, which form about one-third of the weight of the muddy liquors, are agitated in the usual way until the solution of gold is judged to be complete; these then pass into the monteju, and thence are forced into filter presses. The solutions are clarified and sent through zinc boxes.

The second process in use is known after the name of its patentee, Dr. Diehl. It has been installed at the Brown Hill, Hannan's Star and the Lake View. The concentrates are eliminated from the ore, the sands are reduced to a fine state of division, and these with the slimes are treated raw with bromine and cyanide of potassium. The concentrates are roasted, ground and treated in a similar manner. The essence of this process lies in the fact that telluride of gold is soluble in the solution used, although it is not in the ordinary cyanide solutions.

At the Lake View the ore is crushed in the ordinary way in a battery; the gold is amalgamated as far as possible on plates. The pulp then passes on to a Wilfley table, the concentrates eliminated going into an Edwards furnace. The roasted sand is taken back and put through a stamper box reserved for it, the escaping red sand then going through the ordinary operations common to the raw material. The coarse sand is ground by a new and novel method,



Grit Mill.

the tube or grit mill used by the cement manufacturer having been called into service as against the grinding and amalgamating pans. The latter liberates a great deal of finely-divided iron from the shoes and dies; this would use up bromine unnecessarily. The grit mill consists essentially of a wrought-iron drum on a horizontal axis, the interior of the drum being fitted with hard steel liners. Two man-holes are provided for the introduction of the flint stones, which serve to triturate the sand. The mills are made in five different sizes, from A to E, the dimensions of the last being—diameter, 4 feet 11 inches; length, 26 feet 2 inches; the weight being nearly 20 tons. Some four or five tons of waterworn flint stones, said to come from Greenland, each stone being about the size of one's fist, are put in the mill. The stones have much the same appearance as those which we see in our alluvial wash, which no doubt would serve just as well. The sand is fed through a hollow journal into the grinding drum, and escapes at the other end. By an ingenious hydrostatic arrangement, the finely-ground pulp can only overflow when it is fine enough to be carried by the ascending current of the discharge pipe. By this means the heavier particles present and also the gold must be triturated to a greater degree of fineness than the sand which escapes with them. It is a case of equal rising as against equal falling particles.

The horse-power required for the larger mills is about 30, the output being about three tons per hour, for a 200-mesh discharge, and nearly double this quantity for a 100-mesh discharge, the feed being supplied from a 30-mesh sieve. The mill requires very little attention, and is one of the most perfect sliming machines known. The slimes from the battery thickened by spitzkasten, as well as the slimes from the grit mills, are agitated in closed vats with the bromo-cyanide solutions, after which they are filter pressed and the solutions sent through the zinc boxes.

At the Hannan Star works, where the Diehl process is in operation, the ore is dry crushed in ball mills, the dust being led into a mixer, where it is moistened with a dilute solution of potassium cyanide. The pulp from the mixer is fed into a spitzlutte, the sand escaping at the bottom running over copper plates $\frac{1}{8}$ inch in thickness; these were eaten through in about two years. The coarse free gold and some of the fine is caught on the amalgamated plates; the product passing the plates passes on to a grit mill, which grinds it to pulp. Precautions are taken to send any coarse material over the plates and through the grit mill. The pulp, thickened by spitzkasten, passes into large agitation vats 26 feet in diameter and 6 feet 6 inches deep. The agitators are kept going for from 18 to 26 hours. The pulp passes into the montejus and filter presses, the escaping liquor being passed through zinc extractor boxes. Precipitation is said to be perfect. The addition of bromine to cyanide solutions is not to be made in any haphazard way, otherwise both solvents may be destroyed. The proportion should be one-fourth of BrCy to 1 of KCy, or with a .25 per cent. solution of KCy, .0625 per cent. of BrCy should be added. The bromo-cyanide may be added in crystals, but it is more usual to make it by adding bromine water to solutions of potassium cyanide, when the following reaction takes place:—

$$KCy + Br_2 = KBr + BrCy.$$

It is stated that if KCy is in excess that the following reaction takes place:—

$$KCy + BrCy = KBr + Cy_2,$$

and that this leads to the rapid decomposition of the bromo-cyanide, which is a fairly stable compound by itself. As may be readily understood, alkaline solutions may not be used when bromo-cyanide is added.

At the Brown Hill crushing is effected in a 20-head stamp mill, the stamps weighing 1100lb. each, with a 7-inch drop, falling 106 times per minute. Dilute cyanide solutions are used in the boxes; the pulp passes over copper plates and on to Wilfley tables, one for each 5 head. The concentrates eliminated amount to about 5 per cent., and these carry about 30 per cent. of the gold present in the re The concentrates are roasted in Edwards' furnace, the time taken being about 8 hours. The roasted concentrates are taken back to the battery, one box, as at the Lake View, being reserved for this purpose. Part of the gold amalgamates, part is caught on the Wilfley with any fused or unroasted particles, which go back to the furnace again. The balance flows away with the slimes; these pass into a series of spitzkasten, the clear water going back to the battery, the thickened slimes going to the agitators. The sands flow into two grit mills (size b), 16 feet long and 4 feet in diameter, carrying each about 2½ tons of flint stones. The slimes, which are fine enough to flow away from the mill, are thickened in spitzkasten, the pulp passing to the agitators, which are kept going from 16 to 24 hours. The escaping pulp is filter-pressed, and the solutions are run through the ordinary zinc boxes. This process has been most carefully worked out, and its main virtue appears to lie in the fact that gold combined with tellurium is attacked by the solvent. Pyrites would appear to exercise a bad effect on the solvent, they are, therefore, removed; or it may be that it is cheaper to roast them first and grind the product afterwards, rather than reduce the pyrites to slime. In most cases a large amount of the gold dissolved would be acted on by the cyanide alone. The extra cost of the special solvent and the royalty charged would, with most ores, amount to more than the increased returns of gold.

Regarding the cost of the process, it may be said to compete favorably with most of those at present in use, but, as with others, many items will be reduced as time goes on, so that present costs in such an expensive field are only comparative with others on a like footing. Dr. Diehl has been a benefactor to the mill-men for introducing a successful wet process. The dust generated by some of the dry crushing plants can be better imagined than described. A few years of it would settle those employed. The Diehl process plant at the Lake View, for 8 months ending 31st August, 1901, treated 41,054 tons for 40,120oz.; the concentrates contained 21,217 oz.; gold, per ton, being 1.494oz.; total value, £230,180; value per ton, £5 12s. 1d.; value per oz., £3 15s.; cost per oz., 18s. 11.2d. The detailed expenses per ton are as follow:—

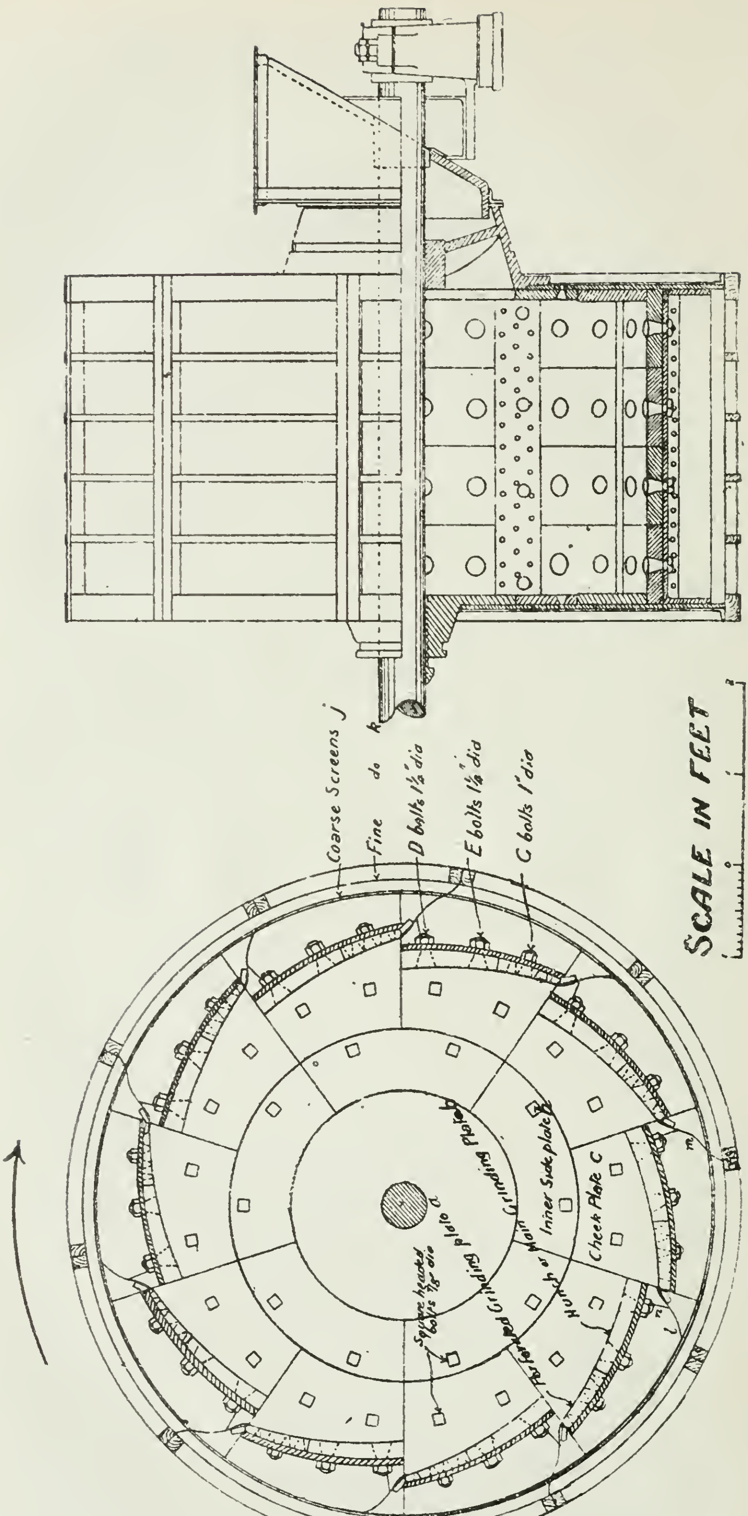
	s.	d.
Superintendence		8.755
General charges and stores...		7.502
Electric light		3.591
Assaying, retorting, and smelting		6.662
Fuel	3	9.533
Water...	4	2.367
Labor (general)	2	0.223
Engine-driving		10.315
General repairs	1	8.625
Screens, shoes, and dies...		.426
Elevating		4.184

	s.	d.
Bromo-cyanide	4	6.660
Pot. cyanide	3	5.560
Filling and emptying	2	9.381
Compressed air	1	10.192
Zinc...		1.648
Chemicals		1.078
Filter cloth...		2.359
	28	3.531
Royalty	1	9.196
Total	30	0.727

The Associated Gold Mines.

The Associated, as this company is generally called, installed one of the largest dry crushing and cyanide plants in Australia. The original scheme included crushing in ball mills and roasting and direct treatment in large vats by cyanide solution. The system was held by subsequent metallurgists to be unsuitable for the ore, and it has been modified accordingly. The works are erected near the main shaft on a ridge about 50 feet high. The plant and buildings form a prominent mark in the golden mile.

The stone as it comes from the mine is dumped on to one of a set of four grizzlies, the bars being set two inches apart. The fines pass to the ore bins, the coarse to four No. C Comet breakers. The mill is arranged into two main sections, so that each part is independent of the other. From the rock breakers and grizzlies the ore passes into two bins of 200 tons capacity each. From these it passes automatically into four Roger rolls, 36 inches in diameter by 16 inches in face. These are driven by belts, and are supposed to crush down to ½ inch to ¾ inch gauge, but are of little service now owing to the excessive wear. The crushed product then falls into revolving driers, of which there are four, 35 feet long and 4 feet in diameter, the heat being supplied by the waste gases from the roasting furnaces. These furnaces have been dispensed with in most of the other mines even when dry crushing, it being found that a small amount of moisture (3 per cent. and less) does not impede the operation. The ore deprived of its moisture passes to bins above the ball mills. There are 10 of these No. 5 ball mills, fed by means of grasshopper conveyors. The Krupp ball mill is universally used. The mill itself consists of a drum rotating on a horizontal axis. The end walls are lined with chilled iron and steel plates. The circumference of the drum consists of a number of steel grinding plates which are curved inwards at one extremity. Through perforations in the grinding plate the coarsely ground material escapes: the larger lumps are returned to the interior of the mill, the finer passes through a screen. The coarses particles in the second product are stopped by a still finer screen and returned to the mill, the fines passing through the fine screen which envelopes the mill. The grinding is effected by a number of chilled steel balls, 3, 4 and 5 inches in diameter. These, as the mill rotates, triturate the ore both by their action on the grinding plate and their attrition against the ore and against each other. The whole mill is encased in a dustproof sheet-iron casing. The mills are made from size O, having a diameter of 26 inches, up to size 8, with a diameter of 107 inches. Size 5 is that commonly used on mines, though the largest size is said to be a much better machine for a greater output. No. 5 mill is driven by a spur wheel and pinion at the rate of 25 revolutions per minute, the belt pulley being driven at 150 revolutions. From the experience of metallurgists in Kalgoorlie, it is not advisable to substitute spur wheels for belts in driving these mills. The approximate horse-power required is about 18. The whole machinery weighs about 10 tons, the balls themselves



Krupp Ball Mill for Dry Grinding. Plan and Section.

amounting to one ton extra. The output depends on the size of the material fed into them, its nature, and the size of the discharge screens used. At the Associated the design of the plant shows that each machine was supposed to do 40 tons per day, but the actual output at present is from 22 to 23 tons. This may be due partly to the Roger rolls not doing their full share of work, and partly to the toughness of the ore, and also to the important fact that much finer screens are used.

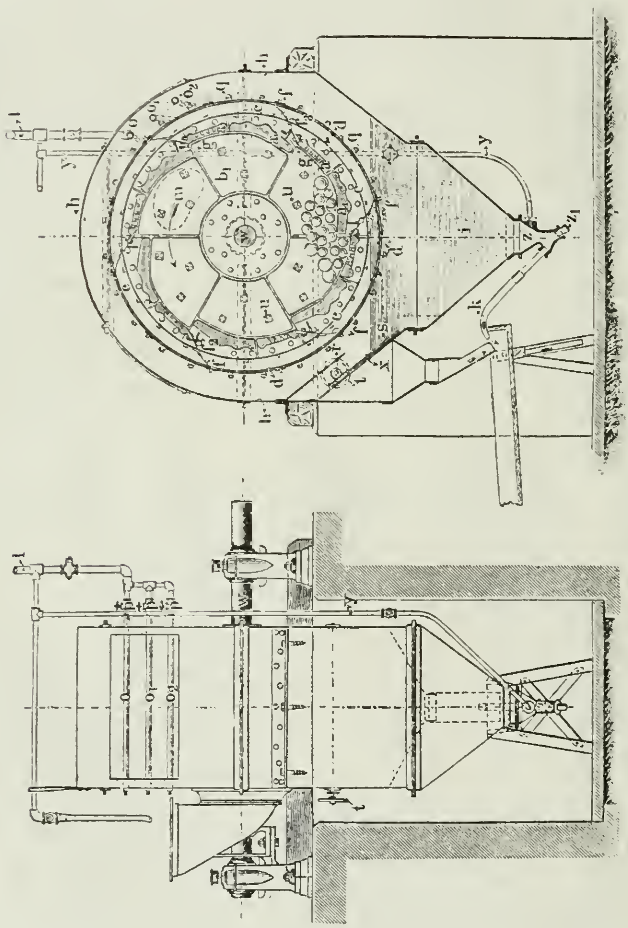
The ball mill may also be used with a continuous feed and discharge for wet grinding: they have the advantage of giving about 30 per cent. higher output. Fresh water is supplied by the tube l, which ends in the spray pipes o, o1, and o2, regulated by the valve p. The spray pipes throw water against the fine sieves of the drum. The external dust casing is transformed for wet crushing into a spitzkasten, which may receive a water supply from below at y. The overflow level may be regulated by a slide s, worked by a hand-wheel t. If desired, the overflowing slimes can be led away separately from the sands, which are delivered through the syphon pipe k.

Mr. Moss, of the Kalgurli mine, states that in dry crushing with a 40 mesh screen 60 per cent. of the material passes through 110 mesh screen, and 15 per cent. more through an 80. The capacity of the No. 8 ball mill for dry crushing amounts to 50 tons per day, or with wet 65 tons, the horse-power required being about 30. While these figures will not show as favorable a result for wet crushing against the modern stamp battery, yet for dry crushing it is doubtful if any machine will compete with them for simplicity and effectiveness.

At the Associated screw conveyors are used to force the dust along to the boot of four chain and bucket elevators, which raise the material into two 200-ton supply bins. These supply four Ropp furnaces, each 120 feet long by 14 feet wide, having six rabble cars and four fireplaces. The furnaces have been materially altered since they were first erected, the crowns having been lowered 17 inches, and a saving of 25 per cent. of fuel is said to have been effected through this alteration alone, and a better roast obtained. Even with the alteration the furnaces are clumsy contrivances, and not in any way designed on scientific principles. The furnace is plainly the one which gave rise to the Brown, Holshoff-Wethey, and such types of American furnaces. One of the main objects of all these furnaces is to put through the quantity, but this is very often done at the cost of the quality of the material roasted. It was found at Mount Morgan (Q.) that the cost of getting a good roast with a Ropp furnace was as great as with their hand rabbled reverberatories. The furnaces put through about 60 tons per day.

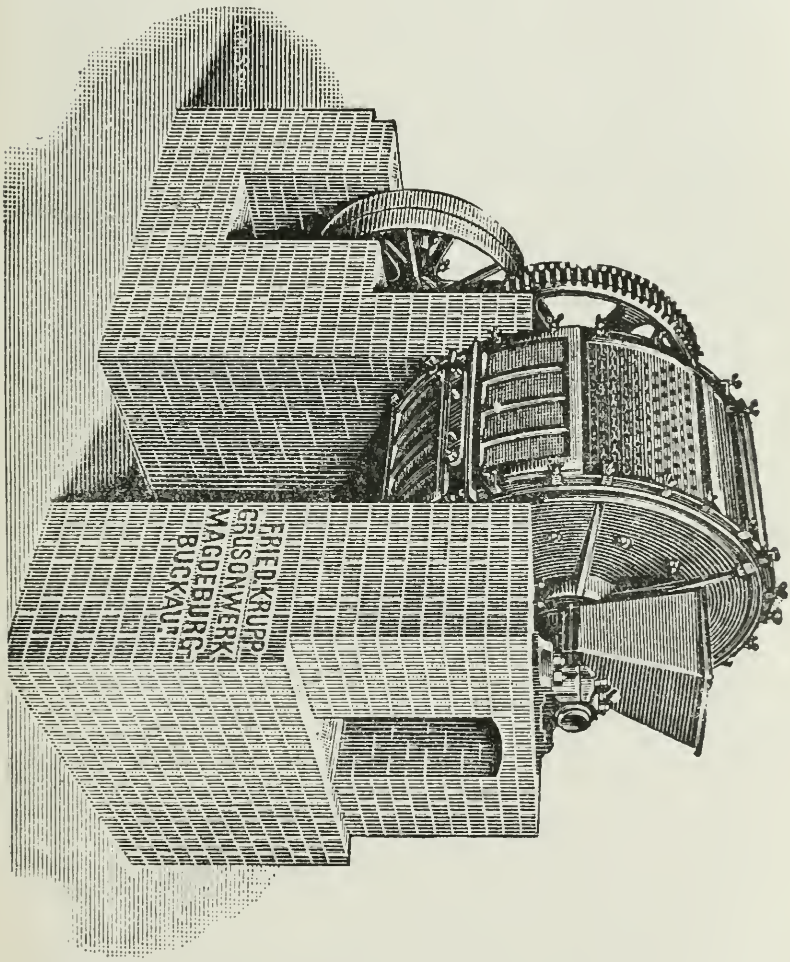
Push conveyors bring the roasted ores to a pair of conveyors which distribute it in shoots leading to four mixers. The previous method of treatment, which has been so severely condemned, was to moisten the cooled roasted sand with a dilute solution of cyanide of potassium—to expose it to the air for at least 24 hours—and then run it into the large steel cyanide vats and treat it with solutions until the operation ceased to be unprofitable. It is held at present that grinding so fine as to prevent ordinary percolation is necessary, hence the vats have not been used for cyaniding in. The pulp from the mixers passes into 24 Wheeler grinding and amalgamating pans, running at 62 revolutions per minute; about 35 per cent. of the gold is caught in these pans. Some experimental work was going on in attempting to concentrate after grinding in the pans. The overflowing material was led on to Wilfley tables, also on to a Wesley-Baird table, in order to remove any gold, amalgam, or particles of fused ore.

The slimes themselves are raised by means of an air lift, and sent to one of three conical sand separators, thence through a series of 12 spitzkasten, arranged in three of four each, all connected. The



Krupp Ball Mill for Wet Grinding.

air lift is used to a considerable extent for raising slimes and tailings. It consists of a well, or bore hole, sunk to the depth equal to the height through which the tailings are to be lifted; a pipe of the size required extends from the bottom of the well to the discharge point. An air pipe from the compressor passes into the bottom of the delivery pipe. By simply turning on a jet of compressed air, sand, water, and slime may be elevated through a height corresponding with the pressure of the air turned on. At first glance it does not look as if such an arrangement would possess a reasonable efficiency: since compressed air itself is, as a rule, a wasteful source of energy, and the method of using it does not diminish its efficiency. The advantages are the simplicity of parts, the comparatively low pressure required for elevation, about half the number of pounds per square inch for the same number of feet



Krupp Ball Mill, showing Mounting.

raised, and the ease and regularity of working. The small amount of friction and the absence of moving parts are advantages possessed by no other forms of elevators.

The spitzkasten are similar in construction to those used at the Boulder; in fact the type is the same throughout the field. The sand from the small conical boxes heading each row of spitzkasten is led back to the pans for regrinding. The clear solution from the spitzkasten is led back to the grinding pans and mixers. The pulp let out through the bottom of the spitzkasten has a consistency of $\frac{5}{8}$ or $\frac{3}{4}$ to 1; this is run into one of 12 agitation vats. These are 18 feet in diameter and 5 feet deep. They are provided with suspended paddle agitators driven at the rate of four revolutions per minute. The driving is effected by worm on a horizontal shaft gearing into a spur wheel on the suspended paddle. The strength of the cyanide solutions is made up in these vats, and they are kept going for about 20 hours. The slimes are then run into the usual montejus, and from these are forced into one of six filter presses at a pressure of 30lb. per square inch; the cakes are washed and dried at a pressure of 80lb. per square inch with compressed air. The gold bearing solutions are sent through a small clarifying press and then run through zinc boxes, of which there are six, each 20 feet long, 3 feet wide, and 2 feet 6 inches high, having nine compartments. The cleaning up room is a fine building, fitted up with furnaces for roasting the gold slime, a lead lined pan with hood for acid treatment, a press for the gold sludge, Faber du Faur smelting furnace, as well as ordinary smelting furnaces and retorts for amalgam.

As part of the original design there are 24 steel vats, 35 feet in diameter and 7 feet deep. These are now made use of for storage of water, and also for clarifying slimes or holding slimes. In some cases the cloudy liquor overflowing from the spitzkasten is led into one of these large vats, the slimes allowed to settle, and the clear solution led back to the mixers or pans; in other cases they are made use of as agitating vats by fixing a set of four agitating paddles in each.

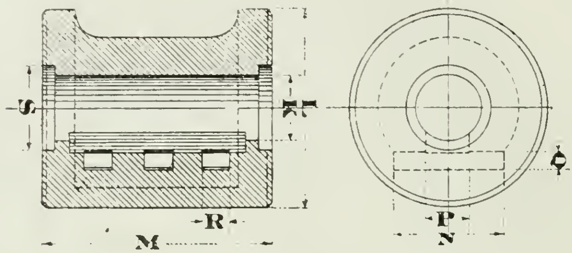
The general manager of the mine, Mr. T. Hewitson, formerly well known in the Clunes (V.) district, is strongly against dry crushing plants as adopted on this mine. Mr. John Dunstan, the metallurgist, has had an uphill task in transforming a plant designed for one method to that suited for an altogether different one. Mr. Jas. J. Dunstan, as assistant metallurgist, has had to share this task. I am indebted to all these gentlemen for the courtesy in supplying information.

The Ivanhoe Gold Corporation.

This mine, which adjoins the Great Boulder, locally known as the Ivanhoe, is one of the foremost producers on the field, and is specially interesting in that ordinary battery treatment is largely adopted. It is stated that even on the golden mile the many lodes which lie alongside each other differ widely in their mineral contents. In some the gold is wholly combined with tellurium; in others the gold is wholly free. It is therefore impossible to compare the different methods of treatment and their results, for a system which might work well on one mine would prove an absolute failure on a mine only a few hundred yards away. The alkahest of Van Helmont was just as visionary as the one method treatment for Kalgoorlie ores.

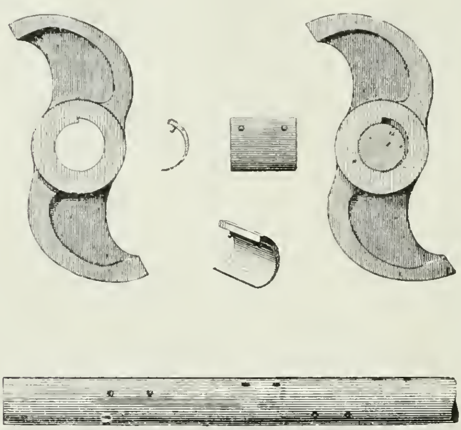
The ore is dumped on to a grizzly, with bars spaced two inches apart. The lumps pass through a No. 5 Gates breaker, the whole of the material passing into a 400 ton bin. From this bin it is trucked to the battery, which consists of 40 head, by Fraser and Chalmers, and 60 head of Australian make, the stamps in the latter weighing 900lb., and the former 1200lb. It was decided to replace the lighter stamps for heavier ones. It seems strange, from a Victorian point of view, that a stamp of 900lb. should not be considered heavy enough, but figures are against even a stamp of this weight as compared with the 1200lb. one, both as to proportionate crushing power and cost. The duty per stamp head averaged over 3.7 tons per 24 hours. With the new plant it will be above four tons. The number of drops in the old mill was 90 per minute, and the discharge took place through a screen equivalent to a 250 punched one. There is little doubt that the old order is changing with regard to battery construction. The old mill-men, with their light stamps and long drop, with measured beat and slow, are left behind in the present race for output and gold. It has been urged that the old type of batteries run on the old lines are better amalgamating machines, and are more easily cleaned up than the massive boxes now introduced. There is no doubt a good deal of truth in this, but the modern mill-man looks upon his battery mainly as a crushing machine, relying upon other methods to give him a closer extraction. Part of the blame may rest with the manufacturer, who prefers to stick to his old patterns rather than risk any innovation, but the greater part of the blame rests with those who order batteries fitted up in an antiquated way. The Western Australian field was supplied at first very largely by Australian firms, but the engineers from the Rand and America wanted heavier types than the Bendigo batteries, and introduced others from England and America. This resulted in an awakening of some of the Victorian and South Australian firms, and it is pleasing to note that many of the latest designs have been adopted.

There are a couple of details in connection with batteries which are introduced in all modern mills in the West. These are the gib tappet and the Blanton cam. The former is coming more generally into use in our States, but the latter is rarely adopted. Both have overcome the difficulties connected with jambing and slipping. The tappet consists of the usual disc, bored out so as to give a good sliding fit on the stem (K). The gib is made of forged steel, planed on the back and edges, and the front bored to a curve about $\frac{1}{8}$ inch smaller diameter than the stem. There are three horizontal rectangular slots through the tappet behind the gib. Slightly tapered keys are driven in; these wedge up the gib against the stem, caus-



Three-Key Tappet.

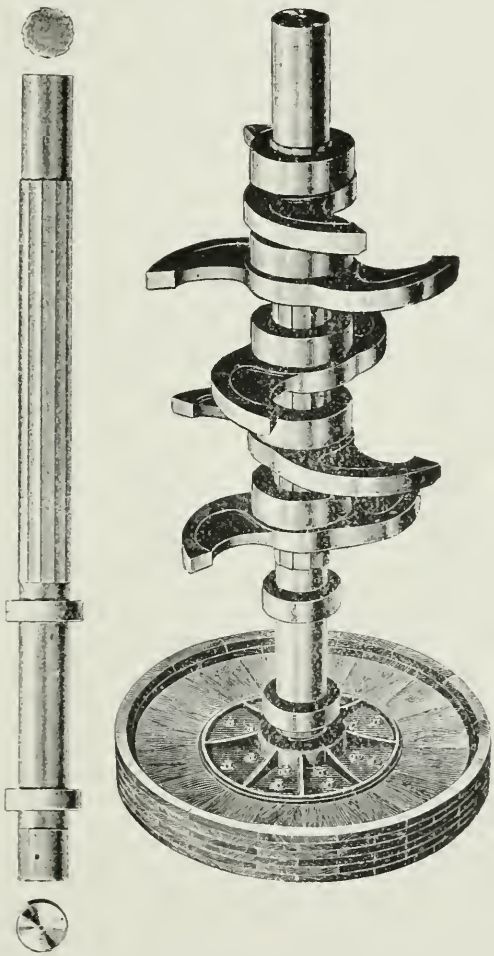
ing it to grip, and so render the whole tappet practically immovable. When it is desired to alter the drop the wedges or keys can be driven out easily and the tappet slid up or down with very little loss of time. The Blanton self-tightening cam is so simple that it should be universally used. A wedge-shaped recess is cut out of the cam. Into this fits loosely a curved wedge, shown in the figure. A couple of small set screws serve to hold the wedge in position on the cam shaft until the cam has gripped; when once



Blanton Self-tightening Cam.

this has taken place it tightens itself so as to be immovable. If it is desired to loosen the cam a tap on the back edge with a hammer is sufficient. The slotting of the cam shaft as well as the cutting of keyways in the cam itself are both obviated by this simple device. A later modification of the same principle consists in having ten taper faces on the cam shaft, all arranged at equal intervals. This arrangement permits of the ten cams being adjusted according to the order of drop, since each cam has ten equal recesses cut for the taper faces to fit into. The tightening in this case takes place over the whole shaft, and is proportional to the work required. With keys the tightening is local, and an unnecessary bursting strain is very often imposed. The cam often receives a slight tilt owing to the taper of the key.

Challenge feeders are almost universal on the mines, and this type is adopted on the Ivanhoe. The pulp passing through the



"New" Blanton Patent Cam and Shaft.

screens passes over amalgamated copper plates. Wilfley tables and canvas strakes are used for saving concentrates—the former for the coarse, and the latter for the fines. The canvas tables are 30 feet long, with a fall of $1\frac{1}{4}$ inches per foot. About $3\frac{1}{2}$ per cent. of from 12 to 14 tons per day of concentrates are saved by the Wilfley; these assay about 4oz. per ton. The fine material caught on the canvas has a very high value, probably owing to the presence of finely-divided tellurides.

The sands are separated from the slimes by means of a spitzkasten and run into Wheeler pans, where they are ground and any coarse gold amalgamated. The product escaping from the pans is pumped up and run into large cyanide vats, each 21 feet in diameter and 7 feet deep. Each vat is provided with the ordinary Butters distributor: the sands settle while the slimes overflow. The sands are cyanided for three days in the upper vats, and are then dumped into a series of vats of the same size placed directly underneath. They are treated for four days in the lower vats, after which they are discharged. The double vat system is very common on the goldfields. The beams supporting the joists are sometimes strengthened and stiffened by strutting and centre pieces, sometimes by a truss, after the type of the fink, and sometimes by vertical supporting posts resting on the floor of the lower vat.

The slimes are thickened and run into agitating vats 20 feet in diameter and 9 feet deep, where they are agitated for about 12 hours. These are then forced into filter presses and washed in the ordinary way, there being two montejus and four presses of three tons capacity, turning out 2-inch cakes. Four men are employed on each shift, and upwards of 5400 tons are turned out per month, at a cost of 6s. 6d. per ton, for this branch of the work.

About 1000 tons of concentrates had been treated, but the cost with the plant at the command of the company was so excessive that the system was discontinued. Arrangements were being made for installing a couple of Edwards' furnaces; the concentrates will then be roasted in these, ground and amalgamated in Wheeler pans, and the overflowing slime cyanided and filter pressed according to the methods already described.

The whole system of treatment commends itself to Victorians, on account of its apparent simplicity and the adaptation of each machine for its part. It is specially interesting, because it shows that when the gold is fairly free—that is, not in chemical combination—the slimes may be treated without roasting.

The total percentage of pyrites, judged by the amount eliminated, will be somewhere about 5 per cent. The amount of slimes formed by batteries runs about 45 to 50 per cent. of the stone milled. Curiously enough, when it was desired to increase the quantity it was found that a very slight percentage was added by diminishing the size of the mesh of the screen.

By dealing with the richer part of the ore by itself—in this place the concentrates—the extractions should be better than when the products are mixed. In other words, a 90 per cent. extraction from the slimes and sands, which run about 12dwt., should leave the residues worth only 1.2dwt. per ton, while if the 4oz. concentrates were reduced to 3dwt. per ton the extraction would run up to 96 per cent. The gold caught in the battery would bring the total percentage recovered to very high figures. When the material is all treated together the loss is diffused evenly through the residues, and if it amounts to about from 2dwt. to 3dwt. per ton the limit of profitable handling may be reached; while if the products were kept separate the average loss may be less than this. Notwithstanding the apparent advantages of such a system the treatment in bulk, as carried on at the Boulder and Perseverance, does not cost more

The South Kalgurli.—The Riecken Process—The Mumford Process.

The South Kalgurli Gold Mines, Limited, have the property on the ridge next to the Associated. I am indebted to Mr. J. Iles, the manager, who kindly supplied all details and gave such information concerning the Riecken process as to clear up many doubts.

The sulphide ore is broken down with stone breakers, and is afterwards pulverised in Griffin mills. The capacity of these mills and the effect of their blow on dynamite charges left in the ore have already been stated. The powdered ore, without further grinding, is elevated by bucket elevators, to the hoppers over two Brown straight-line furnaces, having a hearth area of 147 feet long and 10 feet wide. These deal with the product from the Griffin mills, about 80 to 100 tons per day. The roasted ore is sent along by a push conveyor to a cooling floor of the same length as the furnace. Rabbles pass through it as in the furnace, and turn it over. The cooled ore falls into a bin, from which it is elevated to the mixer, where it is mixed with a cyanide solution in the proportion, by weight, or one part slimes to one part solution. The fine gold is dissolved, while the coarse gold is rendered more amalgamable by contact with this solution; from this agitator, it is discharged into a Riecken vat.

How many processes have we had in which the term electricity was used as a form of abracadabra? We are now in the age of doubt about such appliances, no matter how plausible their pretentions are. Mr. Iles shared the current opinion about this process, and strongly objected to it going up on his mine. After a year's trial on many thousands of tons, he has altered his opinion about it. The vats are made of steel, each being trough-shaped, with sloping sides and curved bottom. The ends were made vertical in those first constructed, but have a slight slope in the case of the later ones, the dimensions of the vat being 13 feet 4 inches by 8 feet 3 inches on top, and 11 feet deep; when charged with the pulp, there are about 15 tons of dried slimes present. A horizontal shaft, with wooden paddles attached, passes longitudinally through the trough, stuffing boxes being provided to prevent leakage of pulp. The shaft is driven at a speed of about 12 revolutions per minute. The whole of the sides, and, in later vats, the ends, are covered iron sheets 1-8th inch thick; to these are rigidly attached copper plates 1-16th inch thick. All the plates are in sections, so as to be capable of easy removal from the vats. The copper is amalgamated, and is kept bright by a continuously flowing stream of mercury over the whole face. This is the essential feature of the process. It has been long known that gold may be precipitated on mercury as a cathode, but the trouble has always been to get it down in a coherent amalgamable state instead of a black powder. The flowing cathode appears to have overcome the difficulty, and also that connected with the crusting of salts on the linings of pans. A quantity of mercury is added sufficient to maintain an even flow for wetting the plates. This is kept in circulation by elevating it with an air lift. Mercury flows from the bottom of the vat through a pipe at a lower level; a jet of compressed air is blown in here, and the mercury is carried up between the air bubbles rising in the ascending tube. Provision is made for an air escape at the top, the mercury passing on to a reservoir. This reservoir is connected on to an iron pipe on each side of the vats by a rubber pipe. The iron pipes are 12 feet 10 inches long and $\frac{3}{4}$ inch internal diameter. They are perforated on the side facing the copper plates with 1-16th inch diameter holes, 6 inches apart. Mercury runs from the reservoir, through the rubber, and thence through the iron pipe, escaping through the holes on the sides of the vat in a number of streams By giving the pipes a reciprocating motion for a distance of about six inches the whole of the plate is moistened with a flowing stream of mercury.

So far as the vat has been described, it is simply a very good form of amalgamator. The large mercurial surface exposed, the agitation and the constant flowing stream of mercury, would account for a high proportion of gold, without any charm word such as electricity or mesopotamia being introduced. It would be an interesting problem to run such a vat for the same time on the same class of ore as that with the current as an adjunct to see what the real value of the electrical method is. In other plants a high percentage of gold is caught by simple pan amalgamation, and this easily amalgamable gold should not be counted as being specially saved by the Riecken process.

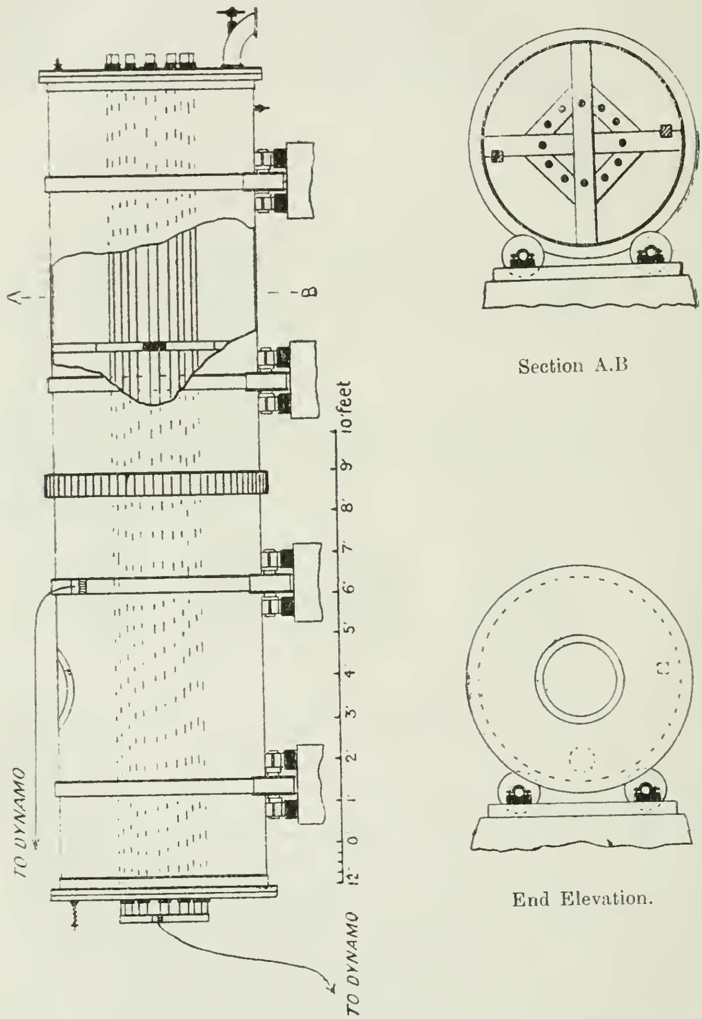
The electrical part of the apparatus is supplied by having two iron girders running lengthwise over the vat. To these are attached the anodes, which consist of iron bars 3 inches wide and 1 inch thick, with their lower ends curved so that the whole row of them remains parallel to the cathode. The latter are attached to the negative terminal of a dynamo, the former to the positive. The alkaline solution, carrying the pulp being so highly charged with salt, offers very little resistance to the passage of the current. The current used for each vat is about 200 amperes of a potential of only 2.5 volts. This represents less than one electrical horsepower. Solution of gold goes on to a certain extent within the vat itself, the amount of KCy present running from 0.05 to 0.032 per cent.; the solvent action is said to be accelerated by the oxidising action of the current. Decomposition of the aurocyanide is effected by the current, which deposits the gold on the flowing cathode, forming an amalgam: the cyanogen liberated at the iron anodes is converted into ferrocyanide, which of course are useless for further gold extraction. A certain amount of salt in solution is also decomposed, giving rise to sodium, which passes into the mercury at the cathode, and chlorine, which also attacks the iron or oxidises the solution. The iron anode is therefore a weak spot in the apparatus, and it is proposed to use carbon anodes so as to regenerate the cyanides. Ordinary carbon anodes rapidly disintegrate, but those made of graphitised carbon are durable. These are made by graphitising the shaped and baked carbon anodes. The sodium liberated is said to unite with the mercury and thereby keep it bright and active, and so give a better amalgamating surface. From actual experience, it is stated that the greater portion of the gold is rapidly precipitated, but that it takes from 12 to 14 hours to obtain the necessary extraction.

When precipitation is considered complete, the slimes are discharged by opening a valve about 6 inches from the bottom. The clean up is effected by squeezing the mercury, and by removing and scraping the amalgam from each plate. The amalgam, as might be expected from such fine material, is soft and buttery, and carries about 27 per cent. gold; this retorts to a very clean bullion. Through the courtesy of Mr. Iles the cost of working the process has been obtained.

	s.	d.	
Labor...	1	8.525	per ton.
Steam	1	0.437	per ton.
Mercury lost...	0	3.000	per ton.
KCy	3	3.597	per ton.
Repairs...	0	1.844	per ton.
Water...	0	3.000	per ton.
Assaying...	0	11.500	per ton.
Royalty, etc.	2	0.650	per ton.
	9	8.553	per ton.

At the South Kalgurli, the slimes, after having been discharged from the Riecken vat, are filter pressed, so as to obtain a dry cake, and return the water to the works. This extraordinary method of introducing a process only indicates how rich the ore must be to stand the cost of it. One could understand such a method being used in place of filter pressing and zinc precipitation, where conditions were favorable for the deposition of the sludge and the partial return of the water used; but to instal such a process in order to get over the simple, effective and cheap method of precipitation on zinc is a difficult matter to understand outside Kalgoorlie.

It is undoubted that some such process will play an important part in the future metallurgy of gold, but the large amalgamated cathode surface would lock up a large amount of bullion which small companies could not spare. For instance, after the ordinary clean up, as much as 700oz. are left on the plates, and for every square foot 3½oz. of gold. Whatever metals, such as copper or antimony, pass into solution in cyanide, will be deposited as amalgam with the gold, thereby not leaving it any cleaner than impure zinc bullion, while the destruction of cyanide and its subsequent waste are facts which cannot be overlooked. The novel feature about this process is said to be the flowing cathode; but as a matter of fact this is not new, for it was used years ago, and is in use now in the Castner process of depositing sodium on mercury. In this case, the cell or vat is tilted, through an eccentric, giving to it an oscillating motion. A rival process was started on the field known as the Mumford process, after the name of the inventor, Mr. Frank T. Mumford. The appliance used in this case for dissolving and precipitating the gold from the slime is an iron or steel barrel, with its axis horizontal. The barrel is lined with copper plate. The ends of the barrel are of wood, and through these pass a number of 2-inch iron bars, which constitute the anodes. Provision is made for admission and expulsion of slimes, also for escape of any gas formed. A plug is placed in the bottom of the cylinder to remove mercury. The current is about ½ ampere per square foot of cathode surface, and 2 volts is found to be sufficient to overcome the resistance, using water containing 2 per cent. or more of salts. The cylinder, lying on friction rollers, is rotated by means of a spur wheel at the rate of five revolutions per minute, the mercury



Section A.B

End Elevation.

Side Elevation.

Electrolytic Slime Amalgamator.

inside thus continually coating the copper plate and giving the same effect as in the Riecken vat. By making the barrel long enough, Mr. Mumford claims that his process could be made a continuous one. Owing to the relatively larger cathode surface, precipitation is said to be effected in a much shorter time. It is also claimed that the first cost of the plant is much less than that of the Riecken, that there are fewer mechanical parts, and that no air is required to lift the mercury. The method proposed to be used in cleaning up is to place sand within the cylinder, which will scour the amalgam off into the mercury, from which it may be extracted. It is not claimed for either of these processes that gold can be extracted unless it is amalgamable or passes into solution, nor that they are suitable for anything but slimes. The fullest and most candid information was supplied by Mr. Hinman, who is representing the Riecken people, and by Mr. Mumford, an old pupil of Berringer, who has carried out a larger number of experiments with his own plant.

Since the foregoing was written the plant erected at the Boulder No. 1 mine proved to be unsuitable because precipitation was incomplete. It is therefore plain that the work done at the South Kalgurli was mainly due to amalgamation.

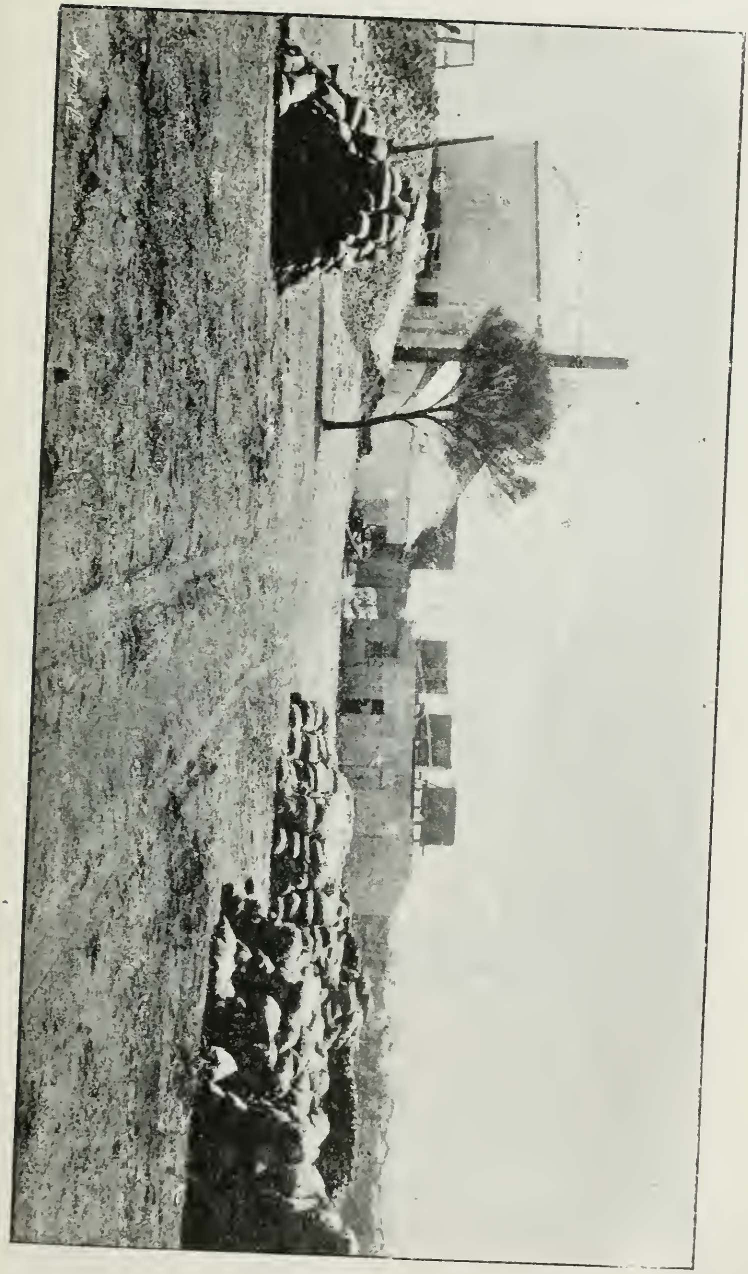
Public Batteries.

The Mines Department has largely helped to develop the Westralian mining industry by the establishment of public crushing plants. These were started in 1898. Mr. David White, of Victoria, was appointed as officer-in-charge. A large sum was expended in equipping these plants, sixteen of them having cost over £80,000, or an average of more than £5000 each. The value of the work done may be readily inferred from the fact that they have opened up and revived fields which would otherwise have languished, or from which the prospectors must have moved on through want of a suitable testing plant. During the comparatively short time they have been in operation, up to December, 1901, 66,493 tons have been treated, for a yield of gold valued at £282,624, or a quarter of a million has been won by those who most deserve support—the pioneer prospectors.

It was recognised from the very outset that the toy testing plants such as have been erected in Victoria would have been of little or no use in developing a field while the cost of working would have been unnecessarily extravagant. The smallest batteries erected in the West have five head of heavy stamps, and the larger ones 20 head. Some are equipped with a suitable cyanide plant. It is well recognised now in all places but those in which the primitive battery and amalgamation methods still prevail, that tailings from rich ore, and especially where there is fine gold, should never be allowed to run to waste without further treatment. With a public crushing plant, whether large or small, tailings having high values will accumulate. The Mines Department found that it was not possible to treat each small parcel of tailings by itself. A scheme was devised which treated the prospectors fairly, and one which shows that such matters are understood by the men who have to deal with them. The rules which follow might, with slight alteration, be taken as models for any State which desires to help the mining industry by the erection of public batteries.

Crushing.—1. All stone for treatment at the Government batteries to be estimated at 22 cubic feet per ton. 2. The manager of the battery to have power to refuse any stone considered too poor to pay crushing charges unless a deposit is paid in advance. 3 The charge per ton for crushing at each battery to be determined from time to time by the Hon. the Minister for Mines. 4. A minimum charge of £5 to be made for parcels of six tons or under. 5. Payment for crushing to be made to the manager on completion of treatment. The gold will be held pending payment of the amount due.

Purchase and Treatment of Tailings.—1. The manager of a battery to have power at his discretion to refuse to purchase any tailings, should he consider them unsuitable for treatment by the cyanide plant erected at the battery. 2. The number of tons of tailings suitable for treatment by the cyanide plant installed at the battery, and which will be purchased by the Department, to be determined by the manager, whose decision shall be final: provided in

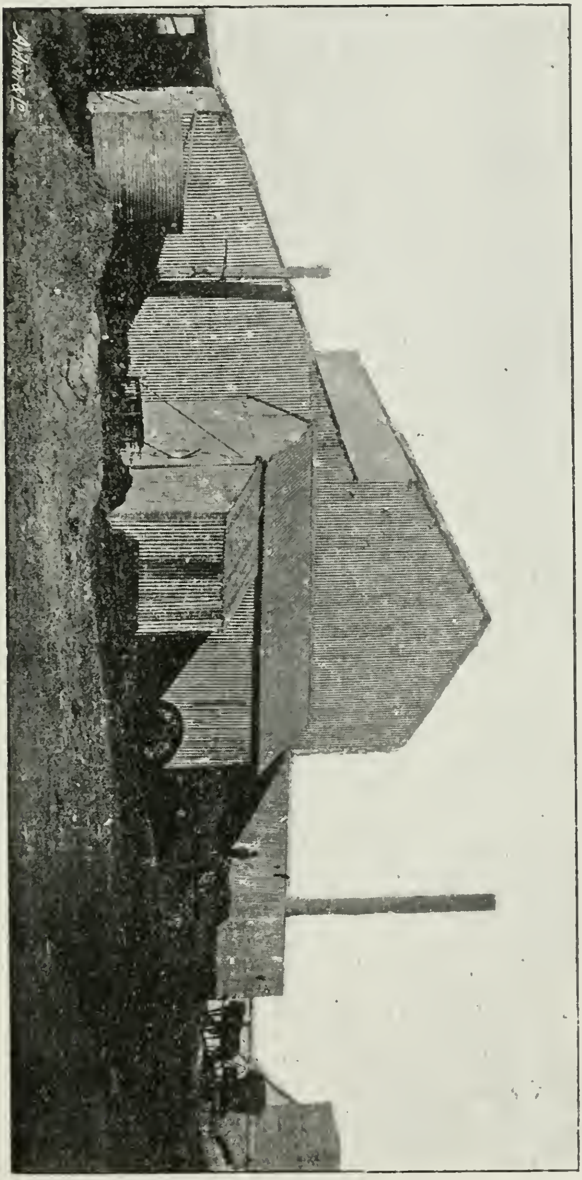


Government Battery, Mulline, North Coolgardie G F no case shall the number of tons to be purchased exceed 80 per cent. of the number of tons milled. 3. The manager to take samples of the tailings as they pass through the battery, or from the pit into which the tailings may be deposited. The tailings collected to be thoroughly mixed and divided into three samples: one for assay by the manager; one for the owner, and one sealed and kept by the manager for reference. Should the owner dispute the result of the manager's assay, and produce an assay certificate from a competent assayer, the results of the two assays to be averaged, provided that the difference between them does not exceed six grains; but should the difference in the assays exceed six grains, and should the owner refuse to accept the results of the manager's assay, the sealed sample previously referred to to be forwarded for assay to the Government Assayer, Perth, and the result of the assay as furnished by him to be considered as final. The cost of such assay to be borne by the party whose assay differs most widely from the referee's assay. 4. Upon the value of the tailings being determined as aforesaid, the Department within 31 days to pay the owner 50 per cent. of the assay value, based upon a 75 per cent. extraction, less the charge for treatment. 5. A clean-up to be made every 3 months, and the owner of the sands to be paid the balance due to him, such balance being based upon the average extraction made by the plant during that period. 6. All tailings assaying less than 3dwt. 3gr. per ton to become the property of the Government. 7. A charge of ten shillings per ton for treatment, to be the first charge on all sand treated, and the gold contents of tailings to be paid for at the rate of £4 per ounce. 8. All tailings collected by the owner to be treated at the rate of 27 cubic feet per ton. 9. The Department to reserve the right to suspend operations from any cause whatever, and to accept no responsibility for delays in treatment.

It is a pity that a weighing machine is not supplied with each battery. The system of averaging quartz and other rock at 22 cubic feet per ton leads to an undesirable laxity of method.

The rules relating to the purchase of sand indicate that the Department does not shut its eyes to the fact that there is still an amount of gold in the tailings from amalgamating mills which should be extracted. The provision with regard to the value of the tailings as determined by assay is somewhat more difficult than here assumed. If the tailings are the usual battery quartz tailings, or, say, from 10 to 15dwt. per ton, then each of the three samples if assayed separately by the same assayer, who chooses his assay sample so as to have his result correct, to say three grains per ton, will probably differ by more than the stipulated amount.

Slimes do not appear to be paid for at all, while cyanide treatment is restricted to certain classes of ore: yet on the whole, even such a start as this indicates a laudable effort not to stop at a point where metallurgical work commences. It is noteworthy in connection with this matter that in Victoria the Bairnsdale district School of Mines some years ago undertook to deal with ore at its testing plant on much the same lines as those adopted in Western Australia to-day. No restriction was placed on the class of ore sent in, which was treated by amalgamating, cyaniding or chlorinating methods: the free gold won was handed to the owners, while 90 per cent. of that contained in sands, slimes or concentrates was paid for after the cost of treatment had been deducted.



Government Battery, Mount Ida.

As may readily be imagined, such a number of public batteries scattered over goldfields hundreds of miles apart require to be very closely looked after, otherwise they may become such a drain on the State coffers that their advantages would be more than counteracted. By judicious management at every plant, and by a selection of site based on careful observation, the loss to the State should be rendered trifling, while the use of each public battery would in no way be curtailed.

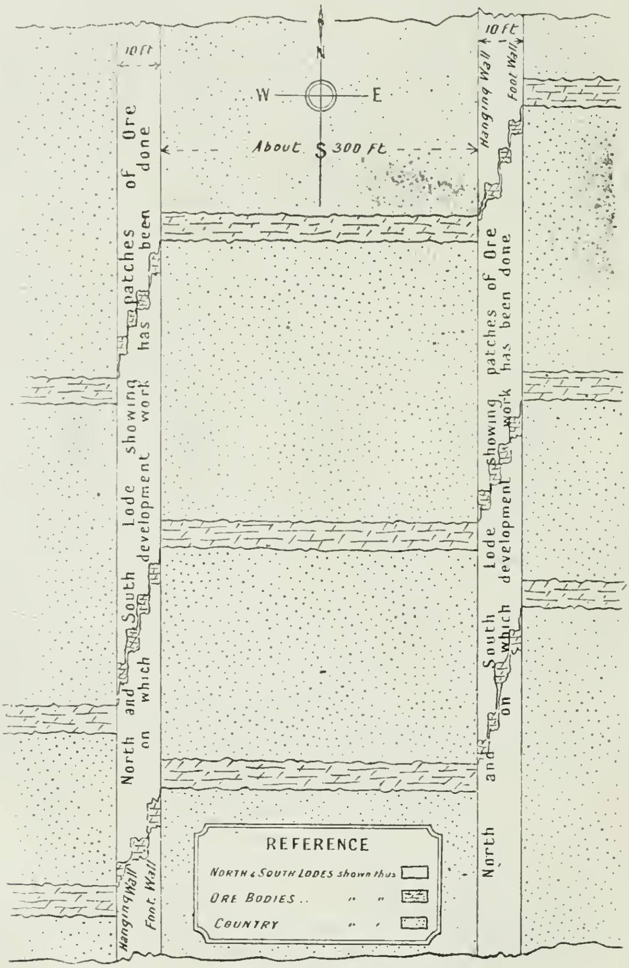
In the near future the policy to be adopted by the Mines Department should be one of leading, and not following, in methods of ore treatment. In this case not only will the simple classes of ore be dealt with, but more complex and refractory material should be successfully handled, and new sources of wealth opened up. The objection that such public plants interfere with private enterprise does not hold good, for the State does not seek to compete with existing batteries, but rather encourages prospectors to give fields a trial which would probably never be developed if some means of crushing were not available. From the very nature and purpose of these plants, it cannot be expected that they as a whole will ever be worked at any considerable profit, yet with care they will form a very important factor in increasing the wealth of the State.

The Associated Northern Blocks.

The latest plant on the Boulder is now nearing completion. Mr. George Roberts, who was metallurgist at the Great Boulder, and is now acting manager of that mine during Mr. Hamilton's absence, is general manager of the Associated Northern Blocks. He has had excellent opportunities for acquiring special metallurgical knowledge, and has availed himself of these to the fullest extent. For some years he was assayer to the Broken Hill Proprietary Co., and became chief assayer. Leaving for Western Australia in its early goldfields days, he has had a hand in the growth and development of metallurgical practice there, and the latest plant, which is almost erected, may be considered one which combines all the good points of the varied plants on the field. The ore is trucked from a shaft and tipped over a grizzly into a No. 5 Gates breaker erected over a 400-ton bin. As the ore leaves the breaker, it passes on to a Robins belt conveyor, having two trippers to distribute the ore in the bin. The ore is then fed automatically into 3 No. 5 Krupp ball mills—(Mr. Roberts would have preferred the No. 8 mills as being more economical, but the former were ordered before he took charge). The crushed material from the ball mills passes through a 30-mesh screen, and is conveyed away by a spiral conveyor to a 100-ton bin, from which it is conveyed to four Merton furnaces by means of a double push conveyor. An automatic sampler is attached to this conveyor, dropping an ounce per minute of the crushed sand. Each furnace will roast 18 tons per 24 hours. The hot sand from the roasters is conveyed by means of a Krupp elevator to a mixer, from which it passes into six Forwood amalgamating and grinding pans. From these pans it passes into a series of spitzlutte and spitzkasten; the former separates the sand from the slime, and the latter the excess of solution. The thickened pulp is agitated with a 0.08 per cent. KCy solution and filter pressed in the usual way. The presses are filled with pumps. The sands are settled in two vats and then dumped in the ordinary leaching tanks and treated the same as the oxidised ore. The plant, which has a capacity of 70 tons per day, was designed by Mr. Roberts and his engineer, Mr. Ridgway, and it is anticipated that it will do about the cheapest work on the field, with extractions equal to any.

The Mount Charlotte Gold-Mining Co.

The cost of mining on the Boulder field must strike any unprejudiced observer as being exceptionally high, and the cost of treatment until recently as extravagantly so. The ground is favorable, the lodes wide, and water has not to be contended with. On the other hand wages are high, and labor is said to be less efficient than



Plan of Lodes, Mount Charlotte.

in other States. The latter contention is contradicted by most of the managers on the field. As an offset against the general high mining costs the work done on the Mount Charlotte Gold-mining Co. deserves special mention. The figures quoted were obtained through the courtesy of Mr. H. L. Read, the attorney for the company, and Mr. Wylie, of Adelaide.

Kalgoorlie, originally better known by the name of the pioneer prospector Hannan, owes its origin to the Reward claim, pegged out on a hill only a few hundred yards from the town. The Boulder is a couple of miles further on. The brown-red soil for a considerable distance around Hannan's Hill is pitted like the old alluvial leads of Bendigo. These workings are said to have yielded more surface gold than any others in the West, and even now the fossicker or dry-blower can win his few pennyweights. The original Hannan's Reward and Mount Charlotte companies have been amalgamated, and underground prospecting has revealed a most peculiar formation.

Two parallel lodes, running almost due north and south, are each about 10 feet wide, and about 300 feet from each other. A large amount of prospecting work has been done on these lodes, but with disappointing results. It was noticed in driving that these north and south lodes were in reality faults—channels which had cut a number of east and west parallel lodes and given them a left-hand heave. These east and west lodes have proved to be payable so far as prospected. As many as 30 of them are known as exist, varying in width from 18 inches to 30 feet and 500 feet in length, and have been proved downwards for a depth of several feet.

The cost of mining amounted to 13s. 2.2d. per ton; of milling with a 10-head battery (950lb. stamps), 9s. 9.8d. per ton: or a total of £1 3s. per ton. The value of the gold won per ton of ore is £1 16s. 8.7d.; cost of mining and milling, £1 3s.; profit per ton, 13s. 8.7d.; profit for tons treated (5930), £4079 12s. These figures show that with a free milling ore the expenses are as low as they would be in a place much more favorably situated. Some good mining and metallurgical work would be done if the Boulder mines would tackle a six pennyweight proposition. Mount Morgan has done it with success, and the economical and effective work done there still remains an object lesson to the world as far as gold mining and ore treatment are concerned. Even four pennyweights in the oxidised ore, which contains about one per cent. of sulphur, is treated at a profit. Yet the stone is dry crushed, roasted and chlorinated.

General Remarks on the Boulder Field.

The Boulder is the most interesting field in Australia, and probably outside Australia, for multiplicity of methods of treatment. Out of all the chaotic processes suggested and even started on a colossal scale, the present ones have emerged. Many of the methods tried were commercial failures, yet contained in them good points now dovetailed into other systems. Very few fields have been fortunate enough to have had such scientific and technical talent focussed on them as this; the trained experience of the world has been brought to bear on the problems which have arisen; the engineer brought his most modern mechanical appliances; the technical chemists introduced apparatus and methods previously unknown in auriferous ore treatment: the silver metallurgist similarly gave suitable appliances, while the trained chemists and analysts brought valuable experience and methods from every branch of chemical industry.

The healthy rivalry on the field will bring out more facts connected with methods and machines in a single year than would be obtained in a generation from places where "Lead and I follow" is the motto.

As all difficult undertakings when successfully accomplished are apt to be made light of, so there is a tendency to belittle the work done on this field. This should not be, for while the end of methods of treatment has not been reached, greater strides in the time have been made towards it than on any other field. The fact that 15dwt. stone could not be counted as an asset in a mine only a short time ago is a blot which has been cleared away by the present men. The cost of treatment has been reduced in some instances to half of what it was a little more than two years ago, and even with the lowering of costs a better extraction is being obtained. Though it is not easy to state exactly what gold is being lost now it is probable that with such continuous plants as are in operation, occasional batches of bad extractions occur, and that these flow on to the tailings dumps with the general bulk of the material reduced to two or three pennyweights per ton. Although this loss may appear to be small, yet it is a most serious one, and is not equalled in aggregate on any other field in all the States. It certainly amounts to not less than £300,000 per annum, so that the Golden Mile is pouring gold into its tailings heaps at the rate of £1000 a day! What a chance for good men to repay their salaries to their companies some ten, some fifty, some a hundred fold.

While the Boulder men have solved, or are on the way to solve, their own difficulties, it is doubtful if their methods will be adopted on many other fields. As I pointed out before, with the exception of a small percentage of tellurides there are no refractory minerals in the ore, and the roasting of it is simple when compared with other Australian ores from which gold has to be won by chemical methods. It is claimed that after removing the coarse gold fine grinding is necessary, even after roasting, because grains of sand still contain kernels of gold. If the gold were present as a telluride it is almost certain that the temperature of the roasting furnace is sufficient to cause the dissociation of the compound and the bursting of the fragile shell, thereby exposing the metal, while if the gold were uncombined the difference between the coefficient of expansion of gold and its gangue is usually adequate to free the metal. It would be thought by anyone following Boulder methods that concentrates at least could not be treated without the elaborate grinding and sliming methods adopted. This may be necessary for some of the Boulder ores. It is doubtful whether it is necessary for much of it, because it has proved altogether unnecessary for rich concentrates elsewhere, which can be reduced from 8oz. to 2dwt. or 3dwt. per ton. Fine grinding as a corrective to indifferent roasting is of advantage.

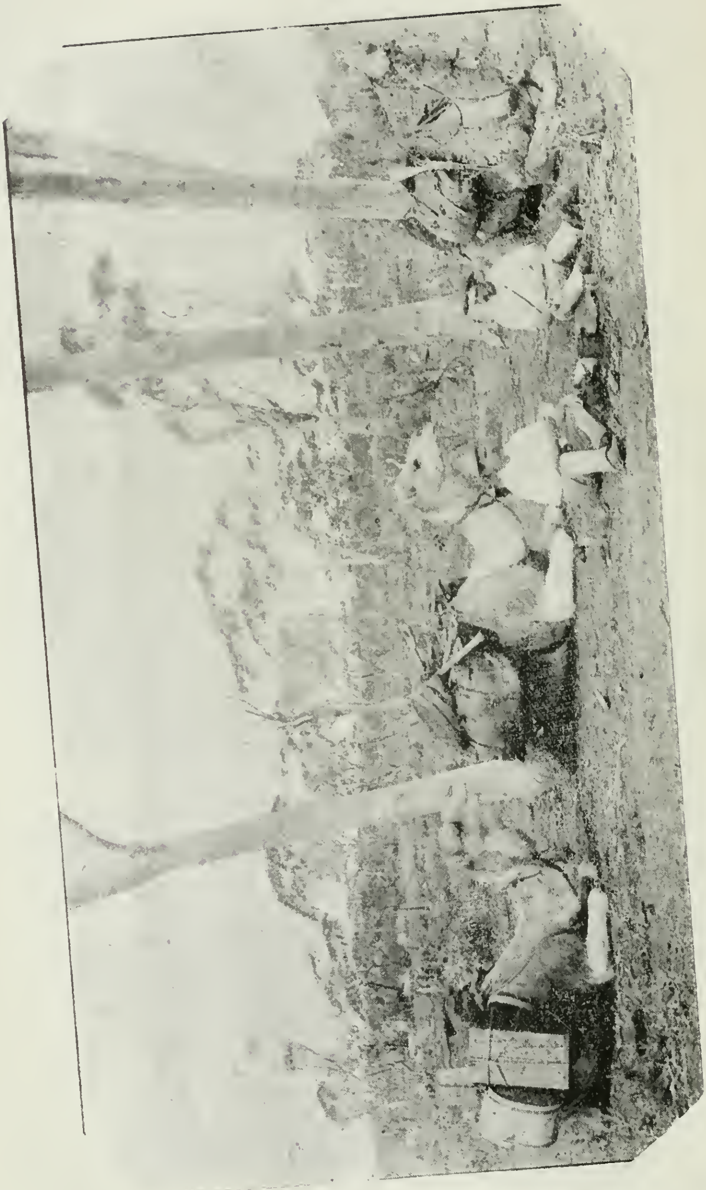
Filter pressing is a necessity on the Boulder, but it is not likely to displace the simpler decantation methods on fields having poorer ores where water is abundant and where slimes may afterwards run to waste. Clarifying presses will be useful adjuncts to ordinary decantation processes, but the cost of installing and working with filter presses alone could not be borne by most mines outside the Boulder.

It has not been made clear that Dr. Diehl's process has any advantages commensurate with the extra cost when applied to raw ores not containing tellurides. It is extremely doubtful if it has, so that it will hardly take the place of the simpler application of cyanide solutions. The electrolytic methods have not proved successful. Smelting the ores on the field can hardly be viewed as a commercial proposition, while the entraining of ores and concentrates for nearly 300 miles to the coast seems to indicate some metallurgical weakness.

At this stage I also desire to acknowledge the general assistance given by many whose names have not appeared. The visit to this field proved a most instructive one. Much has been learned from successes, almost as much from failures, which after all guided men to success, so that the words of the poet may be parodied into

'Twere better to have tried and failed,
Than never to have tried at all.

A great deal of experimental work is still going on which, if fully recorded, would save other mining fields from unnecessary investigations and expense. This work is being carried out on scientific principles—not the usual plunging in the dark in the hope that the unexpected may happen. It can be said of the majority of the Boulder managers and metallurgists that they do not deceive themselves, and from this fact we can hope for a still further advancement in original metallurgical methods.

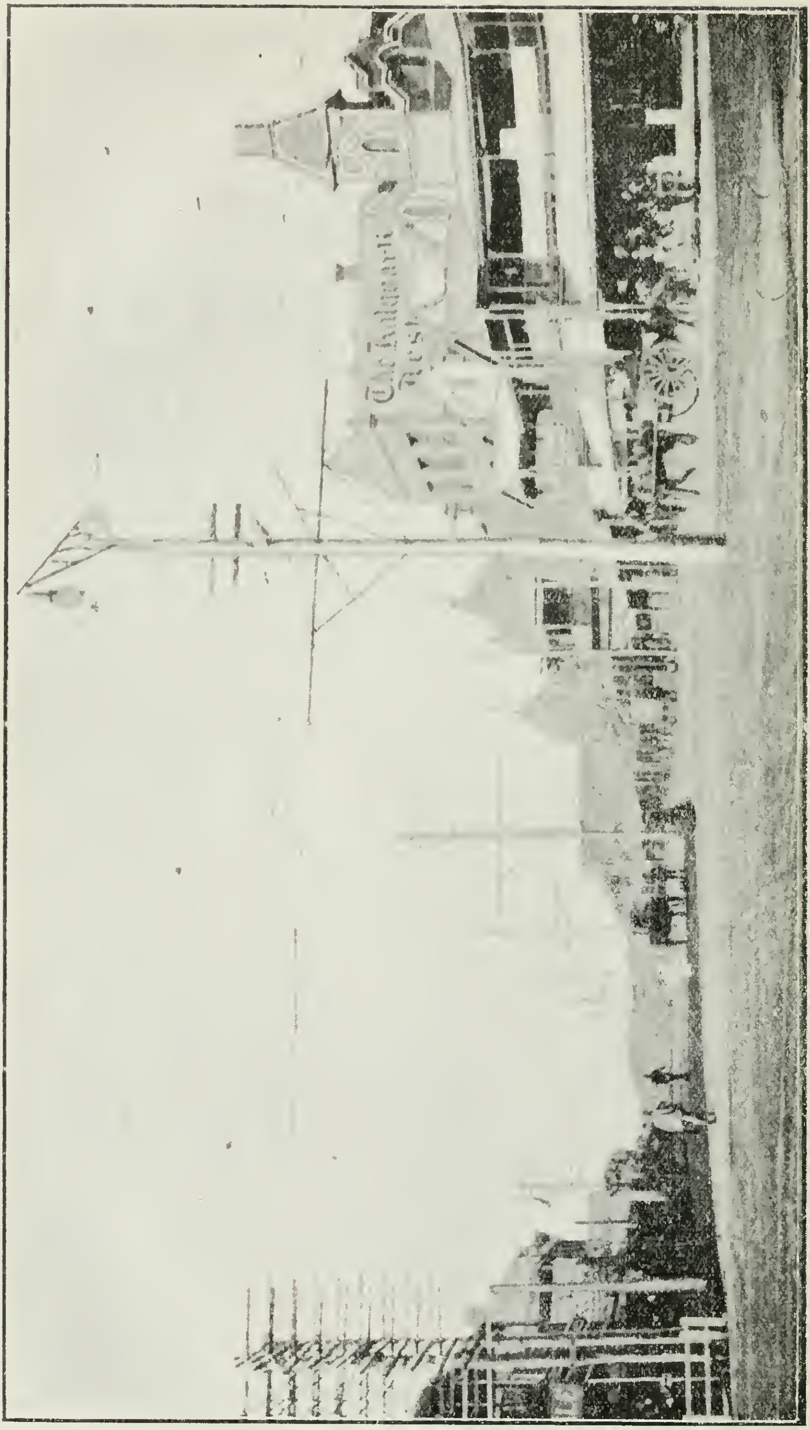


Prospecting Party—A Mid-day Spell.

The Western Advance.

How slow the progress of a colony may be in the absence of the stimulus to development furnished by gold mining was attested by the early history of the Western State. The territory was first occupied as a British possession under the name of the Swan River Settlement in 1829. The Dutch navigators who had visited it at earlier date had named the country "Provincia Aurifera," and its gold-bearing character was further avouched by the reports to that effect of the old buccaneering explorer, Dampier, who spent some time himself in searching for the precious metal on some insufficiently described portion of the north-west coast, to which he gave his name. But, despite repeated attempts, no success attended the first efforts of the early settlers to draw upon these golden resources, and with a territory eight times as large as that of Great Britain and Ireland the population was so sparse that only a small area in the south-west corner could be regarded as occupied. Indeed, so arduous was the struggle against the conditions surrounding the handful of settlers that after half a century of effort the population did not exceed 30,000; revenue stood at about £250,000, and imports balanced exports at a total of about £500,000.

Many were the unsuccessful attempts made to find gold during the earlier years of Westralian settlement, but the first promise of more fortunate result was that given by the discovery due to Mr. T. Hardman, Government Geologist, of the Kimberley Goldfield in the north-east corner of the State. This was in 1882, and as the yields were encouraging it stimulated the further search, which led to other important discoveries. The Yilgarn field, about 200 miles east of Perth, was the next important discovery, the pioneer being Mr. Anstey, and rich returns were obtained from the different divisions of the area, Southern Cross, Golden Valley, and Parker's Find. The Pilbarra field, in the north-west, was next proclaimed in 1889, and the Ashburton, at the eastern end of the Capricorn Range, in 1890. From both these gold was obtained in large quantities. Quickly followed the discovery of the Murchison by Messrs. Macpherson and Peterkin, in 1891, the district being on the Upper Murchison River, about 200 miles from the coast. Gold had previously been found in this locality at Mulga Mulga and Yuin, but it did not pay for working. On the Murchison field proper, however, the record was very different. Gold was found in liberal quantity in the outcrop of the reefs, and the rudest appliances enabled the diggers to obtain rich returns. They crushed the stone by hand, and realised magnificent results. The gold was distributed in richly-paying quantities through a number of reefs, and certain of the specimens sent from the Murchison to Perth equalled in gold-bearing capacity anything obtained throughout the world. It was stone in the gold more than gold in the stone, and the tide of immigration that had for the past few years been steadily setting toward the West now increased in strength and volume. A wonderful impetus was given to business, and



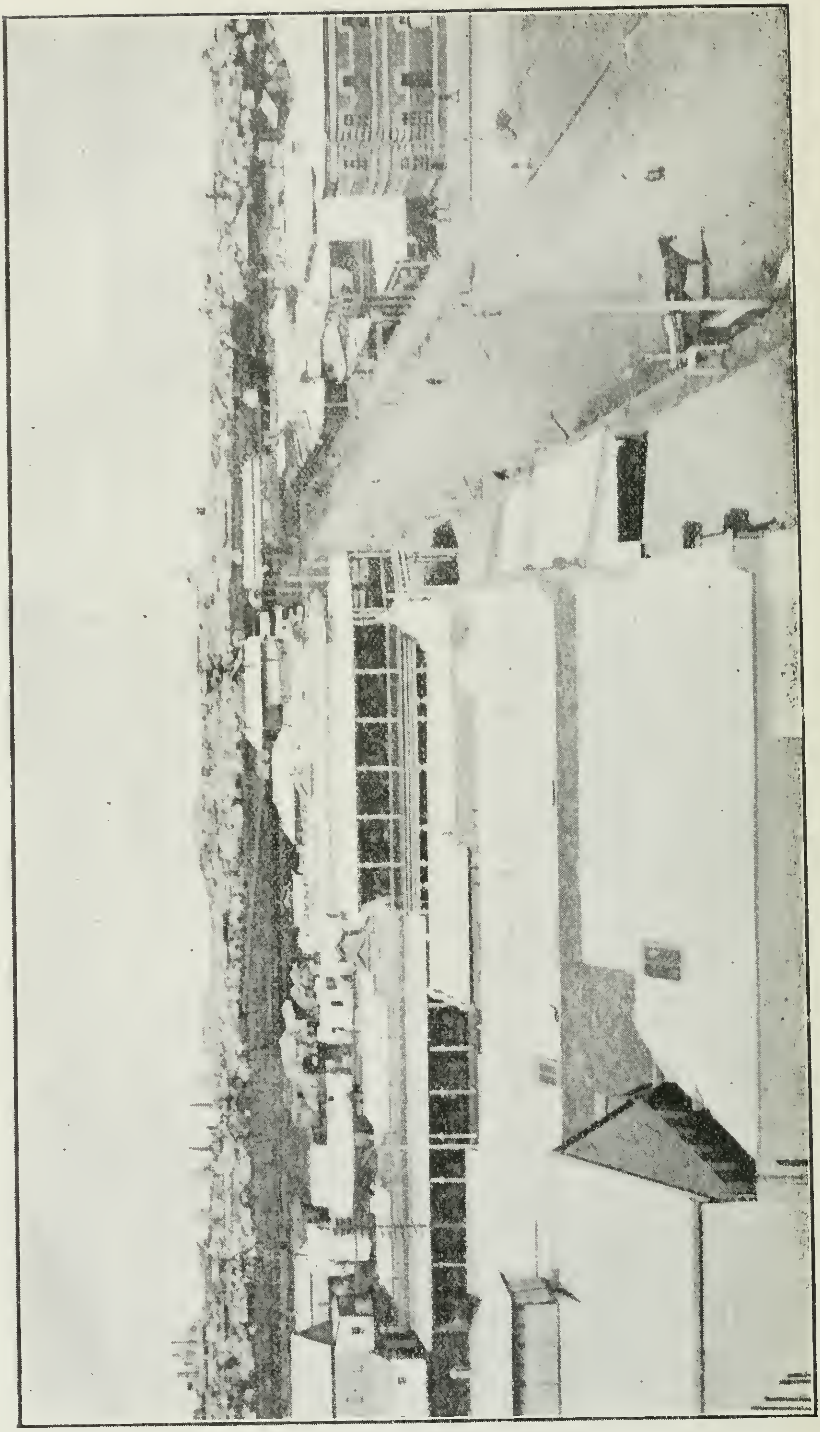
Hannan's Street, Kalgoorlie.

fresh energy infused into the work of prospecting new districts by the stream of gold now steadily pouring from the field. The output of the precious metal for 1892, the year after the opening of the Murchison, amounted to 59,548oz., of which Murchison contributed 24,356oz.; Yilgarn, 21,209oz.; Pilbarra, 12,893oz.; and Kimberley, 1088oz. The rapid advance in the gold yield to the point of the epoch-making discovery, which opened up the eastern fields, appears from the following return for the years underwritten:—

	oz.	Value.
1886...	320	£1,207
1887	4,873	19,492
1888...	3,493	13,098
1889	15,492	58,871
1890...	22,809	86,664
1891	30,311	115,182
1892	59,548	223,305

The discovery from which the great advance dates was that of the Coolgardie field, in September, 1892. This was reserved for Messrs. Bayley and Ford, who had been unsuccessful at Southern Cross, and had started out prospecting in an eastern direction. At the outset ill-luck attended them and they were returning, their provisions being exhausted, when they made the discovery which caused the rush to the goldfield, which soon obtained world-wide celebrity. At the time, however, even the prospectors themselves did not realise the importance of their discovery, and Bayley sold his reward claim to Mr. Sylvester Browne, brother of "Rolfe Boldrewood" Browne, for £6000 and a one-fourth share. Mr. Browne set to work with only two hammers and a pestle and mortar, and in less than six months he had dollied out over 6000oz. of gold. Hereupon followed the great influx from the eastern colonies, and the out-spreading of the miners from the Coolgardie centre, prospecting the country before them. Many were the discoveries made, till within a space of time almost incredibly short for such important developments, Black Flag, White Feather, Kurnalpi, the Wealth of Nations, and Menzies, were among the busiest mining centres in Australasia. These, however, have in later years been completely dwarfed by the stupendous results of that other discovery upon which the great superstructure of Westralian mining prosperity now so solidly rests, at first named after its discoverer, Hannans, and now known only as Kalgoorlie, the rival of the Rand as premier gold producer of the world.

To rehearse the story of the marvellous rise and rapid progress of the great Westralian mining centre would be to enter upon a recital all too long for space at command in these pages. It is best told in the contrast of the barren, arid waste presented but little over ten years ago with the populous, well planned, well built city of to-day, with its abundant water supply, its electric-lighting, power and traction system, its wealth of vegetation, promising to make it in the near future a city of groves and gardens, of flowers and grateful foliage, and its every up-to-dative convenience. It is told in the never-ceasing throb of its mighty machinery, in the Niagara roar of its tireless industry, and in the solid results of that well-directed energy allied to high capacity in mining science which gives the field its admitted rank as the most advanced of modern mining centres. It is told in the number, the extent,



Boulder City.

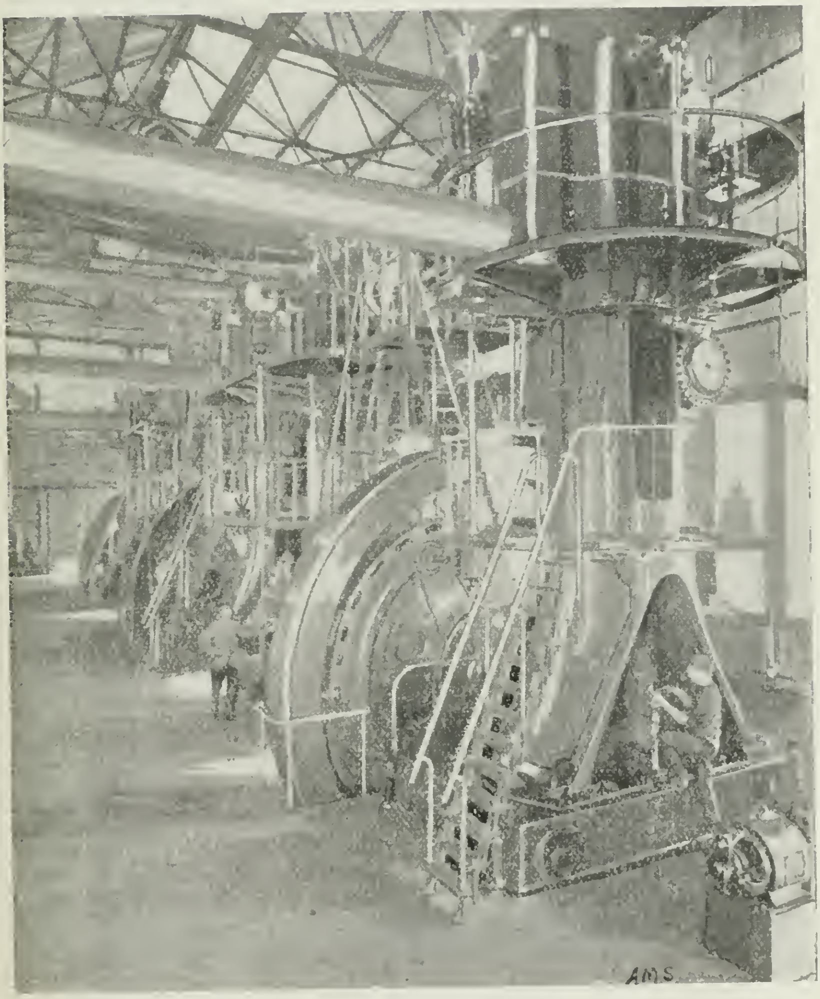
and the industrial importance of its many progressive outliers—Boulder City, Kanowna, Bulong, and the many minor centres, which are included in the East Coolgardie group; and it is still more expressively told in the statistics of steadily increasing gold production, which are given on other pages; in the record of dividends distributed, now approaching a total of ten millions sterling, and in the proven ore reserves it is estimated will, in the near future, yield from the best eight mines a monthly profit of over £200,000. Permanence and stability is assured by the fact that there is more ore in sight in the working of the principal mines than there has even been before, and that science, system, and method, which have so materially reduced the cost of extraction over the past few years, are still working in the same wealth-producing direction.

On this head, a few remarks from Mr. Algernon C. Moreing, formerly president of the Institute of Mining and Metallurgy, London, and a mining engineer, whose Westralian experience is extensive, may be quoted. He said: "A point which the general public have omitted to follow has been the extraordinary reduction in working costs which has taken place in mining in Western Australia. It is that reduction in working costs that has brought about a great part of the profit which is now being made. I will give a few instances. The working costs of the Great Fingall Company at the end of 1901 amounted to 36s. per ton; at the present time they are 19s. 5d. Think what that means when they are working, I think, 10,000 tons a month. The working costs of the Sons of Gwalia at the end of 1901 were 35s. 8d. per ton, and at the present time they are 20s. 7d. The Cosmopolitan Company's costs were 39s. 1d. in 1902, and they are now 18s. 9d. The Lake View Consols working expenses in 1901 were 56s. 8d. a ton, and they are now 27s. 9d. The Great Boulder Main Reef have reduced their expenses from 45s. 6d. to 24s. 6d., the East Murchison United from 29s. 9d. to 20s. 4d., the Oroya-Brownhill from 43s. to 26s. 9d., and the Bellevue Proprietary from 40s. 9d. to 29s. 10d. I could give many other similar instances."

Reduction in mining cost, it need hardly be pointed out, means much more than an increase in the profit-yielding capacity of rich mines. This is the least significance it has for all but the shareholders directly concerned. For the community at large, it means all the benefits which are to be derived from bringing large low-grade propositions under contribution, thus broadening the basis of the industry, enlarging the field for employment, and benefitting every branch of trade and commerce.

What has been done at Kalgoorlie in connection with mining and gold production, the metallurgic methods employed, the different processes of treatment, etc., has received exhaustive treatment in the pages preceding. What will be its prospective output has received careful consideration in a paper on "The Future Gold-Production of Western Australia," read before the Institute of Mining and Metallurgy, London, in October last, by Mr. C. Hoover, which admits of the following summary:—The author takes as the basis of his calculation the sixteen leading mines, eleven of these being in the Kalgoorlie area, which had to date yielded 4,749,815oz., valued at £17,988,790, from 3,183,116 tons of stone. The dividends from these during the year then current he estimated to exceed £1,800,000, the actual amount divided being £1,880,289, and for this year he sets the total at £2,100,000. His calculation takes into account the extent and value of the ore bodies, and the capacity of the respective plants. In January, 1903, he says, the reserve of ore in the 16 mines was 3,039,914 tons, containing 3,518,147oz. fine gold, of the value of £14,791,609. The average value of this ore, as shown by assay, was 1.12oz. per ton, but the average yield was only 1.04oz. per ton, thus proving that the rich ore shoots were being drawn on slowly. The group is now treating about 100,000 tons of ore per month, but, with extensions of plant, will before long be able to treat 120,000 tons per month. If the ore mined is added to that exposed in reserve, there is, and has been, in the mines 6,223,030 tons of ore, containing 7,757,037oz. fine gold, of the value of £32,780,399. If the factor of excess occasioned by the presence of oxidised ore is discarded, Mr. Hoover estimates the 16 mines to have an average of 7534 tons per foot of depth, for 9391oz. fine gold, or an average value of £39,688 per foot. This means, roughly, a profit of £23,000 per foot of depth. Such figures will bear a liberal discount, and still be very attractive; but there is, in the Kalgoorlie, no necessity to reserve a broad margin for error.

All the foregoing has to be taken into account in reviewing the wonderful process of expansion and development which has transformed Western Australia from the least progressive to the most progressive of the Australian States; but there also has to be considered the related items presenting every equivalent of solid value by which the progress of a whole community can be measured. As some of the most prominent among these may be mentioned the following:—Taking 1893 as the point of the great gold discovery, over the last 11 years the population of the Western State, which then stood at 65,000, had increased to 232,000; imports have advanced from 1½ millions to 7 millions; exports have increased from 1 million to 10½ millions; total trade from 2½ millions to 16 millions; shipping, in and out, from one million tons to 3½ millions, and the area under crop has advanced from 80,000 acres to 230,000 acres. Keeping pace with these giant strides ahead, the general revenue, which in 1893 stood at little over half a million, last year totalled nearly 4 millions; the advance in general expenditure on public works being proportionate. The train mileage ran in 1893 was about half a million; last year it was over 4½ millions. The leading items on the export list appear in the following figures for the year last past, with the corresponding figures for the preceding twelvemonth: Gold specie, £4,566,192—£4,149,869; unminted gold, £4,061,767—£3,318,958; wool, £443,743—£458,078; timber, £619,705—£500,533; hides and skins, £128,580—£111,456; pearls and shells, £224,322—£178,699; sandalwood, £37,913—£61,771; other articles, £162,562—£219,692. Total exports, £10,324,732—£9,051,358. The Hon. the Treasurer claims that there is no State so favorably situated as is Western Australia to survive the tightness of the London money market. This year, he holds, should see a gold production of nearly £10,000,000, which must mean expenditure of nearly £7,500,000 for production. The timber companies will pay, roughly, £1,000,000 in wages, and other industries will probably pay from £500,000 to £750,000 for the



Electric Light and Power Co.'s Electric Power Station, Kalgoorlie.

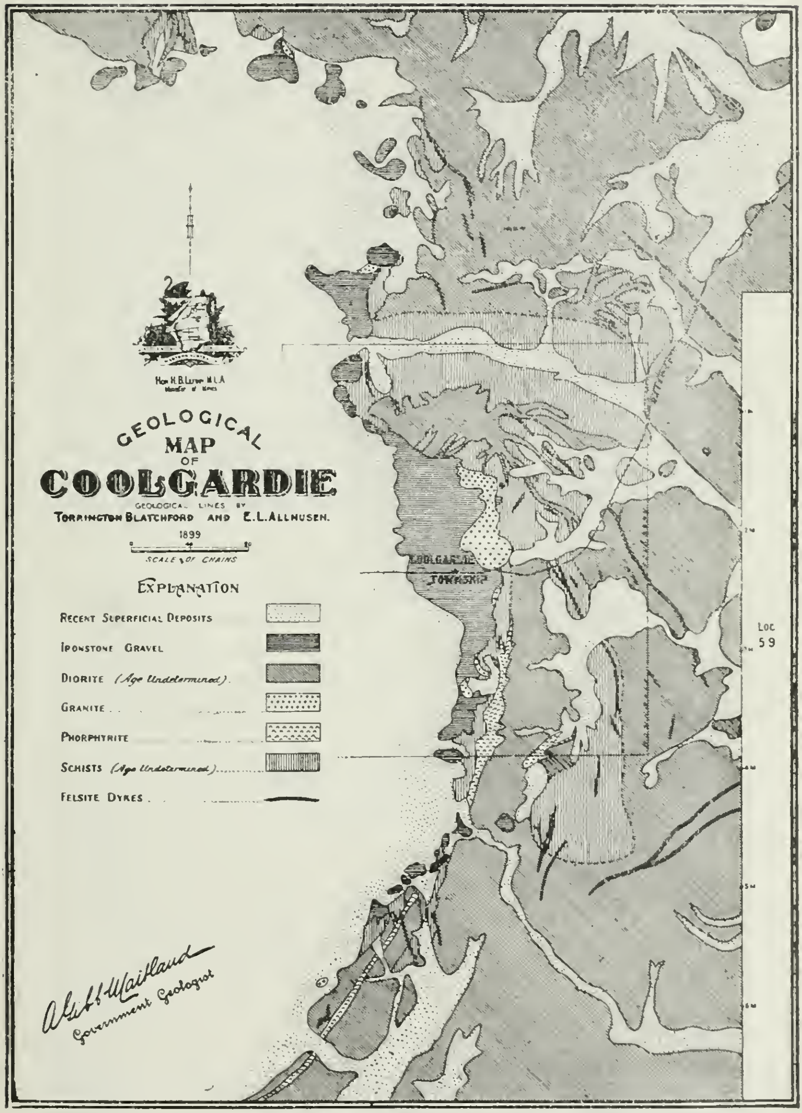
same purpose, making, roughly, £9,000,000 to be expended amongst a population of 232,000 people, irrespective of large sums put in circulation by private and other institutions. Here in itself, he claims, is sufficient prospect of keeping the State solidly prosperous, and in addition, in his opinion, there is not the slightest doubt that the agricultural development during the next three or four years will be unrivalled in the history of Australia.

As, therefore, the latter-day progress of the Western State has been most remarkable, so its prospect is most hopeful, and the attractions it presents as a field for the profitable investment of capital will go far to realise the forecasts as to its future. To this point but a limited area here and there of its vast extent has been turned to account. The value of its tin and copper resources are only now beginning to be realised, but year by year it may be reasonably expected that a better knowledge of this richly endowed State will obtain, and more systematic efforts will be put forth to lay them under large contribution for common benefit.

Gold Production and Dividends.

Westralian gold production on the higher scale commenced with the year following the discovery of Coolgardie. This was in September, 1892, and whereas the output for that year was below 60,000oz., for 1893 it was over 110,000oz. The increase had been in a healthy state of development for two or three preceding years, but from that date it shot up with remarkable celerity. Last year it was 2,335,425oz., and to that date total production reached the magnificent aggregate of 11,662,315.19 crude ounces, equal to 10,270,003.37 fine ounces, valued at £43,624,202, or an average value of £2,423,567 each year. The chief producing centre has been East Coolgardie (Kalgoorlie), which stands for over 50 per cent. of the total output, the yield being 5,846,949.10oz. Murchison is the next highest, with 1,067,473oz., Mount Margaret is second with 916,745oz., North Coolgardie third with 913,694oz., and Coolgardie fourth with 724,256oz. The Golden Horseshoe, Kalgoorlie, is the largest producer in the State, and during 1903 treated 150,873 tons for 209,054oz., averaging 1.38oz. per ton. To date the mine has produced £3,000,000 worth of gold, and has paid over 1½ million sterling in dividends. The Horseshoe has been opened up to a vertical depth of 1100 feet, but sinking to attain a greater depth is now in progress, and the end of 1904 should see the mine tested down to 1500 feet. The ore now blocked out in its workings is said to contain sufficient gold to maintain a monthly output of from 17,000 to 18,000oz. for several years, while the ore in sight, but not blocked out, is estimated to be of greater bulk and value than that referred to above. The Great Boulder Perseverance during 1903 treated 132,593 tons for 194,578oz., averaging 1.39oz. per ton. The main lodes have been proved to a vertical depth of 1100 feet. Both bodies are of big width and high average grade, but are said to show a slight falling off of values in depth. To date the mine has yielded over 2½ million pounds worth of gold, and has paid in dividends a shade under three-quarters of a million sterling. The Great Boulder Proprietary is turning out nearly 171,957oz. a year; Ivanhoe Gold Corporation, 130,000oz.; Oroya-Brownhill, 99,600oz.: Associated, 77,228oz.; Lake View Consols, 57,893oz.; Kalgurli, 45,662oz.; Great Boulder Main Reef, 13,397oz.: while other large producers include the Great Fingall Consolidated, Limited, Day Dawn; Cosmopolitan Proprietary, Limited, Kookynie; Ida H., Laverton; Menzies Consolidated, Menzies; Peak Hill Goldfield, Limited, Peak Hill; Princess Royal, Norseman; Sons of Gwalia, Limited, Leonora; Westralia and East Extension Mines, Limited, Bonnievale: Westralia Mount Morgans, and White Feather Main Reefs, Kanowna. The following table gives the total production of gold, showing the gross weight reported to the Mines Department, its equivalent in fine gold, and the sterling value thereof:—

Goldfield	Previous to 1897	1897	1898.	1899.	1900.	1901.	1902.	1903.	Total to date
	ozs.	ozs.	ozs.	ozs.	ozs.	ozs.	ozs.	ozs.	ozs.
Kimberley	12,734.00	229.30	440.17	917.15	571.15	297.06	346.40	740.00	16,275.23
Pilbarra	28,470.56	6,825.26	14,413.79	19,291.98	16,616.85	10,264.32	12,170.46	11,330.12	119,383.34
West Pilbarra	337.91	860.06	326.70	1,934.80	953.65	231.29	2,223.09	5,936.31	12,803.81
Ashburton	—	302.95	500.63	1,659.10	1,704.00	992.00	978.00	960.00	7,096.68
Gascoyne	—	13.55	13.50	333.77	74.00	90.00		..	524.82
Peak Hill	11,070.16	10,883.23	14,969.32	31,953.65	26,571.63	20,255.47	37,487.05	35,656.08	188,846.59
East Murchison	2,576.00	20,995.07	37,080.32	45,038.90	64,698.03	76,236.10	91,308.85	102,896.26	440,829.53
Murchison	140,432.28	62,316.19	79,256.39	80,548.71	105,722.31	146,591.93	210,813.86	241,791.39	1,067,473.06
Yalgoo	7,227.00	3,455.79	3,298.95	12,135.94	10,101.86	9,238.25	5,853.37	3,841.75	55,152.91
Mount Margaret	4,992.10	22,592.09	49,717.77	79,923.72	145,688.75	190,032.15	211,308.77	212,490.60	916,745.95
North Coolgardie	26,962.85	61,362.82	72,878.88	116,968.14	106,773.97	148,305.00	185,016.58	195,426.42	913,694.66
Broad Arrow	9,129.25	14,464.54	27,726.43	48,194.38	52,433.32	34,675.44	19,675.20	29,969.13	236,267.69
N.E. Coolgardie	8,975.95	40,453.10	170,441.73	112,825.45	70,745.86	63,651.70	67,108.60	62,919.74	597,122.13
East Coolgardie	143,828.70	296,764.11	422,391.86	860,371.72	737,970.98	991,378.20	1,118,615.71	1,275,627.82	5,846,949.10
Coolgardie	74,181.76	64,791.48	99,672.84	126,290.04	102,413.01	84,744.43	87,859.69	84,303.27	724,256.52
Yilgarn	94,194.60	17,072.82	11,769.40	16,371.78	29,155.42	26,587.41	23,129.69	23,615.24	241,896.36
Dundas	3,979.90	19,283.52	36,798.48	44,213.30	41,083.63	37,084.09	34,750.60	40,173.60	257,367.12
Phillips River	—	—	—	—	39.00	712.84	8,494.36	7,689.37	16,935.57
Donnybrook	—	—	14.65	511.49	453.10	3.86	100.73	58.05	1,141.88
Goldfields generally	—	—	—	1,278.90	146.56	126.78	—	—	1,552.24
Total—Gross Weight	569,093.02	642,665.88	1,041,711.81	1,600,762.92	1,513,917.08	1,841,498.32	2,117,241.01	2,335,425.15	11,662,315.19
Fine Gold	509,108.04	574,925.98	931,910.66	1,432,035.25	1,354,343.36	1,669,072.31	1,819,308.12	1,979,299.65	10,270,003.37
Sterling Value*	£2,162,554	£2,442,130	£3,958,505	£6,082,899	£5,752,885	£7,089,768	£7,727,930	£8,407,531	£43,624,202


GEOLOGICAL
MAP
OF
COOLGARDIE
GEOLOGICAL LINES BY
TORRINGTON BLATCHFORD AND E.L.ALLHUSEN.
1899
SCALE OF CHAINS
EXPLANATION
RECENT SUPERFICIAL DEPOSITS
IRONSTONE GRAVEL
DIORITE (Age Undetermined).
GRANITE
PHORPHYRITE
SCHISTS (Age Undetermined)
FELSITE DYKES
COOLGARDIE TOWNSHIP
LOC 59
Government Geologist

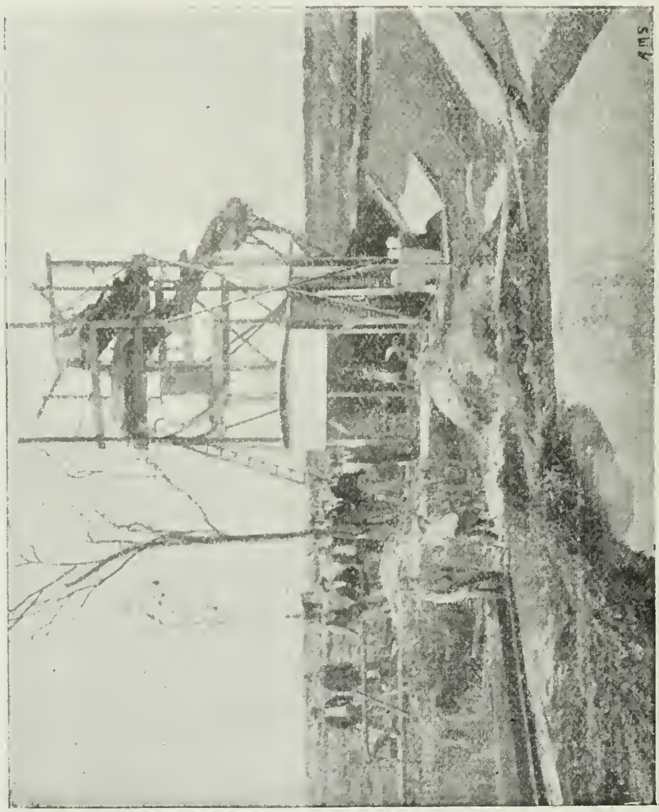
The most favorable evidence of the prosperity of the Westralian gold mining industry is the payment of dividends by the companies. Since the year 1890 the number of dividend-paying companies has increased from two to twenty-two, and there are signs of still further additions being made to the list. The distribution in 1890 was £1250; 1891, £5326; 1892, £1875; but in the next year the amount jumped to £34,350, due chiefly to £30,000 from Bayley's Reward, Coolgardie. In 1894 this company made a further contribution of £96,000, the total for the year being £110,642. For 1895 the amount distributed was £82,183; 1896, £168,216; 1897, £507,732; 1898, £606,124; and in 1899 the total reached £2,057,421, the principal contributors during this year being the Lake View Consols, with £625,000; Golden Horseshoe, £371,175; and Oroya-Brownhill £176,250. For the following year, 1900, the total reached £1,392,866; 1901, £1,093,605; 1902, £1,424,272; 1903, £2,024,155. It will be seen that the amount paid away last year was the second largest since the inception of the industry, and was the result of operations extending throughout various parts of the State. The Great Boulder Perseverance, Kalgoorlie, paid away the highest amount, £350,000, which was the same as in 1902. The Golden Horseshoe, Kalgoorlie, was second, with £270,000; the Great Boulder Proprietary, Kalgoorlie, next, with £262,500; Great Fingall Consolidated, Murchison, £200,000; Oroya-Brownhill, Kalgoorlie, £191,250; and Ivanhoe, Kalgoorlie, sixth, with £180,000. The following tabulated statement shows the

Dividends for 1903.

Company.	Goldfield.	Amount.
		£
Associated	East Coolgardie	49,539
Associated Northern ...	,,	87,500
Brownhill Extended ...	,,	45,000
Burbanks Birthday Gift ...	Coolgardie	4,500
Cosmopolitan Proprietary...	North Coolgardie... ...	60,000
Golden Horseshoe	East Coolgardie	270,000
Great Boulder Perseverance	,,	350,000
Great Boulder Proprietary	,,	262,500
Great Fingall Consolidated	Murchison	200,000
Ida H.	Mt. Margaret	28,606
Ivanhoe	East Coolgardie	180,000
Island Eureka	Murchison	1,000
Kalgurli	East Coolgardie	60,000
Lancefield	Mt. Margaret	3,960
Menzies Alpha	North Coolgardie... ...	1,000
Oroya-Brownhill	East Coolgardie	191,250
Peak Hill Goldfield ...	Peak Hill	30,000
Princess Royal	Dundas	32,000
Queensland Menzies ...	North Coolgardie... ...	19,800
Sons of Gwalia	Mt. Margaret	79,500
Westralia Mt. Morgans	Mt. Margaret	60,000
White Feather Main Reef	N.E. Coolgardie	80,000
		£2,024,155

TOTAL DIVIDENDS TO 1903.

Company.	Goldfield.	Authorised Capital. £	Total Dividends. £
Associated	East Coolgardie	500,000	308,289
Associated Northern ...	,,	350,000	262,500
Australia United	Mt. Margaret ...	100,000	4,500
Bayley's Reward	Coolgardie ...	480,000	126,600
Bayley's Gold Mines ...	,,	155,000	62,000
Bendigo and Coolgardie ...	,,	24,000	832
Brown-Hill Extended ...	East Coolgardie	100,000	45,000
Burbanks Birthday Gift ...	Coolgardie ...	180,000	99,500
Central	Yilgarn	100,000	11,305
Cosmopolitan Proprietary	North Coolgardie	400,000	79,000
Day Dawn	Murchison ...	—	2,750
East Murchison United ...	East Murchison	180,000	60,000
Fraser's	Yilgarn	78,125	25,208
Fraser's South	,,	100,000	7,500
Fraser South Extended ...	,,	125,000	3,731
Gem of Cue	Murchison ...	5,000	1,250
Golconda	,,	100,000	5,916
Golden Horseshoe	East Coolgardie	1,500,000	1,481,375
Great Boulder Main Reef	,,	130,000	50,000
Great Boulder Perseverance	,,	1,500,000	743,750
Great Boulder Proprietary	,,	175,000	1,550,550
Great Fingall	Murchison ...	125,000	406,250
Ida H.	Mt. Margaret ...	60,000	41,012
Island Eureka	Murchison ...	10,000	16,000
Ivanhoe	East Coolgardie	1,000,000	923,750
Kalgurli	,,	120,000	90,000
Lady Loch	Coolgardie ...	75,000	3,000
Lady Mary	Dundas	60,000	15,000
Lady Robinson	Coolgardie ...	48,000	1,152
Lady Shenton	North Coolgardie	160,000	120,000
Lake View Consols ...	East Coolgardie	350,000	1,317,500
Lake Way Goldfield ...	East Murchison	150,000	2,750
Lancefield	Mt. Margaret ...	25,000	14,040
Long Reef	Murchison ...	150,000	3,335
Menzies Alpha	North Coolgardie	120,000	1,000
Menzies Gold Reef... ...	,,	175,000	17,381
Morning Star	Murchison ...	45,000	14,061
Mount Burges	Coolgardie ...	80,000	2,667
Mt. Malcolm Proprietary...	Mt. Margaret ...	250,000	10,000
Mt. Yagahong Exploration	Murchison ...	250,000	6,051
Murchison Proprietary ...	,,	—	832
New Chum	,,	160,000	8,000
North Boulder	East Coolgardie	200,000	27,500
Oroya-Brownhill	,,	450,000	728,741
Peak Hill Goldfield ...	Peak Hill ...	300,000	160,666
Premier	Coolgardie... ...	50,000	21,625
Princess Royal	Dundas ...	40,000	100,000
Queen Margaret	N.E. Coolgardie	100,000	8,316
Queensland Menzies ...	North Coolgardie	33,000	89,100
Robinson Gold Mines ...	N.E. Coolgardie	80,000	7,905
Sons of Gwalia	Mt. Margaret ...	350,000	141,300



Hydraulic Sluicing Plant, Coolgardie.

Total Dividends to 1903.—Continued.

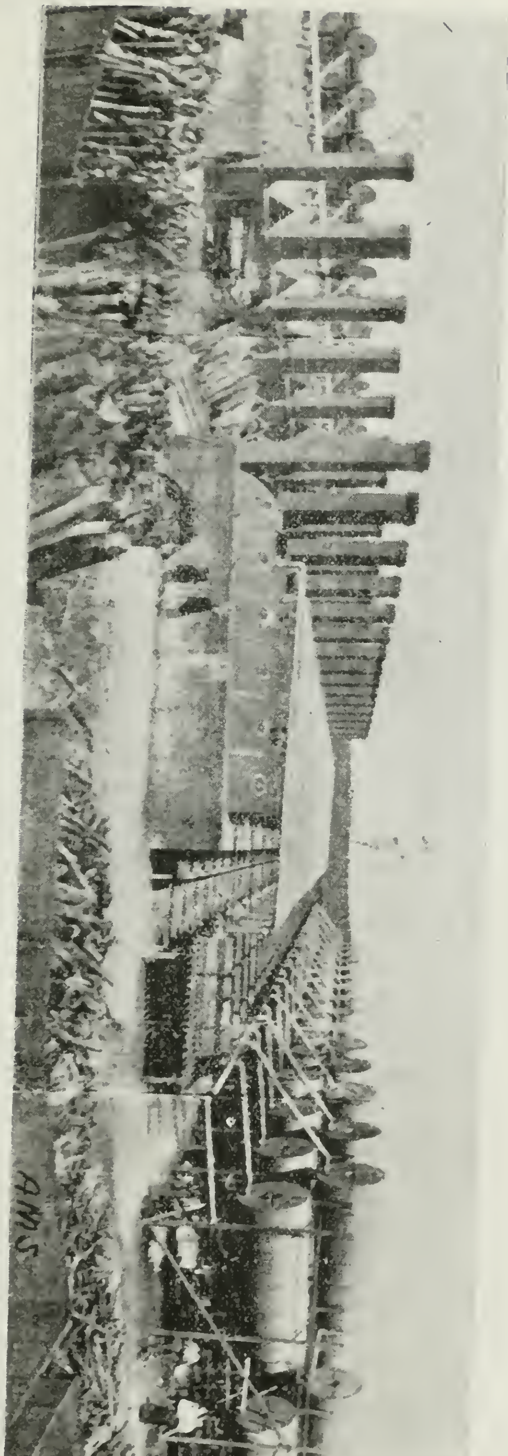
Company.	Goldfield.	Authorised Capital.	Total Dividends.
		£	£
Star of the East	Murchison ...	200,000	12,000
Vale of Coolgardie... ...	Coolgardie ...	90,000	5,625
Victory United	Murchison ...	40,000	4,000
Westralia Mount Morgans	Mt. Margaret ...	125,000	196,832
White Feather Main Reef	N.E. Coolgardie	160,000	61,070
			£9,510,017

The Goldfields Water Supply.

A moderately cheap and efficient water supply for the goldfields' centres had been the theme of those controlling the affairs of Western Australia in 1892, amongst its leading advocates being Sir John Forrest, the then Premier. He recognised that without adequate means for proper water conservation the cost of winning the precious metal would not be in proportion to its value, that mining progress would be stagnant, and that the advance of the colony, which depended so much on the active development of the mineral resources, would materially suffer. It was the gold discovery at Coolgardie, in 1892, which led to the development of the now numerous fields, and it was then recognised that sooner or later the question of water supplies would have to be faced with a firm mind. The population rose rapidly. The supplies were obtained from wells sunk in pockets or immense mamma holes around the main rocks, filled with alluvium of decomposed granite. When the supply in these pockets was exhausted the wells gave out, and were useless until the pocket was refilled by the rains. The making of the wells was very expensive, as they had to be stained from top to bottom to keep the walls standing. Afterwards tanks and dams were employed, and these sufficed for a time. The boring for water was carried on to a large extent, and with much success, but the many new rich finds in other parts of the Cinderella of Australia, and where there was practically no water, caused Parliamentary action to be further ripened. Sir John Forrest, speaking on behalf of the Government in 1893, said—"We look upon this matter of water supply as a very serious business, and we are determined to do all we can to meet the difficulty. We have already done a great deal. Unfortunately, this gold is found in localities where there is no water, and you cannot find it by sinking. We have got the boring machine to work, and we are doing the very best thing we can in this matter, and I can tell the House this, that it is costing an immense amount of money. But we intend to persevere and do all we can to solve the difficulty." Following on these lines, the route to Coolgardie was comparatively honey-combed with tanks and dams, and later on at Coolgardie an immense condensing plant, consuming 120,000 gallons of salt water per day, and producing 100,000 gallons of fresh water, was erected. On the outbreak of other fields, which have proved such important factors in the up-building of the colony, the supply was principally artificial, and condensers were largely used. In the meantime the heads of the Water Supply Department were at work devising a gigantic scheme. Many difficulties were confronted, but Sir John Forrest was not to be daunted, and when speaking at the opening of the Coolgardie railway, in March, 1896, referred to the determination of the Government to provide the goldfields with an adequate water supply, and briefly outlined the scheme, which was to "bring the water from the coast." There was no harking back: the scheme was adopted, the necessary loans to enable it to be carried out floated, and now water is carried from

Wall of the Mundaring Weir and Reservoir.

the source of supply in the Helena River to Kalgoorlie, the premier gold centre of Australasia, a distance of 352 miles, at the rate of many million gallons per day. The official ceremony in connection with the completion of this great work, was made the occasion of an imposing demonstration on the 24th of January last year, and in commenting upon it the "Australian Mining Standard" paid the following well-deserved tribute to the gentleman whose name is most prominently associated with this undertaking and its successful issue:—Sir John Forrest had reason to be proud of the position in which he found himself last Saturday, when in the presence of a distinguished company he performed the final act to that point of turning on at Coolgardie the water he had brought to it over a distance of 328 miles. In one way this great undertaking compares with the other of cognate character recently achieved, the completion of the Assuan Dam, each being the greatest modern achievement in its class of which the world holds record. In this connection the names of Sir John Forrest and the late Mr. C. Y. O'Connor may worthily be bracketed with those of Sir Benjamin Baker and the Messrs. Aird; and having regard to all the surroundings it may be questioned whether the greater credit is not due to the Westralians. Egypt is a land of stupendous engineering works, undertaken and completed during what is to us the day-dawn of a civilisation that has declined and become almost legendary. But its colossal monuments will remain in the ruins of Thebes Karnak and Luxor. The great pyramid of Gizeh, with its seven million tons of stone, dwarfs as a constructive work anything that modern history can show, and the waterworks of ancient Egypt were on the same scale of magnitude. There is a tradition that the Nile itself once flowed at the foot of the Lybian Hills, and was diverted from that course into its present bed, an undertaking so stupendous in its magnitude as to apparently verge on the fabulous; but the great irrigation reservoir of Lake Moeris, construction between the Feiyoon and the river by Amenemhat III. of the twelfth dynasty, which held 11,800 millions cubic tons of water between high and low water mark, is a work that cannot be thus discounted. Egypt is, therefore, a land which presents on every hand suggestion and incentive, but the arid western wastes of the unhistoric continent are barren of both, beyond the incentive which the need of water to develop its resources and alleviate the distress of its people communicated to the practical mind of its pioneer Premier. To undertake with a doubtful outlook, and an overburdened revenue, a pumping scheme greater than is to be found operating in any other part of the world; to bring a flow of five million gallons per day over a distance of more than 300 miles; to bid a river of water run through the wilderness and make its waste places productive, was to embark on an enterprise requiring grasp, faith and firmness of purpose. Sir John Forrest has manifested his possession of all these qualities. If he could have been daunted his scheme would have failed under the weight of discouragement and misrepresentation with which it was assailed. His working colleague could not sustain the insidious attack, and sank by the way within sight of the goal; but Sir John Forrest held on according to his wont; and has reaped a rich reward in the success that has so triumphantly answered all adverse criticism.



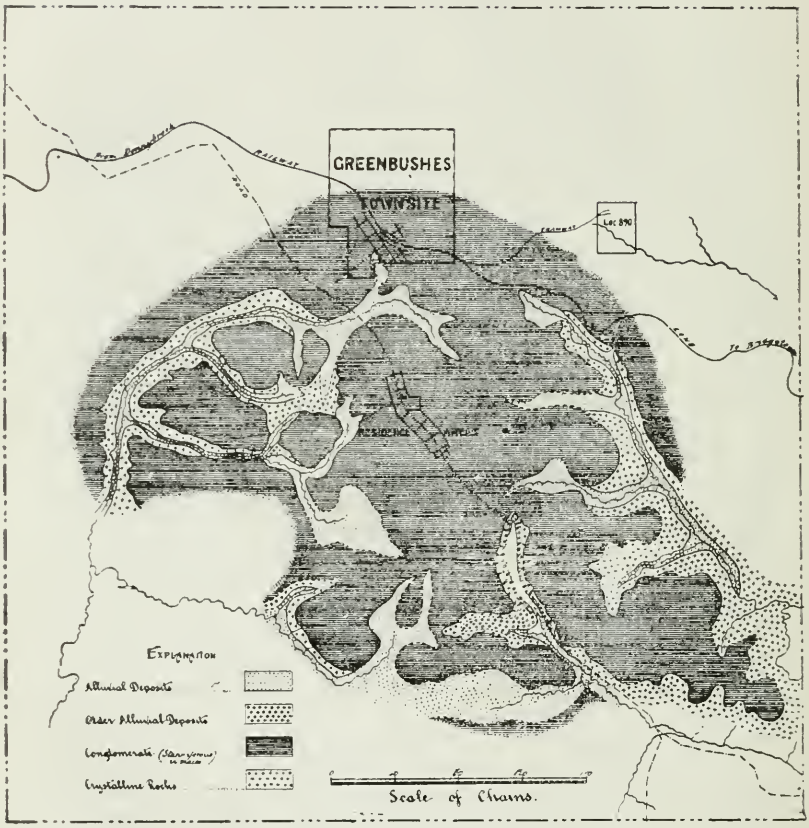
Government Condensers, Coolgardie.

Erected along the pipe route are twenty sets of pumps, distributed at eight pumping stations. The various mining companies on the Hannan's belt have agreed to purchase a daily minimum of 500,000 gallons for the next three years, to allow the salt water raised on their several mining properties to run to waste, and to purchase water from no other source whatever. In consideration of these terms the companies are exempt from the water rate, but are charged 5s. per 1000 gallons consumed. As showing the purchases by the companies for the months of December and January last, the following tabulated statement given:—

	December.		January.	
	Total Consupmtion. Gallons.	Daily Av'rge. Gallons.	Total Consumption. Gallons.	Daily Av'rge. Gallons.
Associated	1,496,000	48,258	1,652,000	55,067
Associated Northern ...	823,000	26,548	834,000	27,800
Devon Consols	275,000	8,871	350,000	11,667
Golden Horseshoe	3,082,000	99,420	3,046,000	101,533
G. Boulder Main Reef	501,000	16,161	538,000	17,933
G. Boulder Perseverance	1,756,000	56,645	1,786,000	59,533
G. Boulder Proprietary	2,612,000	84,258	2,279,000	75,967
Hainault...	545,000	17,581	581,000	19,367
Hannan's Star...	381,000	12,290	558,000	18,600
Ivanhoe	2,432,000	78,452	3,239,000	107,967
Kalgoorlie G.R. Co. ...	49,000	1,580	69,000	2,300
Kalgurli...	1,021,000	32,935	1,123,000	37,433
Lake View Consols... ...	1,531,000	49,388	1,721,000	57,367
Lake View South	47,000	1,517	12,000	400
North Boulder	19,000	613	41,000	1,367
Oroya-Brownhill	1,397,000	45,066	1,600,000	53,333
South Kalgurli	127,000	4,097	178,000	5,933
Brown Hill Extended ...	59,000	1,900	60,000	2,000
	18,153,000	585,580	19,667,000	655,567

Tin.

Tin has been found in several parts of West Australia, notably at Greenbushes, in the south-western portion of the State; Marble Bar, in the Pilbarra goldfield district, and at Kimberley, but up to the present, active operations have been chiefly confined to the first-named field, though a considerable amount of alluvial mining has been done at the Pilbarra field. The total quantity and value of the metal produced in the State to 1903 was 4995.02 tons, valued at £288,172. Regarding the Greenbushes field, the principal line of stanniferous country (which varies in width from half a mile at the north to a few hundred yards at the south) can easily be traced across the field by the rich deposits of angular tin, which are met with in the alluvium. The main line is characterised, particularly in the southern portion, says Mr. H. P. Woodward, in his geological report on the field, by the great size of the tin pieces of the solid tinstone, weighing several pounds, having been met with in patches, but unfortunately a good deal of titanic iron is associated with the wash, which renders the dressing of the finer samples more difficult. At the north end of this belt, at the head of Dumpling Gully, the wash is met with over a large area, not being confined in a small gutter, as at the south, and numbers of perfect crystals of tin are met with where the wash rests directly upon decomposed porphyry, in which disseminated crystals of tin are also found. When this line of lode was prospected, the tin was found in rich bunches in a well-defined lode formation. The eastern belt does not appear to have been very rich or of great extent, as little stream tin was found, and that was of poor quality, which is rather characteristic of tin on the eastern fall. The western fall appears to be of greater extent. Rich patches of lode tin have been worked as alluvium, as from its decomposed condition, it would be almost impossible to distinguish it from wash. One of the most noticeable features in the structural geology of the field is the ferruginous conglomerate and gravel. In its mode of occurrence, the conglomerate presents one important feature, viz., that it does not form a horizontal tableland, but occurs at different elevations, and seems to have adapted itself to the original contour of the ground upon which it originated. The conglomerate covered a much larger area than it at present occupies, and denudation has gone on to a large extent since it formed part of one continuous formation. The thickness of the conglomerate is nowhere very great, operations having shown that it rarely, if ever, exceeds 20 feet. The conglomerate is not of sedimentary origin, but has apparently been formed by the alteration in situ, and subsequently cementation of the underlying rocks. In some portions of the field, this conglomerate carries a certain quantity of tin. The ore, however, is not evenly distributed throughout but seems to be concentrated in certain comparatively isolated patches. Both the modern alluviums and the residuary sands, gravels and conglomerates have yielded by far the greater portion of the tin turned out from Greenbushes. The tin-bearing granite consists of granite passing in places into a foliated and highly micaceous



Geological Map of Greenbushes Tinfield.

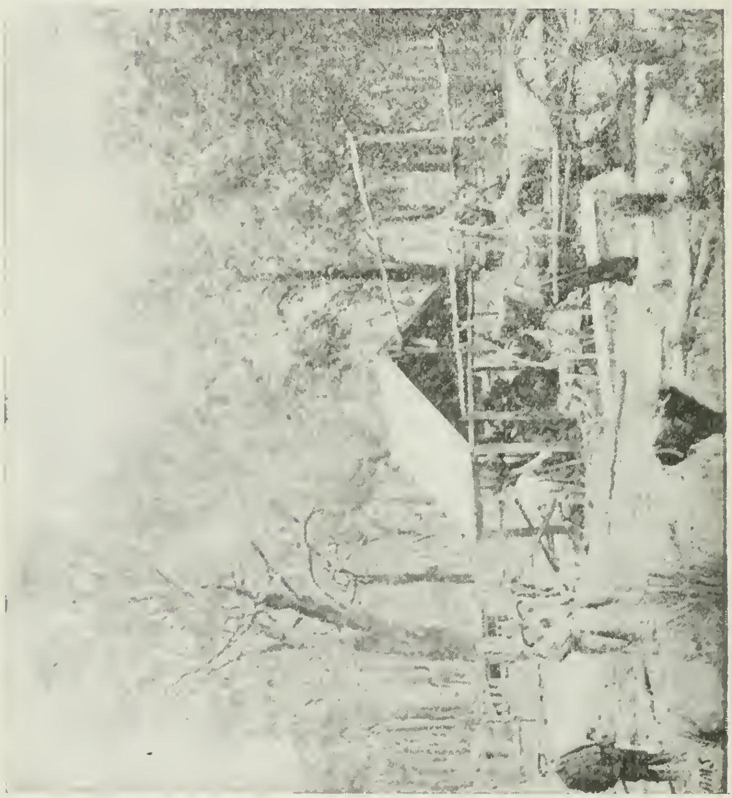
granite, with little or no felspar. This granite (greissen) contains tin, tourmaline, zircon, garnet, as accessory constituents. In some parts of the field, the tourmaline occurs in such quantity in the gneiss as to give a distinctive character to the rock, and would be better described as a tourmaline gneiss. Some specimens of this highly micaceous rock yielded on assay tin to the extent of 1.79 parts per hundred. A "tin floor" has been worked at the head of Spring Gully, on an old lease at a depth of 16 feet from the surface. The floor, a tin vein, was found underlying at a low angle to the north-west. The vein had been followed for about 40 feet to the rise, i.e., south-east. The tin, which is associated with tourmaline, quartz, and a little mica, is confined to a zone of about 1 foot in thickness. The country rock is a decomposing granite. On the southern bank of Bunbury Gully, not far from its head, another well-marked tin floor has been worked at a depth of about 30 feet from the surface. The floor underlies at a low angle to the west. The material forming the floor is about 2 feet 6 inches in thickness, and consists of mica, quartz, a little tourmaline, and tin. The deposit occurs within the zone of decomposition of the tin-bearing granite. The tin-bearing granite occupies a definite and fairly well-defined belt, trending approximately north-west and south-east from Hester's Troughs, Bunbury Gully, across the heads of Dumpling Gully, and a little to the east of Horan's claim; it includes Bishop Gibney's ground, the heads of Spring Gully, and Cowan Brook. This direction coincides with that along which lines of weakness have been produced by the earth's movements of considerable intensity; it is along these fractures that mineral-bearing solutions have penetrated and deposited the tin. This granite has been reticulated by a number of tin-bearing veins, forming a stockwork, and many have been worked in the zone of surface decomposition as alluvial deposits. The tin ore is contaminated with certain minerals of the same specific gravity, nor can they be separated by any mechanical process. These minerals are tantalite and stibio-tantalite; the former a tantalo-niobate of iron and manganese, and the latter a tantalo-niobate of antimony. Mr. G. A. Goyder (S.A.) has analysed this mineral, and given it the name stibiotantalite.

	Per cent.
Niobic Oxide Nb_2O_5	7.56
Tantalic Oxide Ta_2O_5	51.13
Antimony trioxide Sb_2O_3	40.23
Bismuth trioxide Bi_2O_3	0.82
Nickel protoxide NiO	0.08
	99.82

Specific gravity, 7.37.

Mr. E. S. Simpson, the W. A. Government Mineralogist, has analysed some tin concentrates from the Bunbury end of the field, with the following results:—

	Per cent.
Loss on ignition	0.22
Tin Dioxide SnO_2	53.14
Titanic Oxide TiO_2	0.67
Silica SiO_2	1.61
Ferric Oxide Fe_2O_3	4.11
Alumina Al_2O_3	0.42



Bonanza Crusher and Puddler.
(Spring Gully, Greenbushes.)

	Per cent.
Manganese protoxide, MnO	1.61
Lime CaO	0.69
Magnesia MgO	0.39
Antimony trioxide Sb_2O_3	15.13
Bismuth trioxide Bi_2O_3	Trace
Tantalic Oxide Ta_2O_5	19.
Niobic Oxide Nb_2O_3	3.56
	101.39

The ratio of tantalic and niobic oxides to antimony trioxide in this analysis corresponds closely to that in Goyder's analysis, and we may therefore conclude that in this specimen of dressed ore there is present 35 per cent. or more of stibiotantalite. Stibiotantalite.—Orthorhombic. Mostly in water-worn pebbles. Cleavage, not apparent. Fracture subconchoidal to granular or sub-fibrous Brittle H = 5 to 5.5, G = 6.4 to 7.4. Lustre adamantine to resinous. Color, various shades of yellow and brown. Subtranslucent to opaque. Composition, a tantalo-niobate of antimony, $Sb_2O_3(Ta\,Nb)_2O_5$. B.B. Infusible, colors flame greenish grey. Reduced to metallic antimony by fusion with potassium cyanide. In closed tube gives no sublimate by itself; on adding sulphur gives a sublimate black when hot, but brownish-red on cooling. Soluble in hydrofluoric acid; this solution on adding a little potassium fluoride, evaporating somewhat, and cooling deposits a felt-like mass of colorless crystals of potassium fluotantalite. This acid solution also in a platinum dish leaves a black stain on the dish when a small piece of pure zinc is dropped into it. Decomposed by fusion with potassium bisulphate. The following table gives the returns from 1891:—

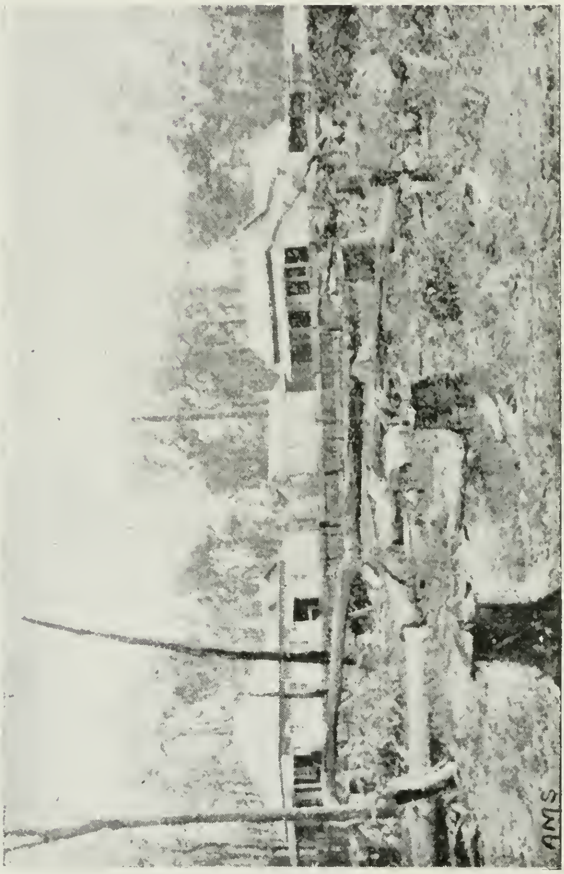
Year.		Tin Raised. tons.	cwt.		Estimated Value. £
1891	...	204	0	...	10,730
1892	...	265	9¾	...	13.843
1893	...	171	10	...	7,664
1894	...	371	5	...	14,325
1895	...	277	3	...	9,703
1896	...	137	5	...	4,338
1897	...	95	11	...	3,275
1898	...	68	2¾	...	2,760
1899	...	278	8¼	...	21,138
1900	...	435	10	...	29,528
1901	...	321	0	...	18.852
1902	...	403	0	...	24.680
1903	...	525	0	...	34,362
		3553	4¼	...	195,198

REMARKS.

The Mining Registrar at Greenbushes reports: "Of previous years there is no record either at Bunbury or Fremantle, and I believe the amount to be inconsiderable."

In 1889 and 1890, 72 tons 10cwt. of tin, valued at £5700, were exported from the Colony, which the Collector of Customs reports to be "in all probability the produce of the Greenbushes Tinfield." These figures have not been included in the table.

At Marble Bar, in the Pilbarra district, a considerable quantity of alluvial work has been done at Moolyella and Cooglegong. A promising find of lode tin has been made at Wogiña Hill, but the development work upon it has been small. The output from the Marble Bar field is:—

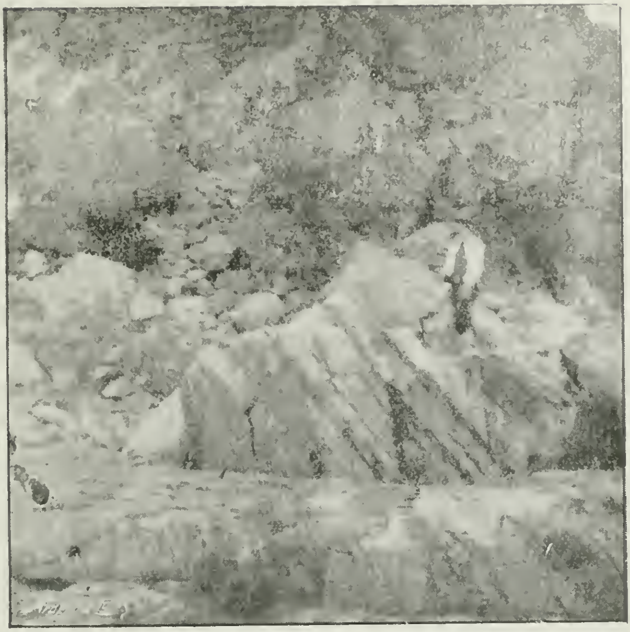


General View, Greenbushes Tin Smelting Works.

	Quantity.	Value.
Previous	Tons.	£
To 1899	75.45	4,419
1899	57.50	3.612
1900	387.87	27,174
1901	412.98	21,148
1902	216.35	15,103
1903	292.11	21,528
	1442.26	£92,984

Copper and Lead.

The production of copper and lead gives promise of playing a most important part in the future of the Westralian mineral industry. While the first discovery was made in the Northampton district in 1842, there was no production until 1855, and since then, owing to the price of the metals, the difficulties of transport and the absence of smelting facilities, operations have not been carried on to the extent warranted by the prospects. The metals have also been found at Day Dawn, Mount Malcolm, Pilbarra, Roeburne, Kimberley, Irwin River, Wongan Hills, and at Phillips River. In the Northampton district several copper mines were opened, then lead mines, the latter ultimately predominating over the copper mines in the larger output. The value of lead ore raised in the district up to October 31st, 1899, was £364,514, being the value of 33,317 tons of ore shipped. The value of copper ore raised up to the same period was £457,944 from 9349 tons of ore. For the last 30 years most of the mines, however, have been practically idle. It is conceded that there are abundant supplies of lead and copper ores, but the low prices of the metals is against their being worked at a profit. Principal attention is now being given to the Phillips River goldfield district. The absence of any local smelting works, the high cost of carriage to the coast, and conveyance to smelting works in the eastern States, combined with the low price of copper, have rendered the mining of any but high-grade ores unprofitable, and consequently the development of the copper properties has been slow. The Government within the past year took up the question of affording facilities to the companies for the disposal of the ore, and appointed Mr. J. Provis to purchase ore, advancing money at the rate of £50 per ton on its copper assay value. During the past six months nearly 3000 tons have been carted to the ore depot, on which £12,000 has been advanced. The total value of the ore stacked is, reckoning copper at £50 per ton, about £23,000, or nearly £8 per ton. In February last the Minister for Mines, accompanied by the State Mining Engineer, Mr. Alex. Montgomery, visited the field with a view to the erection of a smelter, and so impressed were they with the prospects that the Minister has decided to urge upon the Government the immediate erection of a smelter. The Government will continue in force the advancement of money on the ore, and after smelting and realising results, will return any profit there might be to the producers. About 14 months ago Mr. Montgomery made a special report on the field. He states that in going round the mines it is often found quite difficult to distinguish between the intrusive greenstones and certain greenstone schists, which appear to belong to the metamorphic series of rocks. The latter are hornblende schists, with the hornblende often altered to chlorite, and very closely resemble both in composition and appearance some portions of the massive greenstones. A question often arises whether the rock is a greenstone schist or a schistose greenstone, and it is rather probable that both exist. It seems possible



Marble Bar.

Marble Bar.

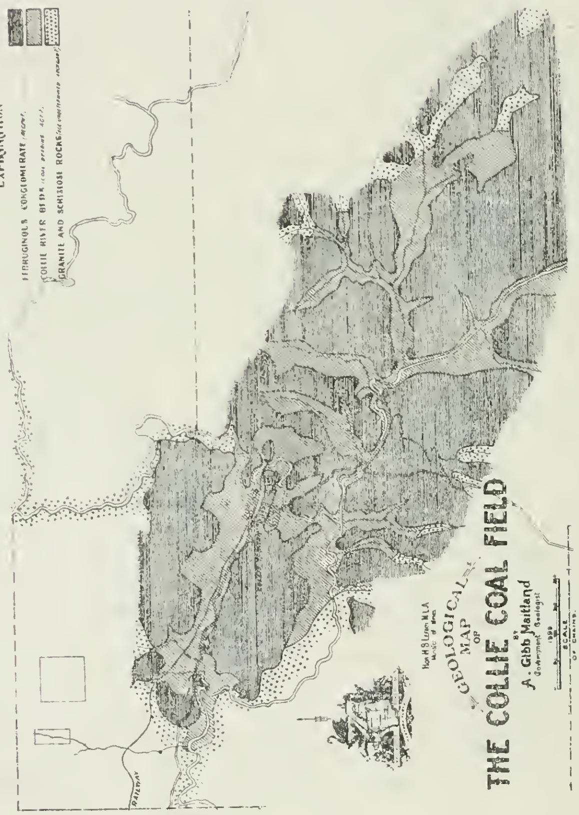
and likely that in the neighborhood of the greenstone intrusions some of the more basic sedimentary beds have been metamorphosed into amphibolite schists, while parts of the intrusive masses may also have assumed a schistose structure, as massive igneous rocks so frequently do, as the result of compression due to the stresses to which they are subjected in consequence of movements of the earth's crust. As every change in the country rock enclosing lodes often appears to have an influence on their ore contents, it may prove of importance as the field progresses to distinguish between these two classes of greenstone schists. At present, from a purely mining point of view, the important feature of the district seems to be that the sedimentary rocks of Archæan age have been intruded through in every direction by dykes and masses of greenstone, causing great disruption of the strata, and affording the conditions favorable for extensive hydrothermal action, the most probable source of ore deposits. The mines visited were — Red, White and Blue, Harbor View, Omaha, Mount Stennett, Elverdton Welcome Stranger, Elverdton South, Elverdton, Baden Powell, Mount Desmond, Mount Decker, Great Oversight, Carbine, Last Chance, Last Chance Proprietary, Emily Vale, Kilmore, Federal, Mt. Allan, Kingston, Mary, Mt. McMahon, Mt. Cattlin, Andante, Grimsby, Marion Martin, Sunset, Surprise, Marnoo, Mountain View, Zealandia, Australia, Mt. Garritty, Pick and Shovel. The total production from the field to the end of 1903 is 2992.72 tons, of the value of £25,865. The great question, says Mr. Montgomery, with respect to any newly discovered copper lodes is that of the permanence of the ore bodies in depth. It is now recognised that the surface portions of copper lodes are particularly liable to be much richer than the bulk of the lodes beneath, inasmuch as they have been enriched in course of time by copper being brought down in solution from the cappings of the veins, as these are gradually worn away by surface agencies. On his recent visit, Mr. Montgomery again inspected the mines, and stated that the work done since his first inspection had proved that the outcrops had good roots. He is satisfied that they will live down and maintain their values. As soon as the erection of the smelter is completed the progress of the field as a copper producer should be rapid. The production of ore at Mt. Malcolm for 1903 was 18,965 tons, valued at £45,567, and the grand total 33,391 tons, valued £128,203. The West Pilbarra production totals 12,340 tons, valued £112,778; Day Dawn, 15.65 tons, valued £167.

Coal Resources.

The development of a local coal supply is a matter of first moment to any State; but it becomes one of still greater importance to a State whose expansion is so rapid as that of Western Australia. To say that in this particular nature has not been as lavish in the bestowal of her favor as in the distribution of gold would, perhaps, be to jump at an over-hasty conclusion; but it can be safely said that if such be the case the development of the one has not kept pace with the production of the other. The coal fields of the State are comparatively few, but the area of the carboniferous deposit is pretty extensive. The principal scene of operations is at Collie, which is situated 25 miles due east of Bunbury, upon the Collie River. The existence of coal was known in 1884, but production on anything like a large scale was not commenced until 1898. The field covers an area of about 12 miles in length in a north-west and south-west direction, with a width of about 4 miles, the area altogether being about 50 square miles. As given by Mr. A. Gibb Maitland, F.G.S.,Government Geologist, the coal measures consist of a series of sandstones, conglomerates, shales and coal seams; but, owing to the peculiarities of the basin, the measures are, however, seldom visible at the surface, being covered by a more recent deposit derived from the weathering, in situ, of the beds beneath. This recent deposit is often cemented together by oxide of iron, forming what is locally designated as ferruginous conglomerate. The coal measures readily decompose into a sandy soil, which contributes in no small measure to the concealment of the underlying rocks. Any visible outcrops of the coal measures are found only along lines of most rapid erosion, and that is along the water courses. At several places in the bed of the Collie River, just below the water-level, in the vicinity of coal mining lease 110, are apparently horizontally bedded sandstones belonging to the coal measures. The latter have been deposited in a comparatively unsymmetrical shallow basin of erosion. Cases occur in which a portion of the seams has been eroded, and the channel so formed filled with deposits of sand. The strata do not appear to have been subjected to any serious disturbance, and to have suffered little or no lateral pressure. This field, so far, has been the only producer, and the output to 1903 is as follows:—

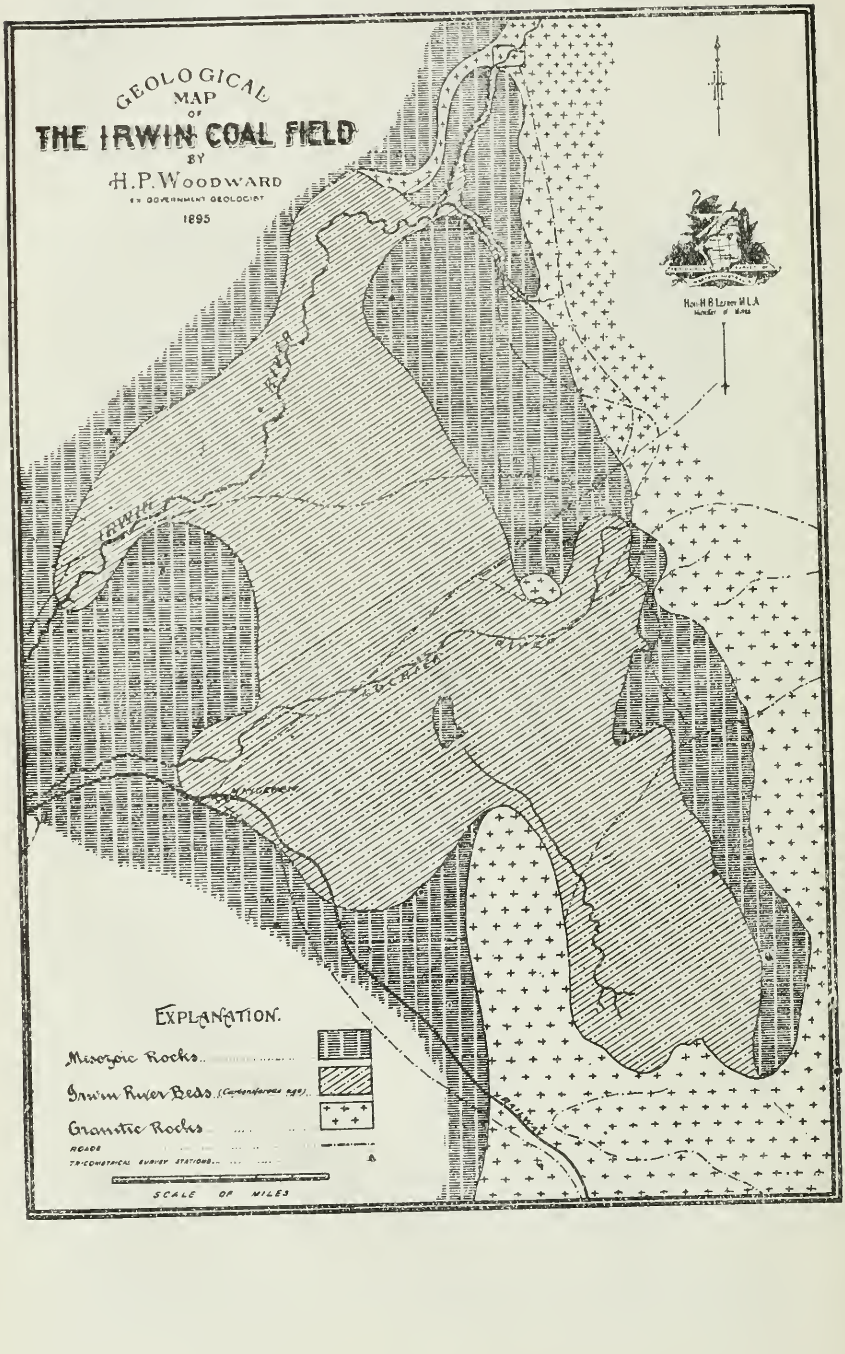
	Production.	Value.
	Tons.	£
1898...	3,508	1,761
1899...	54,336	25,951
1900...	118,410	54,835
1901...	117,836	68,561
1902...	140,884	86,188
1903...	133,426	69,128
Total...	568,400	£306,424

Most of this came from the Collie Proprietary mine. In this property there are five seams aggregating 63 feet 6 inches wide.

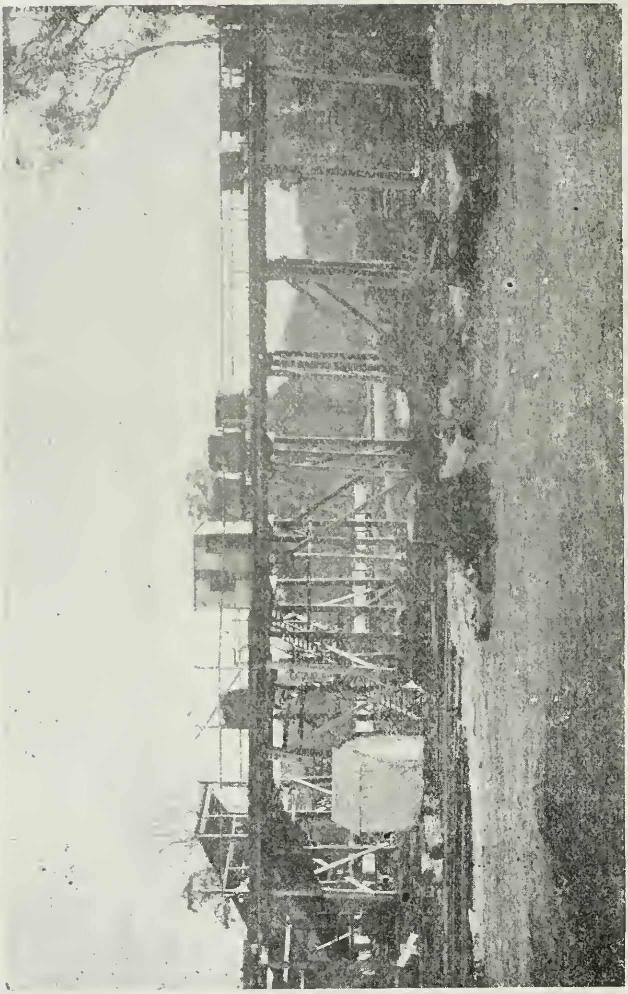

EXPLANATION
FERRUGINOUS CONGLOMERATE
COLLIE RIVER BEDS
GRANITE AND SCHISTOSE ROCKS
RAILWAY
GEOLOGICAL MAP OF
THE COLLIE COAL FIELD
BY
A. Gibb Maitland
Government Geologist
SCALE
OF CHAINS

The property consists of 5000 acres, each acre being calculated to produce 44,000 tons of coal, or a total of 220,000,000 tons The company has only been in existence since 1896, and the wages paid amount to nearly £70,000. As evidencing the extent of the underground workings, Mr. J. Evans, the manager, states that it is possible to walk a mile and a half due east through the mine, and for about 50 miles through it altogether. When he left New South Wales he said he was going to Western Australia to stop New South Wales from sending coal there, and he was going to do it. The mine has already been developed, and without further recoveries the company can take out 100 tons per day for the next 10 years. The coal contains a high percentage of moisture, however, and while it is not specially adapted for gas-producing purposes it has a high value for steam and domestic uses. A particular feature of the coal is that it is charged with gases which during the process of combustion retain their heat for an abnormal period of time. In connection with its use in water-tube boilers, and where the fire-bars are not too widely spaced, and a good draught, the indications go to show that, properly handled, the Collie coal will give excellent results. The Westralian Wallsend and the Collie Proprietary seams have been proved to be the same, and during 1902 the latter company acquired the property of the former. The railway of the Proprietary has been extended to the south-east of the Collie Boulder leases, where a considerable amount of preparatory work has been done. The line is also to be carried to the Collie Cardiff leases. At present the largest consumer of the fuel is the Railway Department. The gold mines are well-supplied with firewood. The mine-owners' attention is now being devoted to an extension of the coal market.

In the Irwin coalfield boring operations have been carried on near Mingenew on the Midland railway line. At a depth of 900 feet 10 inches, a seam of coal 6 feet 2 inches, resting on a seat of clay of 1 foot in thickness, was found. The cores were analysed by Mr. Simpson, Government Metallurgist, and gave the following: —Specific gravity, 1.410; calorific value, pounds steam evaporated, 12.38; British thermal units, 11.959; proximate composition—moisture, 2.66; volatile hydrocarbons, 29.68; fixed carbon, 52.92; ash, 14.74. The reports adds: "The greater part of the sample submitted is a clean hard, bright bituminous coal, with which are associated a few small duller pieces, which may perhaps represent 'stone' partings in the seam, and to which the comparatively high percentage of ash may be due. The coal is of the coking variety, yielding a dense, hard coke, of good quality. The percentage of moisture is low, being about the same as that of Newcastle coal. The ash, though high, does not clinker readily, the coal in this respect, as in many others, resembling the New South Wales southern district coal. In heating capacity the sample is superior to the best Collie coal, and about 16 per cent. better than the average of 23 samples from the Collie coalfield. In this respect it is about 10 per cent. less efficient than good Newcastle coal. This coal appears well suited for all ordinary purposes, coke-making, gas-making, and steam-raising, in either stationary, marine, or locomotive boilers" The Irwin coalfield is situated upon what is known as the Upper Irwin. The tract of country lies between 30 and 40 miles from the coast, and is extremely fertile.

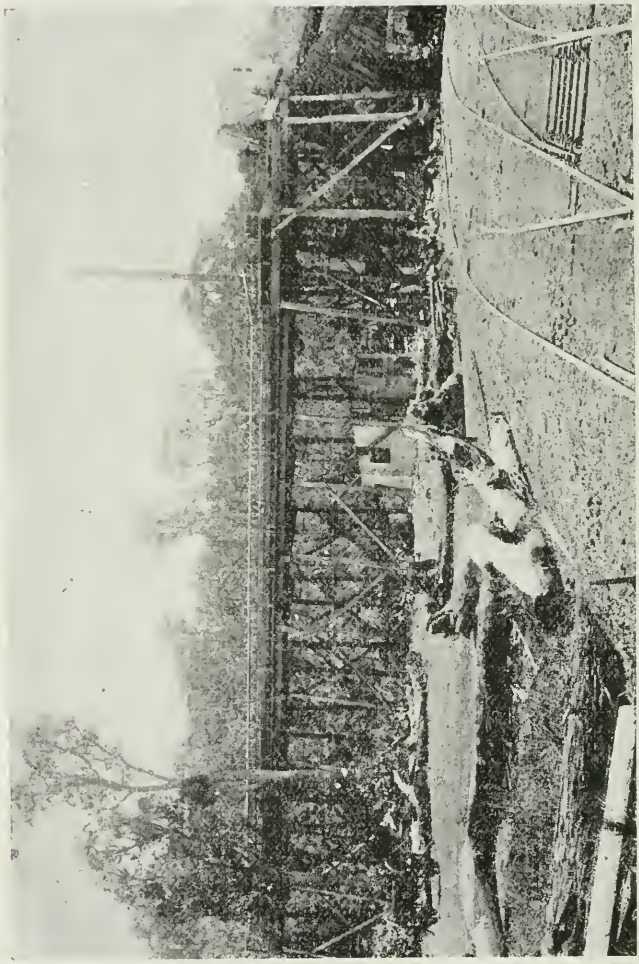

GEOLOGICAL MAP
OF
THE IRWIN COAL FIELD
BY
H.P. WOODWARD
1895
IRWIN
RIVER
EXPLANATION.
Mesozoic Rocks
Irwin River Beds
Granitic Rocks
ROADS
SCALE OF MILES
RAILWAY

The carboniferous basin spreads out to the eastward, covering a fan-shaped area, which is surrounded on most sides by cliffs of horizontally-bedded sandstone about 200 feet in height, which form the edge of the sandy tableland, and it is at the base of these beds where they rest directly, but generally unconformably upon the shales, that springs break out at several places. Indications of the existence of coal were first reported in 1846, but the field was never looked upon as a likely producer until 1888, when some fragments of coal were found on the north branch, which proved to be of very fair quality. These were traced up to their source and were found to be a seam of about 4 feet in thickness. Into this a drive 150 feet was put down to the dip, but although it improved both in size and quality it did not prove at the time of sufficient value to warrant the expenditure of more money upon its development. More seams were found, but there was no local demand, and the quality was not good enough for export. The carboniferous area spreads out from Mingenew in an easterly direction, covering an area of about 200 square miles, its greatest length from north to south, from Badgeree Pool, upon the north branch, to Mt. Scratch, being about 30 miles, while its greatest width, from Mingenew to Marandagry, upon the Lockier River, is about 17 miles. To the north-west this area is bounded by the high sandy tableland which extends away to the northward as far as the Greenough River. The south is bounded for the most part by the low outcrops of metamorphic rock, which contain many copper lodes; to the eastward by the bold escarpment of crystalline rocks, flanked by horizontally-bedded Tertiary sandstones, which often present towards the plains vertical cliff faces of as much as 200 feet, particularly where streams have cut deep channels through them; whilst to the westward it is bounded by more high sandy plains which extend as far as the coast. Of these boundaries that to the south and east may be taken as the definite edges of the carboniferous formation, but that to the north and west only as provisional, since the sandstones which form the high sand plain in these directions are of much more recent date, and may overlie extensions of the carboniferous formation, and since it is known that carboniferous rocks occur in the river valleys further to the northward it is highly probable that they are part of the same formation: and if this should prove to be the case valuable coal deposits may be found beneath the high sand plains which lie between the Irwin, Greenough, and Murchison Rivers. The coal measures consist of a series of shales, sandstones, and limestones, which are very rich in marine fossils. A good deal of experimental work has been carried out at the Vasse, in the neighborhood of the Vasse River, which enters Geographe Bay near Wannerup, some miles to the north of Cape Naturaliste. In all there have been six recorded bores, in the whole of which 25 coal seams have been reported. The greatest thickness of coal in any one bore was about 3 feet 6 inches. The coal at Fly Brook, the furthest branch of the south-east of the Donnelly River, which discharges itself into the Southern Ocean about 30 miles east of Cape Leeuwin, has been tested by a series of bore-holes to determine the size, quality and extent of the seams. One bore passed through 20 feet of coal in sinking to a depth of 128 feet, consisting of 17 seams, the largest



Collie Proprietary.

being 5 feet 4 inches, with a 6-inch clay parting, 2 feet 4 inches, with a 3 inch parting, and 2 feet 3 inches, with a 2 inch parting. The coal itself was of a highly lustrous variety, having almost the appearance of jet, but lacking its hardness; while the woody structure was clearly visible in some places. The average of three samples assayed was—water, 16.40; volatile matter, 38.23; fixed carbon, 43.32; ash, 1.85. The coal-bearing series consist of sandstone, grits, and clay beds (the latter of which are often micaceous), the whole being overlaid by a bed of ferruginous conglomerate, containing large water-worn pebbles of quartzite, quartz and other metamorphic rocks. According to the researches of Mr. H. P. Woodward, geologist, it is also highly probable that coal will be found in the northern portion of the Kimberley district, where the carboniferous series is largely developed in the quartzite and sandstone-capped flat-topped hills, with shale beds beneath attaining an elevation of as much as 1000 feet.



Westralian Wallsend Colliery.

Iron Ore Deposits

Iron ores are very widely distributed throughout the Western State, but with one or two exceptions the area in which the exploitation of such deposits is actively prosecuted is limited, such areas being at present confined to localities where fluxes can be obtained in considerable quantities. Some of the richest and most extensive deposits are absolutely valueless, owing to their geographical position. The iron ore deposits so far examined can be broadly separated into two main divisions—(a) ores associated with the crystalline schists and other allied rocks; and (b) the superficial deposits of limonite (laterite ore), which occupy extensive areas in many and widely separated portions of the State, and the soft porous deposits of hydrated oxide of iron (bog ore) of comparatively recent origin. The important ores associated with the crystalline schists are developed most extensively in the watershed of the Murchison River, more especially between 25deg. and 28deg. of south latitude and 116deg. and 119deg. east longitude. The most important localities are Horseshoe, Peak Hill, Mount Gould, and Mount No Name, Peak Hill, and Mount Hale, Wild Range (Wilgie Myah), Munara Hills, and Mount Narryer, Murchison. Less important deposits of this nature occur at Marble Bar, Pilbarra, Kilalo Well, Murchison, Wiluna, Mount Townsend, and Mount Marion, East Murchison; Bardoc, Broad Arrow; Mount Jackson, Yilgarn; and Jennapullin, Blackboy Hill, and Green Hills, Avon district. These deposits consist of highly inclined beds, bands and lenses of almost pure hematite (occasionally magnetite), or admixtures in all proportions of hematite and quartz, interbedded with, and sometimes replacing, quartzites and quartz schists. No detailed geological survey of any of these important deposits having yet been made, it is impossible to give even a rough approximate estimate of the minimum quantity of ore in sight in any one of them. That quantity of ore must be large, as is evidenced by the following descriptions:—The sigma-shaped range of hills on the west side of the Murchison, of which Mounts Hale, Taylor, Matthew, Yarrameedie and Eramandoo form the most prominent summits, is remarkably prolific in iron-bearing schists. The summit of Mount Hale is formed of contorted quartz schists, with bands of hematite, which occur in lenticular masses; some bands are often as thin as a sheet of paper, whilst others widen out to considerable dimensions. One band measured 70 feet across, and outcropped for over a quarter of a mile, but varied in thickness in different parts. There were similar bands parallel to it, and equally persistent along the strike. Just under the western summit of Mount Hale, the quartzite is replaced by a great bed of hematite, several huge monoliths of which stand out prominently on the range. This hematite can be followed along the range to a point just south of the summit of Mount Matthew. A partial analysis of a sample of this bed yielded, ferric oxide (Fe_2O_3), 94.05 per cent.; ferrous oxide (FeO), 0.97 per cent. The outcrop of a bed of ironstone forms a conspicuous feature on the surface at the foot of Mount Narryer

Range. The bed, which is vertical, attains a thickness of 8 or 9 feet, and rises about 2 feet above the ground. In the Wild Range, at the head of Roderick River, is the Wilgie Myah, said to be one of the richest iron lodes in the world. The deposit has been opened up to a depth of over 100 feet, and at the bottom of the excavation to a width of 50 yards. The deposit is a banded hematite. The following table shows the results of the assays of the iron-bearing schists, as made in the Departmental Laboratory:—

Description.	Metallic Iron. Per cent.	Silica. Per cent.	Water Hydro-scopic. Per cent.	Water Com-bined. Per cent.
Banded limonite	50.33	7.30	.38	[illegible]
Massive hematite	63.7	?	?	?
Massive hematite	65.62	9.33	4.82	—
Massive hematite	61.91	1.13	.13	3.97
Hard red Wilgi	34.17	21.93	1.11	11.51
Argillaceous limonite ...	35.5	?	?	?
Hematite quartz schist ...	57.45	9.33	4.82	—
Filirous limonite	60.20	1.62	.39	14.79
Siliceous hematite... ...	55.5	?	?	?
Massive hematite	66.55	.91	—	—

Deposits of very pure magnetite have been found in the ferruginous dyke-rocks of the Darling Ranges, at Serpentine, and Collie Rivers. Similar deposits exist in the neighborhood of Ravensthorpe, Phillips River, but their extent is at present unknown. A recent analysis of the magetite ore from a locality 12 miles north of Collie yielded at the hands of the assayer—metallic iron, 52.87 per cent.; silica (SiO_2), 45 per cent.; titanium dioxide (TiO_2), 14.13 per cent. The superficial deposits comprise the laterite ores, and the bog iron ores. The laterite ores, together with the gravel resulting from their denudation, are the most widely distributed ores in the State. The ore of this class has been principally used for fluxing purposes. The bog iron ores consist of soft, porous deposits of hydrated oxide of iron, but up to the present deposits of this class have not been exploited. No coal suitable for smelting has yet been found in the State. The production of iron ore in the State has been as follows:—

Year.	Ore raised. Tons.	Value. £.	
Previous to 1899	100	300	Used as a Flux.
1899	12,852	8,939	
1900	12,251	9,258	
1901	20,569	13,246	
1902	4,800	2,040	
1903	220	88	
	50,692	33,871	

Ploughing Tin Wash, Pilbarra.

Graphite or Plumbago.

Graphite or plumbago is usually found in massive forms, which may be separated easily into leaves or plates, and hence are said to be foliated; sometimes also it is finely granular and compact. It has the same composition as the diamond, consisting also of nearly pure carbon; it is, however, a different substance in its physical characters, and is hence a distinct mineral. They differ in crystalline form; also the diamond is hard and heavy, while graphite is soft and light. Graphite is commonly found in the crystalline rocks called gneiss, sometimes scattered in scales, but occasionally in large beds that can be mined. It is also found in scales in crystalline limestone, and is often treated in an iron furnace. It is the so-called black lead, says Dana, of the lead pencils (but it is only like lead in its color), and would be mined for this purpose, if for no other. It is used as an excellent lubricator; because of its smooth, soapy character when pulverised; also, mixed with clay, for making crucibles, because it is infusible, and not affected by the heat of an ordinary furnace; in electro-plating, because it is a conductor of electricity. In the Champion Bay district of Western Australia graphite has been found in association with certain ferruginous deposits, but the large percentage of iron in it prevented it of being of any marketable value. Some years ago a deposit was worked in the neighborhood of Kendenup. The graphite was of fair quality, but the distance from market was a bar to its economic working. Some years ago a deposit was found near the head of the Donnelly River, and it was proposed to work it, but the low price of the mineral compelled the parties to abandon operations. In 1894, a find was made near the older field. The first of the outcrops was 28 feet in thickness, being followed by 13 feet of schistose rock, containing a small bed 1 foot 6 inches in thickness, whilst the third bed was 8 feet in thickness. Several shafts were sunk, one being on a large bed, a sample, weighing 25cwt., being sent to England in order to ascertain its commercial value. In another shaft, about 15 chains, the deposit was again struck at a few feet from the surface. Mr. H. P. Woodward, geologist, in a description of these beds, states that they should be called plumbaginous schists, since the percentage of graphite contained is so small, the main portion of the deposit consisting of magneisum silicate. The formation consists principally of micaceous and talcose schists, which here strike east and west, dipping at a high angle to the northwards, whilst following along to the southward, close to the outcrop of the graphite beds, is a large dyke of intrusive granite. A little to the eastward of the drive, the outcrop of this deposit is lost, but beds of steatite are met with along this line as far as Wilgarup; therefore, the graphite seams will also probably be found to extend in this direction, the local break in the continuity of the rocks being due in all probability to a fault. To the westward, the graphite can be traced for several miles, but the beds seem to split up, and become smaller upon the claims that were first prospected. This deposit of earthy graphite is due to the alteration of poor shaley coal seams, the metamorphosis being in all probability due to the indurated granite to the southward, which changed the coal seams into graphite, and the shale into schists. It offers exceptional features for cheap working, since the spur upon which it is situated rises so rapidly that a drive following the strike from the outcrop in the creek would have 100 feet of backs in a distance of 20 chains; whilst if crosscuts were driven, the seam would be obtained.

Mica.

The development of the mica formations has not yet been persevered with to any extent. The mica-producing strata are the crystalline schists and allied rocks. Generally it is found that the mica-producing rocks are pegmatitic granites, which traverse the crystalline schists, etc., either in the form of dykes, sheets, or lenticular masses, which are often parallel to the foliation of the surrounding strata. Possible commercial mica is known to occur in the following places:—Nokenena Brook, Northampton; Tambourah, Pilbarra; Mullalyup, Darling Ranges; Bindoon; Londonderry, Coolgardie. The mica at the Darling Ranges occurs in granite (? pegmatite), and dykes, which do not go down vertically. Near the surface they are, as a rule, much decomposed, the mica being valueless; but in or two places hard masses outcrop, where the mica is of good quality. At the Mica Mine, Londonderry, work was carried on by means of an open cut, along the outcrop of a coarse granite dyke, which intersects the surrounding hornblendic rock. The granite, which at times assumes a pegmatite structure, is composed of large masses (in some cases weighing as much as 1cwt. or more) of orthoclase quartz, lepidolite, and cyanite. Muscovite is developed on a small scale, and is generally well crystallised. The only other mineral visible is chalcedony, which is found filling the original holes in the rocks. The most important, from an economic point of view, of the constituents of the granite, is the lithia mica (lepidolite), which occurs generally in rough radiating bunches, although it occasionally appears as somewhat well-defined crystals. The greatest size in which the mica is found is 15 inches by 12 inches, but this is exceptional, the average not exceeding 5 inches to 6 inches. The mineral, when not less than about 1-32nd of an inch in thickness, gives a distinct sherry-red color when examined by transmitted light, but, in sheets split finer than this, it is difficult to detect coloration. Besides these large sheets, the mica also occurs in long crystals, which, when grouped together, as they frequently are, with the longer axes parallel, present a peculiar scale-like impression. The color of such specimens varies from a pale pink to a pale green, or is quite colorless. The cleavage of all the varieties is very perfect.

TASMANIA.

TASMANIAN MINING AND METALLURGY.

Introductory.

In beginning this series of articles I desire to acknowledge the assistance and facilities given by the Hon. E. Mulcahy, Minister for Mines, and Mr. W. H. Wallace, Secretary for Mines, and to acknowledge the valuable information supplied in the publications of Messrs. Twelvetrees, Government Geologist: Waller, Assistant Government Geologist; Montgomery, Morton, and Johnson. Other references will be mentioned in due course.

Tasmania is our smallest State, having an area of only about 26,000 miles, yet after visiting it one only regrets that some of the large waste States could not be compressed to the same area without any corresponding contraction of their mineral wealth. From a geological point of view it is unlike the rest of Australia. Nearly one-third of the island, or a strip parallel to the West Coast, is made up of crystalline and metamorphic schists, clay-slates, conglomerates and quartzites of Archaen, Cambrian, Silurian and Devonian age. These are penetrated at a few places by granites and porphyries, but for the most part there are narrow belts or strips of country which are persistent in direction over considerable distances. Each of those belts appears to have its own particular class of minerals, so that in passing across such zones a variety of mineral substances is met with unequalled in Australia. For instance, gold, iridosmine, tin, iron, antimony, silver, lead, zinc, chrome, bismuth, copper ores, as well as complex and rare compounds of these valuable metals are actually mined within a short distance of Zeehan. This strip of country includes the Blythe River iron deposits, Mount Bischoff, the Zeehan field, the Heemskirk district, Rosebery, Mount Read, Dundas, Mounts Lyell, Jukes, and Darwin. Fully one-half of it lying to the south of the fields mentioned does not seem to have been prospected. Rocks of the same description crop out at Beaconsfield and on the North-East Coast: in these the main gold mines of the State lie. The granites and porphyries, which have burst through these rocks, are for the most part stanniferous.

The next formation in size consists of greenstone; the whole of the central tableland, the banks of the Tamar to Launceston, and large portion of the South-Eastern Coast consists of this rock, which does not appear to be metalliferous. It is most interesting to note that in almost every instance a fringe of coal-bearing rock surrounds the greenstone. At Ben Lomond, one of the most prominent peaks in the north-east, the highest portion of the mount consists of greenstone, while it is surrounded by the palaeozoic or mesozoic beds, similarly many isolated peaks near it. Whether these greenstones have burst through the coal beds or whether they overlie them does not seem to have been definitely settled. During the corresponding period of last year my time was spent amongst the mines of Western Australia. The contrast in conditions was most striking. Instead of the hot, quivering air, the mirrage, the dust storms, the undulating waterless country, the bare stemmed scrub, and the scanty foilage, here was a land where rain falls two days out of three, where the mists almost always envelop the hills, and where vegetation is so dense overhead that the soil never sees the sun. While the goldfield pioneers of Western Australia had to face terrible hardships, so the pioneer prospectors of the West Coast had conditions pitted against them of the opposite kind. The country is so mountainous that even the familiar racecourse is missing in some populous centres. In some places the mines are 2000 feet above the valley below, and it is a matter of danger and difficulty to get to them. With the exception of a few plains on the basaltic tableland near Mount Bischoff, the bold looking hills of serpentine and the button grass country near the sea coast, the field a few years ago was an almost impenetrable jungle. There was no edible grasses for stock, it was impossible to make a road in the low country without corduroying almost every yard of it, while the mountain paths made would only serve for pack horses.

The usual Australian vegetation has been replaced by the beech (locally called myrtle), the ashen grey leaved flowering leatherwood, the tall sassafras and lightwoods, whose foilage form an impenetrable barrier to the sun's rays; underneath these are the tree ferns, musk, hazel, blanketwood, and the terrible horizontal. The last grows with slender stem and a heavy top, and after a time bends over into the position from which it takes its name. This, together with the fallen vegetation, forms such a tangled barrier that every step has to be cleared by the axe. In this awful and solitary forest there is no animal or bird life except perhaps an occasional wombat and a few screeching parrots. What little chance a prospector has of finding lodes, except in a few exposed places or right up on the hill tops. In all other places a mass of decayed vegetation covers a peaty soil to the extent of several feet. To add to his difficulties there is a rainfall of from 90 to 120 inches per annum. A spell of fine weather, even for a fortnight, is an exception, and so little provision is made by householders for storing water that three weeks dry weather means a drought. The rain falls in alternate drizzles and showers, sometimes for weeks at a time, and those who have had experience of camp life in a forest will know what that means. Every article becomes sodden and damp, and it needs an iron constitution to combat such a climate when wholly exposed to it. It is worthy of note that so far as typhoid fever and such complaints are concerned that it is only during the dry weather that they are prevalent. As may be readily understood the pathways up the mountains become cataracts, while the creeks and rivers are nearly always in flood, and discharge large volumes of peaty amber-colored water down boulder-strewn gorges, to deep placid streams below. Many of these rivers are unfordable for most of the year, while prospecting for gold near their beds is a difficult and dangerous undertaking. Bullock teams, so well-known in other such mountainous districts, are here unknown, while even pack horses are out of the question on most fields. As a rule, the men carry their swags, and they are of such a weight as to bring back the saying that there were giants in those days.

A change has come over the scene since the days only a decade ago. The Government recognised the value of the country's mines. Private enterprise has also done a great deal. The railway line from Emu Bay to Waratah and thence to Zeehan has not come up to original expectations so far as profits to investors are concerned: but there is little doubt that in course of time there will be settlement along it from Burnie towards Waratah, and that beyond that point many still undiscovered mines will be opened up. It is a matter of great surprise to me that tourists do not see more of the West Coast, which is undoubtedly the most attractive part of Tasmania. It was recognised by the State that it was as easy to build a railway (or tramway as some of these small gauge lines are named) as a roadway. And from Zeehan some of the most instructive lines have been built. One, the N.E. Dundas tram, has a 2 feet gauge, and traverses country where it would not be possible to take a loaded bullock team. The engines are heavy—said to be 15 tons—and the curves are only two chains radius. It winds about the hills, at the same time rising 976 feet in 11 miles, then gradually descends to its terminus, Williamsford, some 18 miles distant. This line takes the ore from the Hercules, Mount Reid, Fahl Ore and other mines on its course. It sweeps round the base of several waterfalls several hundred feet in height, amongst others the well-known Montezuma falls. No visitor should miss this most instructive trip.

Another line, built by the State, is known as the Comstock. This runs out towards Trial Harbor, and serves to bring in ore from many of the surrounding mines, also fuel. A private railway from Zeehan to Maestries takes the ore from the Dundas mines. In addition to these, the Western mine had a small steam tram line of its own, while there is a steam tram service through the town of Zeehan itself. In fact, steam locomotion seems to be so natural to the place that outside roads are unknown. With the progress of mining a vast amount of timber was destroyed, and the sun was able to shine on undergrowth which had been sheltered for centuries. A few hot days will make the material as dry as tinder, for it is surprising to note how rapidly the ground dries even in a few hours after the heaviest rain. A bush fire is started and sweeps everything before it until it reaches the unaltered forest. The acid fumes from Mount Lyell have destroyed vegetation for miles around. This material also dries in a few hot days, and a bush fire fiercely burns every particle of timber over the surface; it also runs along the blackened peaty soil itself, burning down to the damper material underneath. This action may take place until the organic matter in the soil is burnt out. The local opinion is that these destructive fires are due to the sulphur falling on the ground from the smelters, and subsequently burning; but since the sulphur is wholly or almost wholly burnt on leaving the furnaces, it cannot be burnt the second time. By examining the surface soil in the neighborhood and applying a very simple test it could readily be seen whether free sulphur did exist, as so emphatically stated.

The combined action of clearing and fires will enable a great deal more ground to be prospected than could have been done otherwise. Also in those places where acid fumes do not affect the soil a sweet grass springs up, so that stock thrive where before they would have starved.

The prospector on the West Coast has not the easy proposition that the gold-seeker has. In the majority of cases the latter is able to tell by simple inspection whether his ore is payable or not. The former has to have all material carefully assayed, and it takes him a long time and usually costs a good sum before he can say from mere inspection, even approximately, what his ore will go in silver, antimony, zinc, or whatever metal he wants or does not want. When he does find out he has to sell his product (for he cannot go and get it crushed, and the metal extracted for a small sum like the gold-miner), but in this respect he is better off than in many places, for there are excellently worked local smelters and also ore buyers from other parts.

The State has also gone further, and erected a neat School of Mines building at Zeehan, which is fairly equipped. This should prove a great boon to all those wanting to improve their mining and metallurgical knowledge on such a field so well supplied with complex ores. The building, which was formally opened on February 6th, is central, and situated on the site of the old post-office. It is built of brick, with stone facings, and, although only one storey high, the walls have been specially designed to carry another storey if required. The front room, 23ft. 6in. x 18ft. on the left of the entrance, is set apart for a mineralogical museum, and is equipped with specimens of the various rocks and ores from the West Coast and other centres. Opposite the museum is the surveying lecture room (23 feet 6 inches x 16 feet). and behind this the chemical lecture room (24 feet x 23 feet 6 inches). Other rooms in the building are the mining lecture room (23 feet 6 inches x 22 feet), chemical laboratory (32 feet 6 inches x 27 feet), furnace laboratory (24 feet x 15 feet), instructors' private laboratory (16 feet x 15 feet), sampling room (15 feet x 14 feet), balance room (14 feet x 8 feet), and registrar's office (17 feet x 8 feet). Several of these latter rooms are in the old post and telegraph building, which has been utilised by the architect to form the rear part of the new institution. All the rooms are well equipped and lighted. The artificial illuminant employed for night use is supplied by the Zeehan Electric Light Co. It is to be hoped that this school will be allowed to develop on natural lines. In Hobart or in Launceston, or in any such centre the subjects necessary for a diploma in mining or in metallurgy—such as mathematics, surface surveying, assaying, chemistry, and mineralogy—can be taught with as much and often to greater advantage than in a mining town, but experimental plants should be erected near mining centres, and the advanced students from the other schools should be required to devote some time to this class of work under competent direction. In this way the Zeehan School of Mines would not only achieve a reputation for itself, but would repay the State a hundredfold by a single discovery which would enable low grade copper or other ores, of which there are vast supplies, to be worked profitably.

Mount Lyell.

The Heemskirk tin field attracted many prospectors. From this point as a base, they pushed out along the west coast. In 1882 Mr. Con Lynch, using Macquarie Harbor as a supply depot, went up the King River for 6 or 7 miles. Gold was discovered in a tributary now called Lynch's Creek. A rush took place, and one of the results was that William and Michael McDonough, nicknamed the Cooney Brothers, and J. S. Karlsen pegged off and became possessors of the Mt. Lyell Mine. It was simply pegged out because a few prospects of fine alluvial gold were obtained: the iron blow standing out so prominently attracted their attention, and this was included in the area pegged out. The Cooney brothers sold one share in 1885 to Mr. James Crotty, and made arrangements with Mr. Wm. Dixon to work the other share. Further prospecting revealed payable gold. Mr. Crotty applied for and obtained a reward claim of 45 acres. The gold was won by sluicing and concentration in sluice boxes and blankets. Other partners, including Chas. Karlsen, Allen Karlsen, and F. O. Henry, were taken in. Owing to countless difficulties, no real progress was made, and the Karlsen brothers sold out. The iron blow was found to be rich in gold, a company was formed, and a 10-head battery erected. The crudest treatment was given, and yet 1874 tons gave 1672oz. gold and 852oz. silver, the bullion being valued at £6159. It is extremely probable that this only represented about one-third of the value of the ore. The opening up of this iron blow disclosed the presence of a large body of pyrites, and also scattered over the gossany surface plentiful supplies of native copper. The iron outcrop was opened up for some considerable distance, and all attention devoted to winning the gold from the great masses of blueblack ironstone standing on the surface.

In 1891 Mr. Bowes Kelly and party visited the mine, and sent a bulk parcel of the pyrites to Mr. H. H. Schlapp, the metallurgist of the Broken Hill Proprietary Company; results were so promising that this influential group of mining men became shareholders in the Mt. Lyell Mining Company, No Liability, registered in 1892. Mainly through the advice of Mr. Schlapp, Dr. Peters, the well-known writer on copper smelting, was engaged to report on the mine early in 1893. Dr. Peters started his investigations, and in May, 1893, his report was completed. It is worth noting that in April the company was reconstructed, and became a limited liability company of 300,000 shares at £3 each, 150,000 shares being given to the old shareholders, and 150,000 reserved for future issue.

It does not appear, until Mr. Schlapp took the matter in hand, that the treatment of the pyrites for copper was ever seriously considered. In view of subsequent developments, this may seem surprising, but the west coast differs from most other copper districts in Australia in having an abnormal rainfall; in consequence of this, any copper pyrites which become oxidised is either at once washed away in solution into the streams below, or, sinking into the earth, become reduced to native copper. There is no show of copper stained rocks such as one may see in South Australia, New South Wales, or Queensland; no carbonate or oxide deposits of any magnitude, but simply an uninviting ironstone outcrop or hard, massive pyrites only showing occasional signs of copper. The pyrites deposit at Lyell was large, but the copper contents small; poorer, in fact, than anything of a similar nature which had been successfully worked in Australia. After a lapse of ten years, the main contentions of Dr. Peters have been amply verified. His report deals with the whole problem in a way previously unknown in our mining world. Most of our Australian reports were couched in terms of sanguine expectations, a smattering of scientific terms, and an utter absence of detailed investigation. The expert was supposed to take a Roderic Dhu stand, and evolve from his inner consciousness a forecast or an amazing glimpse into the future, based on what he called his practical experience. Fortunately, the gentlemen connected with this mine did not need glowing reports, based upon a minimum of fact; but proceeded to open up this mine and test it thoroughly before dealing with the question of treatment. Tunnels were driven round the pyritic mass, crosscuts were put in, and the whole sampled systematically. The average value of the ore was determined, the costs of mining and treatment gone into exhaustively, and a margin of profit shown. While due allowance was made in the estimates for deductions as to quantity of ore, insufficient allowances were made for the diminution in quality, and this is the main fault to be found with the report. The work to be done, as generally outlined, has been carried out with even greater success than promised, and while we are apt to look back and to take the work done as natural after such a report, yet the main credit lies with those who have been able to obtain better results than those promised. In spite of the excellent report furnished, it is doubtful if a serious check would not have been experienced had not a bonanza of rich ore been struck in exploratory work in No. 4 level. This rich patch of ore returned £105,000 nett from 850 tons. Money was thus furnished for progressive work, the plant was soon started, and success assured from the outset.

The Mt. Lyell Mine is situated on a peak which is much lower than many on the ridge which claims the name. Its height above sea-level is about 1500 feet. On one flank of the ridge are massive conglomerates which make up the series of mountains of which Mt. Owen is the most prominent; the other side of the ridge or flank consists of schists. Between these two lies the pyritic body now being worked. On the surface this was oxidised, but at a short distance down solid pyrites were struck. A horizontal section of this showed it to be lenticular in shape, or wide at the centre and tapering off at each end, its greatest width being about 300 feet and length from 500 to 600 feet. The general strike is N.W. and underlie to the S.W. The southern end dips under the hill, necessitating a greater amount of stripping of overburden as the depth increases. The angle of dip on the surface is about 60 degrees, but becomes steeper at a depth. The footwall is conglomerate and the hanging wall schist Following the foot wall round and lying between it and the pyritic body is a body of the blueblack hematite, of the same character as formed the outcrop.

Mount Lyell M. and R. Co.'s Mine, Mount Lyell.

It is held to be a secondary product derived from the pyrites, but is remarkable in that it persists to the deepest levels worked—some 800 feet below the surface. There is evidence of metamorphic action near this body, for the conglomerates have been more or less changed into quartzites, and what is more remarkable, the hematite appears to have replaced the siliceous conglomerates. When near the pyrites, the hematite is rich in gold, but the values die away towards the wall. While conglomerates form the wall, there is scarcely a place where they are not separated from the ore body by a band of schist of varying thickness. Leaving the ironstone and crossing the pyrites, the appearance of the latter is unique, for it is almost pure. The color is white, with a yellowish tinge; near the footwall occur splashes and veinlets of fahl ore, galena, blende and chalcopyrite becoming less plentiful, and giving out and passing into almost pure pyrite as the hanging wall is reached. The amount of gangue present is exceedingly small and consists only of barium sulphate, and siliceous material. Typical analyses of the ore gave—Iron (Fe), 40.30; Silica (SiO_2), 4.42; Barium Sulphate ($BaSO_4$), 2.50; Copper (Cu), 2.35; Alumina (Al_2O_3), 2.04; Sulphur (S), 48.50. In other words, the composition is approximately 83 parts iron pyrites to 7 parts of copper pyrites. The great bulk of the ore is even poorer than this, but the analysis given represents the average of ore treated for six months.

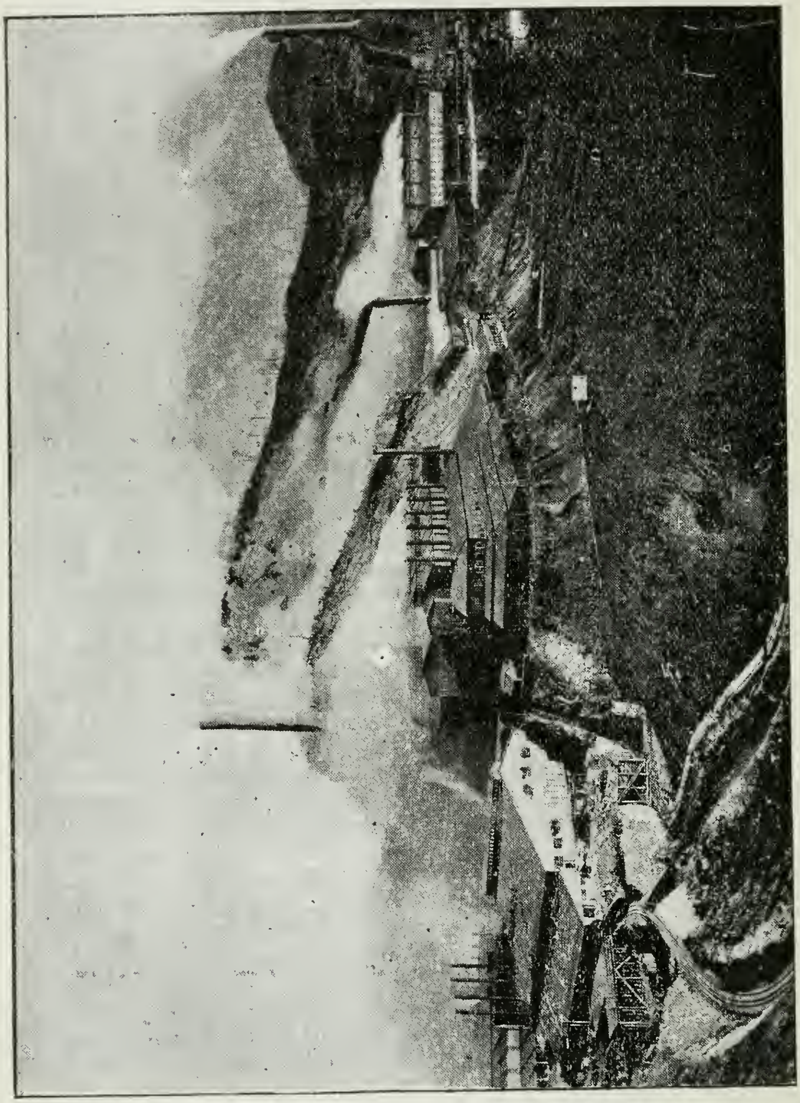
Horizontal sections below the surface show that the ore body maintains the general shape it had above. At the No. 4 level, the total length is 660 feet and the greatest width 270 feet; at No. 5 level, 510 by 210 feet; at No. 6 level, 510 feet by 270 feet; at No. 7 and No. 8 level, both length and width have considerably increased. A highly instructive model of the mass, showing detailed work, has been prepared by Mr. Batchelor, the mine manager.

The main body of pyrites continues down to No. 8 level, but no rich patches similar to those obtained in the upper levels have been struck up to the present. Near the contact, however, at this level, good ore has been got, but limited in amount for a mine of this size. It is possible that deeper sinking may show a higher grade ore. Curiously enough, in all the pyrites, whether rich or poor in copper, the gold contents are maintained; there is also but little difference in the silver values.

The origin of this deposit has not been satisfactorily determined. The opinion held by geologists some time ago that these masses were due to the filling of peaty swamps and marshes with surface solutions carrying sulphate of iron, which was reduced to pyrites, and subsequently covered over, does not appear to be correct, although Dr. Peters put this view forward. It would be an extraordinary coincidence if such masses occurred at or near the contacts of two formations, as they do here between the conglomerates and schist, or at Rio Tinto between porphyry and slate. This theory also necessitates fresh water deposits and rocks, but the rocks adjoining Mt. Lyell are fossiliferous, which shows them to be of marine origin. Nor is there any sign of fossil vegetation either amongst the pyrites themselves or in the adjoining rock. This of course would have been destroyed by subsequent metamorphic action, which has undoubtedly gone on. The tongues of schist running into the pyrites shows that the latter were not bedded contemporaneously with the schists. This pyrites lode only differs from many others in size and uniformity of contents, and any explanation which will account for the former series might with special conditions apply to this. At the same time, it would be as well to consider the possibility of such masses being intrusive. Many dykes come up heavily charged with pyrites, and there seems to be no reason why such a mass may not have been forced upwards along a plane of weakness, not necessarily by a dry heat, but associated with solvents which have disappeared, or have become part of the adjacent rocks.

The method of working the deposit is probably the cheapest that could be devised. This I believe was initiated by Mr. Lindesay C. Clark, and is now carried out on the same lines by his successor, Mr. Batchelor. An excavation was made near the centre of the mass required to be shifted. the material shifted was sent down a pass to a tunnel below and trucked out. As soon as the first excavation was extended far enough laterally a second one was commenced, and worked on the same lines as the upper one, the first in the meantime being extended as far as required. A third followed the second, and so on for as many as were required. This system has gone on at Mt. Lyell for some years, so that there are now nine of these to a depth of 266 feet, or about 30 feet apart. The topmost bench is about 800 feet by 600 feet while the lowest level was about 100 feet in diameter. This system, as carried out, gives a series of tiers or benches one above the other. The ore from each bench is trucked round and run out through tunnels to the ore bins on the other side of the ridge. The barren material or overburden—and this increases at a great rate, owing to the dip and pitch of the pyritic mass into the hill—is trucked round and sent over the dump. The benches are numbered 0, 0A, 1, 2, 3, 3A, 3B, and 4; the tunnels or levels are at 1, 2, 3, and 4. Of these benches, Nos. 0, 0A, 1, 2, and 2A are in overburden; Nos. 3, 3A, 3B, and 4 are in pyrites. On the footwall side and for more than half the width of the deposit, the whole of the ore is removed and sent to the smelters; on the hanging wall side a vast body of the lower grade ore has been left standing.

On each terrace, from No. 1 downwards, may be seen men moving like ants, picking down loose material, smashing up blocks of ore or stone, drilling pop-holes in blocks too large to break, and filling trucks by hand or with travelling steam cranes, while the incessant clatter of 20 or 30 rock drills gives reality to the panoramic show. At about 4.30 p.m. drill after drill goes out of action, and the whole scene alters; holes are charged and tamped, trucks are moved round, gangs of men lift the rails and methodically stack them in safe places, tools are similarly piled, and the stream of workers depart. Red flags are hoisted, bells rung to warn incautious passers-by—the whole air becomes strangely subdued, and all have gone except a few figures on each bench. Presently, at a given sign, each of these, with a red hot lump of iron, runs round and ignites each fuse. The shortest fuse burns for 1½ minutes, but long before this the firers have got back to a place of safety. A report breaks the silence, followed by a rapid succession of exploding shots; then comes the deeper sound from the heavy charges, and the air is thick with flying rocks and stones, which sometimes are projected for hundreds of yards. When all is over, as much as 1000 tons of rock may be dislodged, but the effect appears to be puny compared with the size of the mine. The material is broken and removed to the extent of over 1000 tons per day.



Mount Lyell M. and R. Co.'s Smelters.

Underground the work done consisted of driving around the ore body, and also carrying drives along the contact of the schist and conglomerate in order to locate any fresh deposits. At the No. 8 level, or 730 feet from the surface, and also, to a lesser extent, from other underground levels, patches of ore are removed from the foot wall; these are sent down for smelting, with the pyrites, in the proportion of about 1 to 9. The analysis of this class of ore is given as—Iron (Fe), 24.75; Silica (SiO_2), 30.69; Barite ($BaSO_4$), 1.48; Copper (Cu), 5.33; Alumina (Al_2O_3), 6.30; Sulphur (S), 30.00. The mine itself is nearly dry, in fact, from the tight dense mass of pyrites one would not expect much water to pass through. The small amount which does make it exceedingly acid and rich in copper; it is stated that the drip from the roof will eat through an iron plate half an inch thick in less than a fortnight. The drainage water from the lowest adit carries 23 grains of copper and 200 grains of sulphuric acid per gallon. The copper is precipitated on scrap iron, and the precipitate, containing 50 per cent of copper, sent to the converter plant. The costs of mining ore and overburden are kept separate, but in the latter case the charge was calculated over a number of years and averaged:—

	Mining. s. d.	Overburden Removal. s. d.
In 1897	1 8.27	2 0
1898	1 4.88	2 0
1899	2 5.31	2 0
1900	3 5.00	2 0
1901	2 6.86	2 1
1902	2 3.89	2 1

In addition to the big mine, the company has acquired the Royal Tharsis, South Tharsis, King Lyell, Glen Lyell, and Mt. Lyell Reserve leases. The first two supply a schist which carries 1.64 per cent. of copper, with small quantities of silver and gold. This low grade ore is known as metal bearing flux. Upwards of 32,000 tons of material have been smelted, the average assay being —South Tharsis, 62.67 silica, 8.05 iron, 12.17 alumina; Royal Tharsis, 61.17 silica, 9.94 iron, 10.44 alumina.

Mount Lyell Metallurgic Methods.

The ordinary term pyritic smelting is rather vague, for it may mean, as defined by Dr. Percy, "the smelting of silver or gold ores, which are either free from lead or do not contain it in sufficient quantity to collect the silver, in conjunction with pyrites, in order to produce regulus, in which the silver may be concentrated. Iron pyrites is used for this purpose, and by preference, such as is argentiferous or auriferous; but, failing this sulphide, cupriferous iron pyrites or copper pyrites may be substituted." On the other hand, it may mean a process in which part of the heat is supplied by the burning of pyrites, the balance being supplied by coke; but the true meaning should be the smelting and concentration of pyrites and matte with the formation of a siliceous slag by the aid of the heat derived from pyrites alone. Carpenter, an authority on the subject, writing a summary of pyritic smelting for 1901, Mineral Industry, Vol. XII., indicates that where pyrites are added as carriers of precious metals, the process is a success; but when the pyrites are added for other purposes, i.e., for fuel or a flux, the pyrites are liable to melt and run away from the charge, leaving the siliceous contents unfluxed, which results in a frozen furnace. This, he writes, was probably the difficulty where the smelting of pyritic ores with little or no carbonaceous fuel has been attempted. He also states that the process should not be attempted without hot air.

Dr. Peters, with his American and European experience behind him, recommended roasting in stalls, smelting of the roasted ore in blast furnaces, and bessemerising the matte, but provisionally indicated that pyritic smelting might be successful. It need hardly be said that Dr. Peters' method meant roasting the ore to eliminate portion of the sulphur, and transforming part of the iron to an oxide. The semi-desulphurised material would then be fed into a blast furnace with silica and limestone. The silica, iron oxide and lime would form a slag and be got rid of, while the sulphur remaining would unite with copper and iron to form a matte. A large amount of carbonaceous fuel would have to be used, some, perhaps, in roasting, and at least 15 per cent. in smelting the charge. The matte obtained in the first operation would probably need enriching to fit it for bessemerising, so that the roasting process to eliminate some of the sulphur would have to be gone through again, followed by the subsequent fluxing and smelting. The action as so carried out of oxidising the pyrites outside a blast furnace, and then supplying heat enough from burning fuel to partly reduce what was oxidised, and also to cause the slag-making materials to unite and to be raised high above their melting points is wasteful. The dream of the metallurgist has been to make use of the heat caused by the burning of the sulphur in the pyrites, and the oxidation of the iron to perform roasting and smelting in one operation. In 1856 Bessemer applied for his patent, which subsequently revolutionised the steel industry. The simplicity of the method and the speed with which a very impure iron was converted into a marketable product, caused the copper metallurgist to turn attention to it. Eight years later, the method was tried on copper mattes, but was a failure, because in the case of iron there was no mechanical separation of the pure steel from the impure iron, but in the case of copper matte metallic copper separated, and the heat required by the burning of the material could not be kept up, consequently the copper chilled in the tuyeres or blow holes.

The first who really left his mark indelibly inscribed on this branch of copper metallurgy was Mr. John Holloway, and the results of his experiments were published in 1879. Holloway experimented with Rio Tinto ore, similar in composition to that at Mount Lyell, and carrying from $2\frac{1}{2}$ to $3\frac{1}{2}$ per cent. of copper. The crude ore was first melted in a cupola, and then transferred by a ladle into a converter. The ganister, or siliceous lining formed the acid of the slag supplying the material required for the oxide of iron to combine with. In addition to this sand was fed in from time to time. Into the converter, 1700 cubic feet of air per minute were injected at a pressure of about 20lb. The air passing through oxidised the sulphur, which escaped as sulphur dioxide. The iron was simultaneously oxidised to ferrous oxide, which at once combined with the silica to form a slag of ferrous silicate. The slag so made contained about 55 of ferrous oxide to 30 of silica. The temperature of the escaping gases was 60deg. C. at the commencement of the operation, rising to 726 C. at the finish.

Holloway sought to Bessemerise the whole of the raw ore in the converter, by pouring off the slag, and adding raw pyrites from time to time, until a high grade matte was produced. So far as melting the material was concerned and the concentration of pyrites into matte, he amply verified his opinions: but he saw that the ordinary Bessemer converter was not a suitable vessel, for it had no place for the regulus to accumulate out of reach of the blast. As showing the thoroughness with which the work was done the analyses of the gases was made after the blew had proceeded for six minutes, and then again for twelve:

	Oxygen.	Sulphur Dioxide.	Nitrogen.
After six minutes	—	14	86
After twelve minutes	0.75	10.88	88.37

In the second case a little air got in. In reality, the whole of the oxygen was used up in each case. Assuming that the first atom of sulphur is distilled off and that Fe. S remained, the first reaction would be approximately represented by

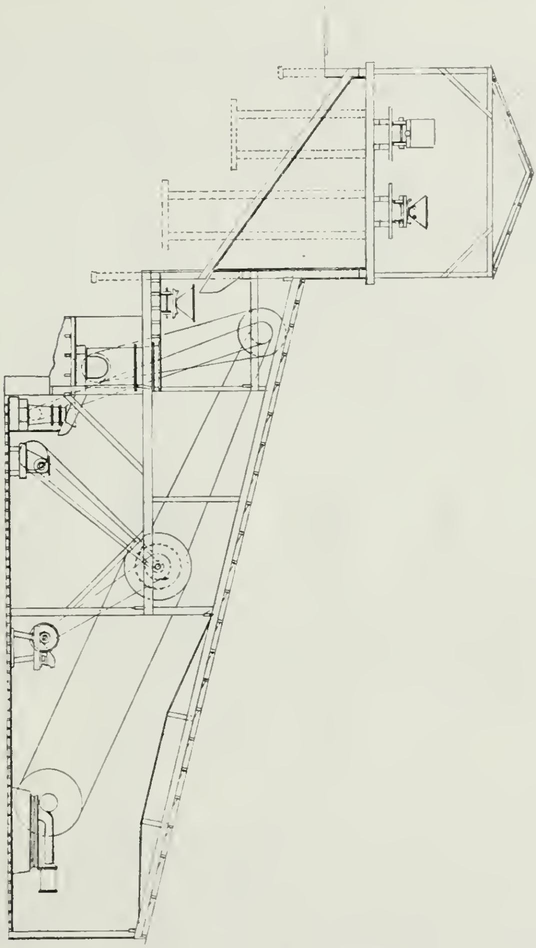
$$3FeS + 5O_2 = Fe_3O_4 + 3SO_2$$

It is difficult to get an equation for the second, except on the assumption that more iron was oxidised than sulphur. Holloway also noticed that a large amount of unburnt sulphur distilled off, and in his eagerness to embrace every saving operation under his process, he set about devising means of condensing this before he had finished with the vital parts of the process. His final experiments were carried on in a modified spiegeleisen furnace. The sulphide was melted as before, poured into the hearth, the blast turned on, and air blown through the molten mass. Fresh pyrites were fed in at the rate of three tons per hour, and when all seemed satisfactory, the blast, accumulating under the too pasty slag, caused a large portion to be suddenly ejected. The next experiment was blocked on account of the condensation of the sulphur, and the slag blown out, choking the flues. These failures, which would now be looked upon as mishaps or small accidents accompanying the running of any new plant, were the causes of the financial failure of the Holloway process. It is of interest to Victorians that this process was adopted at Bethanga, when matte smelting was carried on at the mines. Holloway failed by attempting too much. Had he confined his attention to the enrichment of low grade ores by Bessemerising or the conversion of high grade matte into copper, he probably would have succeeded; but he failed by attempting to do more than is done even at Mount Lyell at the present time. He claimed he could get metallic copper in the one vessel from low grade ore, but only succeeded in obtaining a high grade matte.

Manhes started with the object of producing copper from a high grade matte, and succeeded. He first used a small Bessemer converter with bottom tuyeres. The melted matte was run in and air forced through. The sulphur was rapidly oxidised, and escaped as SO_2. The iron was also oxidised, and the ferrous oxide produced attacked the silica lining, forming a fluid slag. The iron oxidised more rapidly later on, and magnetic oxide separated, not enough silica entering into combination. The result was that a viscid crust formed on the surface of the melted matte, and the pressure of the gases underneath gave rise to great bubbles, which, from time to time, burst and threw the slag out of the converter. The metallic copper liberated, sinking into the tuyeres, clogged them before all the matte was reduced. Manhes at once recognised the cause of failure, and placed the tuyeres at such a level on the side that the copper when reduced could not reach them. This alteration was eminently successful. Copper was rapidly produced, and kept molten by the heat of oxidation, going on above the surface. He also used two converters, one to enrich a lean matte up to 60 per cent., and the second to produce metallic copper. In order to produce more fluid slags, the silica proposed first by Holloway was added, and the success of the process assured. Shortly afterward it was introduced by some pupils of Manhes into America, and whatever it lacked before was soon remedied when the American metallurgists got hold of it. In fact, the true developments may be said to have taken place in the States.

While the Manhes process has been successfully used for many years, the pyritic smelting, as defined at the commencement of this article, and as foreshadowed by Holloway, has never been so successfully accomplished as at Mount Lyell. If Dr. Peters had his doubts about pyritic smelting, Mr. Robert Sticht, from Butte, Montana, who took charge of the metallurgical department in 1895, had none. There was no hesitation on his part in recommending the immediate erection of a plant capable of treating 500 tons per day, to be afterwards increased to 1000 tons a day plant. This work was started in 1895, in 1896 the first furnace was started, and in 1899 the full double plant was completed.

The Bessemerising of Mount Lyell ores is carried on in three stages. First the enrichment of a low grade ore to 12 per cent. matte; second, an enrichment from 12 per cent. to 45 per cent.; third, the conversion of 45 per cent. matte to blister copper. It is not necessary for most of your readers to give at this stage more than a general description of the reactions which go on in the furnaces. A detailed account will be supplied later. The concentration of the raw ore depends upon part of the sulphides present becoming oxides. These, in their turn, at a high temperature unite with silica, and form a slag, the unaltered sulphide containing practically all the copper and all the gold and silver. The silica must be present in the proper proportions to form a slag, which can hardly be called a chemical compound, but a mixture of chemical compounds dissolved in each other at a high tempera-



Cross Section of Sampling Mill.

ture. Their fusibility depends on the proportion of bases, such as the alkalies, lead oxide, copper oxide, manganese oxide, iron oxide, lime and alumina to silica, and also upon the nature of the oxides themselves. For instance, if definite proportions of the bases given are intimately mixed with silica, they will fuse in the order given. It is also well known that if two or more bases are present they will fuse more readily than one. The probable changes which go on when the ore is fed into the furnace is that in the upper regions where reducing or neutral conditions exist, the pyrites lose nearly half their sulphur, which distils off. This may be plainly seen escaping. These pyrites are fed in not in molecules, which alone are represented by reactions, but in masses up to and over a foot in diameter. These decrepitate violently, bursting asunder into fragments in the upper part of the furnace. The reactions which go on may be ultimately expressed as:

$$7FeS_2 = Fe_7S_8 + 3S_2$$
$$3FeS_2 = Fe_3S_4 + S_2$$

Part of the sulphur being evolved. The extra supply of air gaining access to the throat of the furnace causes this to burn freely. As the smaller blocks of pyrites descend to a hotter zone of the furnace, they meet a better supply of air, but can only be superficially oxidised. The tendency at this stage is for this fusible material to run and choke the furnace up. To overcome such a contingency coke amounting to about 2 per cent. of the charge was fed in. This proved to be effective. Latterly billets of wood about 15 inches in length and 9 inches in diameter, have been fed in until recently. These constituted about the same percentage of the charge. This was found to be thoroughly effective. The calorific values of such a material is a mere trifle, and as a matter of fact far more heat escapes from the top of the furnace than is represented by the combustion of such fuel, so that this is pyritic smelting pure and simple. In fact, if some neutral solid could be added which would disappear after a certain zone was reached, the effect would be exactly the same. But at present, even the wood has been discarded, and the pyritic ore is smelted without any carbonaceous fuel whatever, except the occasional use of an insignificant addition of coke, amounting to only $\frac{1}{4}$ per cent. of the total daily quantity of charge treated. The probable changes which take place may be indicated by saying after the expulsion of the free sulphur, a crust of magnetic oxide forms, while within is a kernel of magnetic sulphide. The fusible sulphide of copper is driven towards the centre after the manner of kernel roasting. Silica, coming in contact with the oxide of iron when hot enough, will react as follows:—

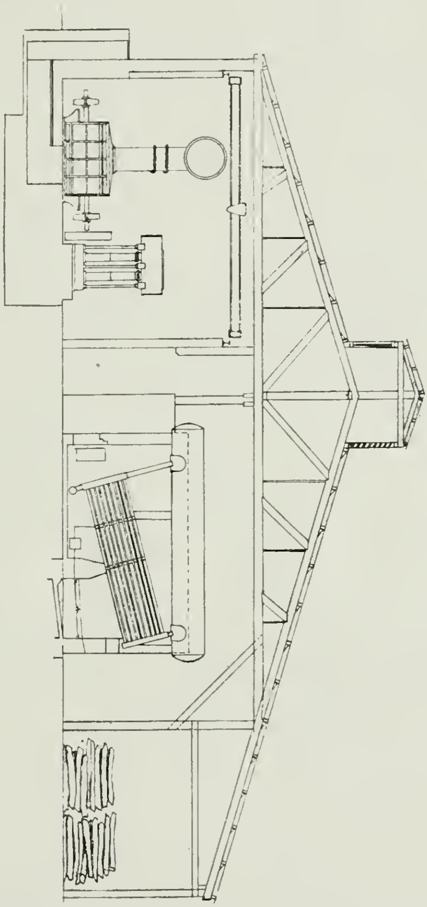
$$Fe_3O_4 + SiO_2 = FeSiO_3 + Fe_2O_3$$

The Fe_2O_3 formed at once reacts with the internal sulphide with which it is in contact, and gives:—

$$36Fe_2O_3 + Fe_3S_4 = 25Fe_3O_4 + 4SO_2$$

a very small portion of the sulphide being sufficient to prevent the oxidation of the iron. In ordinary roasting some reaction corresponding to this goes on, for while oxidation is rapid enough at first, much of the oxidised material becomes magnetic at a higher temperature, even if excess of air be present, a very small quantity of copper pyrites being sufficient to prolong the time of roasting.

The formation of a slag by the union of two infusible substances, or substances which only melt at very high temperatures, is a slow operation. For instance, between silica and lime, if both were solid, it would only be at the point of contact that each would react. A small amount of lime would dissolve in this slag on one side; a small amount of silica on the other. These in their turn would react to form more slag, and so on until the whole of them liquefied. Such an operation could be materially hastened by adding some slag which would act as a solvent for both materials. In practice, this is done. The main base present in the first slag formed is iron oxide, but as the charge sinks down alumina also combines with the silica and behaves as a base, every 102 parts of alumina displacing 216 parts of iron oxide to form the same slag. Slags are generally worked in percentages. Silica, where the base is mainly iron, running from 36 to 44 per cent.; but with



Cross Section of Engine Boiler and Blowers.

alumina in large quantities such percentages would be unsafe guides. In some cases alumina behaves as an acid, and is said to form slags, but as a rule aluminates are very infusible, many of the natural ones being absolutely so. Alumina also has the effect of lowering the specific gravity of the slag. In the first operation the addition of limestone is not essential, a double base being formed with iron oxide and alumina. In addition to these materials, there is in the ore barium sulphate, which passes into the furnace. In what state it is present in the slag has not been, generally speaking, satisfactorily determined. In an oxidising blast, it will melt down as such, and probably only dissolve in this state in the slag itself. On the other hand in a reducing atmosphere the following reaction takes place:—

$$BaSO_4 + 4C = BaS + 4CO$$

In this state it is highly objectionable, and forms infusible masses, which dissolve in neither matte nor slag. The atmosphere of the furnace is highly oxidising, so that the barite is not reduced by carbon, and the reverse of the following reaction takes place in practice:—

$$BaSO_4 + FeS = FeSO_4 + BaS$$

so that it is hardly possible barium sulphide is formed. The third possibility is that it is present as oxide, due to the inter-reaction of barium sulphate, and silica at a high temperature in a current of heated air.

$$BaSO_4 + SiO_2 = BaSiO_3 + SO_2 + O$$

It is probable that this change takes place, and that the barium oxide is present as a harmless or even useful slag making material.

Since writing the above, experiments on Mount Lyell slags have been carried on by students at the Bairnsdale School of Mines. The finely powdered slag was boiled with water. If barium were present as Ba S, the following reaction would have taken place:—

$$2BaS + H_2O = Ba\ (SH)_2 + Ba\ (OH)_2.$$

Possibly the $Ba(OH)_2$ would have united with silica also in solution, but no $Ba(SH)_2$ was found dissolved. Next, by treating the slag with dilute acetic acid, no sulphuretted hydrogen was given off, and yet barium was found in solution, showing that it was present neither as sulphide nor sulphate, so that it must be present as oxide in combination with silica, or as a silicate. Mr. Sticht has also kindly informed me that there is not sulphur enough in the slags for the barium to be in any other state than that of silicate, so that settles the matter.

As the material descends in the furnace it is subjected to a greater oxidising action. The sulphides are rapidly oxidised, and the temperature is greatly increased. The slags have greater solvent power, and descend in liquid streams. By increasing the charge of siliceous material and increasing the air blast a larger amount of pyrites would be oxidised and a greater charge put through. If, however, the silica and blast were both increased then a smaller amount of richer matte would be formed. It is not held desirable to do this to a greater extent than is done at present on account of increased losses that would take place with the production of a richer matte. The molten matte is heavier than the slag, and sinks through it; but more important still is the fact that neither will dissolve any appreciable quantity of each other; in fact, the two liquid materials are as distinct in their properties as oil and water, so that, although their density may approach each other provided the slag is not viscid, they will readily separate. A small amount of copper is no doubt oxidised, and in such a condition must pass into the slag. A small amount in very fine slots of matte has not time to separate from the running stream of slag, and as such passes out; but the losses in smelting in this way are less than if the material is smelted in the ordinary way, for the slags are more uniform in composition. Neither have they suspended impurities, such as particles of unburnt carbonaceous matter, or metallic iron, which serve to entangle globules of metal or matte.

The air is another important factor which must be considered. The following is the main ultimate reaction, which provides the heat for the furnace, and serves for the subsequent changes:—

$$6FeS_2 + 11O_2 = 6FeO + 8SO_2 + 2S_2.$$

In other words every 360 parts of pyrites require 176 parts by weight of oxygen, or very nearly 2 to 1, or one ton of pyrites takes 1096lb. of oxygen. This is accompanied by 3669lb. of nitrogen, which passes through the furnace unchanged, and which only serves to dilute the action of the oxygen. The volume of this would be about 61,000 cubic feet. Considering that each furnace at Mount Lyell is capable of running down 300 tons of pyrites in 24 hours, or at the rate of 0.21 tons per minute, upwards of 12,800 cubic feet would be required per minute to oxidise the whole of the pyrites. The amount now actually used is nearly double this, or 25,000 cubic feet per minute. This is heated to 300 deg. F., and supplied at a pressure of 36oz. and over. It would be interesting to know the amount of oxygen actually consumed, but it seems to be impossible to get a fair sample, and even if it were obtained the results would not show how much could be used, for much of the air escapes up the blow-holes, which form in the column of charge, chiefly at the ends of the furnace, and probably a larger portion never comes in contact with the pyrites at all. Were it blown through a liquid layer, then, as in Holloway's experiments, the whole of the oxygen would probably be used up. It is of little use discussing the thermal conditions which accompany such chemical changes as those indicated. Our knowledge of what takes place at high temperatures is not exact enough. The specific heats of the products formed, the latent heat of the slags and mattes, the certainty of chemical changes, and other necessary factors are too indefinite to build any solid superstructure on. And it would appear that many of the conclusions obtained so laboriously with regard to this branch of the subject are absolutely contradicted by the results at Mount Lyell. The water vapor present in the air is not eliminated, and as the air must be almost constantly at its saturation point, a very large amount must be drawn into the furnace. At a high temperature the effect will probably be to form sulphuretted hydrogen and oxidise some iron, the H_2S in the upper part of the furnace again reacting with SO_2, giving sulphur and water, the latter escaping.

The matte as run out from the first furnace, is broken up, mixed with the requisite fluxes. In this case, limestone is added, and run down as before. The matte from this operation is oxidised in the converters, and in this case a different reaction goes on. At first, the air on being blown in oxidises the sulphide of iron, which turns to oxide, and sulphurous acid is given off. The oxide of iron combines with the silica lining of the converter, forming a silicate. When all the iron has been removed only sulphide of copper remains, Cu_2S. On air being forced through this, the sulphide is oxidised in part:—

$$2Cu_2S + 3O_2 = 2Cu_2O + 2SO_2.$$

This at once reacts with Cu_2S present, giving—

$$Cu_2S + 2Cu_2O = 6Cu + SO_2$$

The oxidising action which goes on causes copper to separate even before the sulphide of iron has all disappeared.

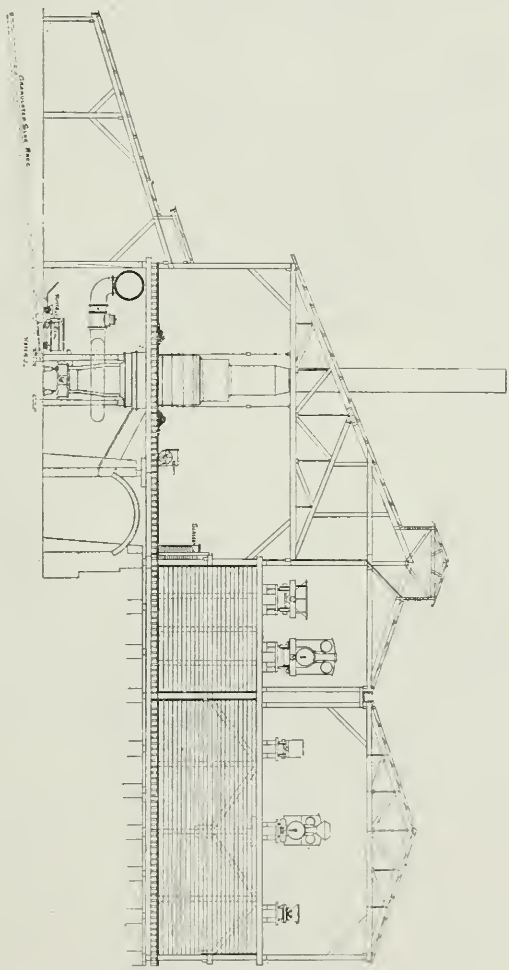
The smelting and reduction works are on the site chosen by Dr. Peters. At first sight, especially to those unused to metallurgical operations on a large scale, it would seem as if a more natural position could have been selected down the glaciated Linda Valley. The original scheme provided for a long tunnel through the ridge, which separates the mine from the works. The ore was to have been sent through this to the smelters. The railway terminus was decided upon on the Queen River side of the ridge, because the Railway Act provided that the railway should run to Strahan, and would not have been passed but for this provision. The ridge could, however, have been surmounted only by a great expense of money and the lengthening of the line by over seven miles. The site was therefore practically fixed through this consideration. As it subsequently turned out, the selection was an admirable one, for nearly all the raw materials and fluxes necessary were discovered within a radius of less than a mile from the works, whereas on the mine side of the ridge, neither water, fluxes nor firewood are available in sufficient quantity, and would have to be brought from miles distant. At the same time, work at the open cut is not now retarded by the fumes from the smelters, while it is certain that if smelting had been started in the contracted space of the Linda Valley, it would not have been long before an injunction restraining the company from working would have been placed upon it on account of the fumes. The limestone is obtained by quarrying a bed, whose dip is nearly vertical, and whose strike is into the face of a hill over 500 feet in height. The material is removed from 5 benches, which are about 7 chains long and about 25 feet high. The rock is sent down to a 10 x 20 Blake style crusher run by a motor, crushed to a convenient size, and deported by means of an overhead rail system of delivery into storage bins below. From these, trucks are filled and hauled by a locomotive to the works. The analysis of the limestone is—

	Grey.	White
SiO_2	10.92	3.26
FeO	.57	.85
Al_2O_3	2.88	.86
$MgCO_3$	3.00	trace
$CaCO_3$	83.40	95.10

The lime required at the works is made by burning in two kilns each 18 feet high and 6 feet 6 inches in diameter. For siliceous flux there are numerous sources in the reduction works valley; in fact, one side of the Queen River consists of quartzite hills. A convenient deposit of this rock was attacked in contact with limestone, which is a bed parallel to it. The material on top of the hill, 500 feet above the valley, is less friable than that below, hence work is carried on above. A quarry is worked on three benches. The blocks of quartzite are sent down to the crusher bins by means of an Otto ropeway 1500 feet long, of the same type as that used for the ore from the mine, which will be described later on. There are two largest sized rotary Comet crushers driven by a motor, which break the material down to 3-inch gauge. The sand is separated by perforated percussion tables, and is made use of for converter linings. The crushed quartzite falls into bins, from which the trucks are loaded direct and hauled along the tramline to the works. Analysis of the silica flux is as follows:—

	White.	Dark.
SiO_2	91.44	88.72
Fe_2O_3	1.54	3.60
Al_2O_3	3.09	7.03

Another fortunate find was a deposit of excellent brick clay, from which the bricks for the works, amounting to 4,000,000 were made,



Cross Section No. 2 Smelting Furnace.

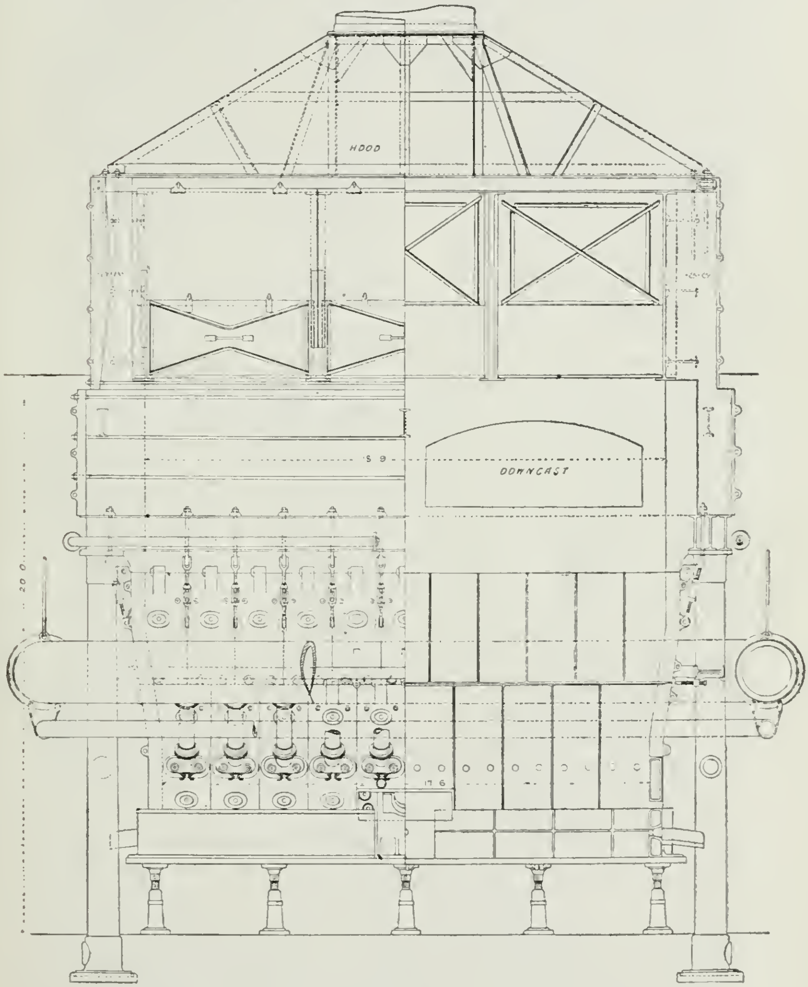
and still another also alongside the works was an excellent fireclay, from which 128,000 fire bricks were made last year.

It is evident that while the proximity of such valuable deposits to the works may be termed lucky, yet how often in Australia have we seen deposits just as valuable altogether overlooked. The credit is due to those who make the most of natural opportunities. Instead of driving a long tunnel and conveying the ore through the dividing ridge between the mine and the reduction works, two systems of transport are adopted. Both make use of the greater fall on the works side of the ridge to haul the loaded trucks or buckets over the crest and return the empties. Almost all the ore from the open-cut is transported from the bins to the works by a large aerial ropeway. The buckets carry 11cwt. of ore, and travel on a fixed rope, which is 35mm. diameter on the loaded and 25mm. on the return side. A continuous ¾-inch rope 16,000 feet long serves as the travelling rope attached to the buckets. On the crest of the hill, the ropes on which the buckets travel are replaced by fixed rails for a length of 300 feet; this has saved the running ropes from the great wear which would have taken place had they been continuous. The speed of the buckets is six feet per second, and in every 30 seconds a bucket is delivered, or 22cwt. of pyrites per minute. As much as 220,304 tons were delivered in a year. The horizontal distance of this line is 6750 feet: the rise on the mine side of the ridge is 430 feet, and on the works side the fall is 920 feet. The mean gradients on the mine and smelter sides are respectively 1 in 2.9 and 1 in 6.3. The span between the standards, which are from 8 to 50 feet in height, is up to 1300 feet, while the ground is as much as 220 feet below the buckets. The total cost of delivery, including maintenance and repairs, is from 6d. to 7d. per ton.

The haulage line chiefly carries underground ore, and may be used, when worked three shifts, to replace the aerial tram. In this the trucks run on a 2-feet tram line, which crosses over the hill, and are hauled up one side and let down the other to a small steam tram, which takes the ore ¾ of a mile to the works, and returns the empty trucks.

The company possesses one of the most interesting 3 feet 6 inches gauge railways in the States, which passes through one of the most picturesque strips of country in Tasmania. The merits and demerits of the Abt system have been so fully commented on that any remarks would be superfluous. A fine engineering shop and foundry makes the company one of the most self-reliant in Australia, while what is most satisfactory of all, there is no evidence of that waste and reckless extravagance so common on many of our large mines. The Mt. Lyell Company may be said to be the only one in Australia which has its works laid out for systematic sampling. The extra small expense incurred is more than recovered by the accuracy of the information gained; further, without some such system it would not have been possible to buy ores on an extensive scale. Every 25th bucket from the open-cut, every 16th of underground ore, and every 18th truck of mineral flux goes to the sampling works. On purchased ores, every fourth to every eighth bucket or truck is taken according to its richness. The selected ore is broken down by crushers, etc., and quartered in the usual way on the sampling floor, the balance being sent direct to the respective ore bins. Each bucket of ore as delivered by the aerial tram is detached and run on rails to the suspended weighing machine, the weight is entered up. The bucket is run round by hand, tipped into its bin below, and returned on the upward line, the whole operation being done with such rapidity that there is no interruption in the continuity of the stream of buckets entering and leaving. The ore and fluxes as delivered in trucks are weighed on a long weighbridge, which takes 25 tons at a time, but can take 75 tons, and weighs down to 56lb. These materials are also tipped into special bins above the smelting floor.

The smelting works are divided into two sections, that which was first erected being known as No. 1 plant. It contains six blast furnaces, each being 40 inches by 168 inches at the plane of the tuyeres, except No. 3, which is

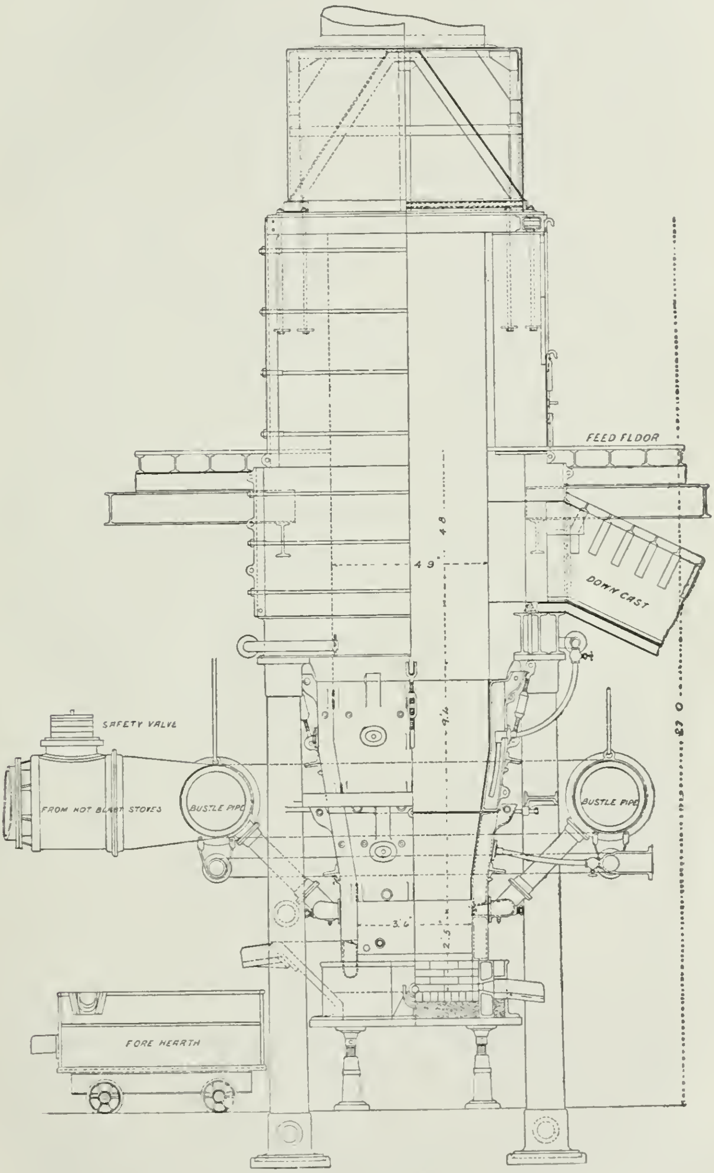


Blast Furnace. Longitudinal Elevation and Section.

36 inches by 126 inches, and 20 feet to the charge floor, the number of tuyeres being 32, No. 3 having 24. After the experience gained in working these furnaces, five larger ones were built, and this group is known at No. 2 plant. Each of these furnaces has 40 tuyeres 3 inches in diameter, and a sectional area of 210 inches by 42 inches at the tuyeres, and are 20 feet in height from tapping floor. Details of one of these furnaces are supplied in the excellent

drawings made by the company's officers, and reproduced. The height of the ore column above the tuyeres, or to the level of the downcast is 9 feet 4 inches, while above this level on the feed floor is 4 feet 8 inches, the depth from the tuyeres to the bottom of the furnace is 2 feet 5 inches. The furnace is jacketed almost up to the level of the downcast, the jackets being made in sections of cast-iron. There are two horizontal sections, the lower being 1 foot 9 inches wide, and 4 feet 6 inches high, with a 4-inch water space; through each of these passes two 3-inch tuyeres, with centres 10½ inches apart. The tuyeres are 1 foot 4 inches above the bottom of the jacket. The whole of the tuyeres are at the sides, there being none at the ends. The upper row of sectional jackets break joint with the lower, and are 4 feet in height, and 1 foot 9 inches wide. The lower portion of the furnace, including the jackets, is carried on cast-iron plates 2½ inches thick. These are supported on screwjacks. The upper portion is independently carried on rolled steel girders, which are supported on cast-iron pillars. The tymp jacket is of steel. Both matte and slag flow out together continuously, the blast being trapped by setting up the slag spout, thus lowering the jacket round which both matte and slag have to flow. The air is now blown in hot at No. 2 plant and cold at No. 1 plant. When heated air is used it is warmed by causing it to pass through a hot blast stove; this consists of a series of cast-iron U tubes, 9 feet long and 9 inches internal diameter. Each U tube practically consists, so far as shape is concerned, of a cylinder doubled round on itself, the doubled portion forming a partition plate, the ends being turned round at right angles and flanged so as to connect to the next tube. The air is forced through a series of 10 rows of seven of these, the external surface of the pipes being exposed to the heated gases from an ordinary grate furnace. For No. 1 plant there are 4 stoves, with 56 U tubes each; that is, a series of eight parallel rows of 7 each; while in No. 2 plant there are 4 stoves, each having 70 U tubes. The air is heated to about 350 deg. F. The air is supplied by Roots blowers, there being nine No. 8, each with 116 c.f. displacement, and three reserve No. 7. The former and one of the latter are each driven by a compound condensing engine, 12 inches and 22 inches by 18 inches, set up in marine style, and coupled direct by flexible coupling. The air, which is thus supplied in units, is delivered to a common main.

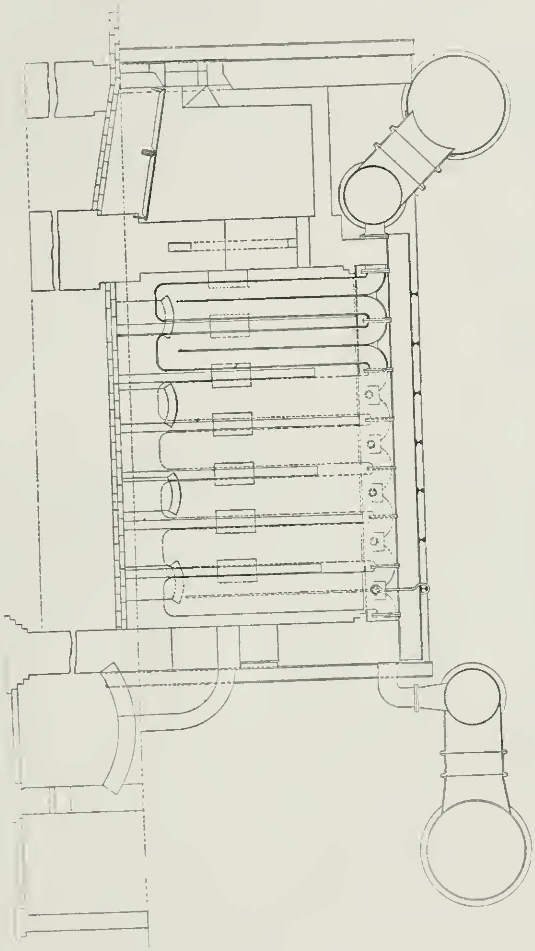
The steam is generated in 12 Babcock and Wilcox boilers, two in each plant being held in reserve. Green's economisers and superheaters are used, the estimated saving effected by the former being 15 and the latter 16, or a total of 31 per cent. This experience tallies with that at Mt. Morgan, where they were introduced some years ago. It may be said that so far as cheap power is concerned there does not seem to be a chance of lowering the present figures at Mt. Lyell. The bustle pipe which passes round each furnace is 21 inches clear internal diameter and brick-lined. At the present time there may be said to be two of the large furnaces working on ore, and on an average they smelt 280 tons of Mt. Lyell pyrites daily; sometimes as much as 350 tons of pyrites are run down, while the daily output of raw material fed in is over 500 tons. Phenomenal tonnages are sometimes reduced, as much as 724 tons being put through in 24 hours by one furnace. These unprecedented returns were obtained by simply supplying more air. At first the air was distributed over from 7 to 8 furnaces, whereas it is now all passed through three to four. Simultaneously the blast pressure has risen, and ranges from 36 to 40oz. at the blowers, and from 30



Blast Furnace. Cross Section and End Elevation.

to 34oz. at the tuyeres. Such pressures for copper furnaces were previously unknown in Australasia, and in fact for the whole world, with the exception of the United States and British Columbia.

As typical of the composition of the furnace charges may be given what is called the "straight pyritic" charge:—Mt. Lyell pyrites, 2240lb.; barren silica, 200lb.; metal-bearing fluxes, 500lb.; limestone, 100-150lb.; slag, 300lb.; coke, 40-60lb., if used. One of the most interesting sights at the works is the feeding of these furnaces. They are open on both sides, and covered with a hood connected with an iron stack, through which any gases escaping above the charge, and not drawn into the downcast, find their way. The bulk of the waste gases, with flue dust, pass to the downcast, which is 7 feet 9 inches wide, and 3 feet high. The dust is caught in chambers, while the gases are drawn through a tall stack. The bins are arranged in a row parallel to the furnaces. The workmen run in their charges, weigh them out quickly; a block more or a block less suffices as a rule to give the exact amount. This is dumped alternately along on one side and then the other of the furnace, then rapidly shovelled in. The pyrites are fed in in lumps up to a foot in diameter, the silica and metal-bearing fluxes in splintery 3-inch pieces, the limestone in fairly large pieces; coke is fed close alongside the walls. The wood experimentally used was in ordinary blocks of about 18 inches by 9 inches. As soon as shovelled in, the pyrites decrepitate violently, a certain amount of sulphur distils off, and instantly takes fire on meeting the external air. In a few minutes the whole mass sinks and is spurting jets of flame from every orifice, while there is a continuous roar from the enormous volumes of gases forced through the furnace. The maxim of keeping the top of the charge cold cannot be carried out here, for it seems impossible to feed each furnace fast enough, so rapidly does the charge drive. A cool top would mean that there would be constant trouble with throat accretions and no advantage whatever gained, except appearances. The burning of the volatilised fraction of sulphur which is unavoidable does not constitute a loss of heat to the furnace. The sulphur distilled off, together with that oxidised lower down in the furnace, is burnt to sulphur dioxide with small quantities of sulphur trioxide; these pour out of the tall stacks, and meeting the moisture-laden air, form a white cloud or a pillar of smoke, which may be seen for 20 miles; the sulphurous acid so formed in course of time is transformed into sulphuric acid, and an equivalent of 1000 tons per day of this material is ultimately precipitated as dilute sulphuric acid. The devastating effect of this acid rain on the district is such that the blackness of ashes now marks where the primeval forest stood, while modern roofs have to be covered with a tile-colored composition or tarred with the dismal black of their surroundings. Some day a portion of these fumes will, perhaps, be used for dealing with the low-grade siliceous ores; but it would not be possible to store or dispose of one-half of the acid that could be made were all the sulphur transformed to sulphuric acid. The ores and fluxes fed into the furnace in less than two hours are running out in the form of matte and slag. These flow into the forehearth, which is a rectangular vessel 44 by 62 inches, and 26 inches deep. It is constructed of cast-iron plates, lined with brick and protected by water-cooled coils between the lining and sides. The overflow spout for the matte from the forehearth is made of a solid piece of cast-iron cast round an ordinary iron pipe bent to horseshoe shape; through this a stream of water runs. Near one corner of the top of the forehearth is placed the overflow opening for slag, which flows into a smaller forehearth for safety; into this the slag runs, and overflows in a regular stream. The falling molten slag is struck by a jet of water, and is instantly granulated and swept



Section Through Hot Blast Stoves.

away. The first forehearth is tapped from every 3 to 10 minutes; the second is tapped occasionally, about once per hour. The pots of matte are poured into shallow sand moulds and allowed to cool. The first matte produced carries about 12 per cent. of copper.

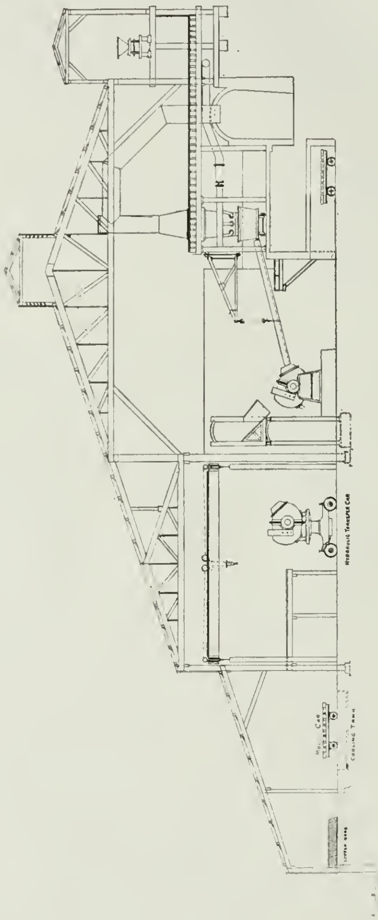
The sluice down which the slag runs is paved with slag blocks, whose glassy, smooth surface offers a minimum of friction to the flowing stream. In the early days there seemed to be room enough for all the slag produced, but when figures passed from hundreds of thousands to millions of tons, and acres of low-lying land became covered with this black, heavy, granulated material, some device had to be adopted to keep it within limits. After some experiments, special centrifugal pumps were used; these have a 10-inch suction, and a 9-inch discharge, with 18-inch beaters or vanes, and are run at 540 to 600 revolutions per minute. The pumps are driven by motors, and have most satisfactorily solved the difficulty. There is a good deal of misconception about this matter, for it has often been stated that the system of granulating slags originated at Mt. Lyell. This is not so; nor was it ever claimed by Mr. Sticht. That slags could be so granulated was known for many years, but could only be adopted where water was plentiful and the fall sufficient. The methods of raising granulated slags by means of a centrifugal pump is new, and this certainly originated at Mt. Lyell. The general analysis of the slags from the first operation is as follows:—SiO_2, 36.66; FeO, 50.67; CaO, 1.20; BaO, 1.90; Al_2O_3, 7.47; Cu, 0.25. The dust from the chambers is mixed in a Chilian mill with about 4 per cent. of clay, and a little water, and made by hand into bricks, each weighing about 14lb. These are partly dried by placing them on shelves, alongside which hot pots of slags are ranged. As much as 4666 tons were handled in a year, the assay being—Copper, 3.5 per cent.; silver, 2.5oz; gold, .07oz. per ton. This simple method of treatment has proved effective, and is much cheaper than more elaborate ones. Experiments were carried on with the use of cold blast, an amount of wood fuel being added to the charge of nearly the same weight as was used in heating the blast. This was wholly successful, and showed that with a small percentage of fuel the hot blast could be dispensed with altogether. Even more interesting and instructive has been the work carried on quite recently, when fuel has been dispensed with altogether as part of the charge and pyritic smelting pure and simple carried out for the first time in its entirety. The first matte is broken into blocks when cold, and transferred to the bins over No. 1 plant. The matte produced is treated nearly the same as ore and run down in the same way as in No. 2 plant. Limestone, however, is added in this case. This furnace was run with a cold blast, and the slag and matte flowed continuously into the two forehearths as before, No. 1 being tapped every 5 to 10 minutes, No. 2 about every hour. The matte in this case contains 45 per cent. of copper. As before, this matte was poured into sand moulds and broken when cold. The slags from this furnace assay in general—SiO_2, 41.70; FeO, 43.14; CaO, 8.16; BaO, 0.16; Al_2O_3, 5.46; Cu, .35. At present the large body of slag produced cannot be put to any use, but it may happen in future that portion of it may be used for the production of metallic iron, to be used for precipitating copper according to wet way processes. This, however, is not very probable.

The second matte, containing about 45 per cent. of copper, passes to the converter plant. This is arranged so as to form two equal sized departments, each capable of producing from 25 to 40 tons of copper per day. There are two remelting furnaces, fourteen converting vessels with stands for six, two No. 5 rotary blowers for the remelting furnaces, each having a separate engine, two Fraser and Chalmers' horizontal compound condensing blowing engine, 16 inches and 24 inches by 30 inches, with air cylinders 30 by 30 inches, delivering 3000 cubic feet of air at 60 revolutions, the average blast pressure at the converters being 8lb. per square inch.

The remelting furnaces are about 4 feet external diameter at the tuyeres and 7 feet 6 inches to the feed floor. Into this furnace broken matte, a small amount of fuel, and occasionally some siliceous purchased ores are fed. When a sufficient amount of the molten matte has accumulated, the furnace is tapped, and the matte flows down a movable slag channel to a converter. The type of converter used is known as the Stalmann, which is square in section, with a trough-shaped bottom. This is mounted on trunnions, and may be readily tilted by hydraulic power, the rack being on the moving horizontal cylinder through which the pressure water enters and leaves. The lining of the converter is made from a fine white clay and silica. The clay is found near the works, and consists of:

	Per cent.
Silica (SiO_2)...	62.52
Alumina (Al_2O_3)	23.89
Oxide of iron (Fe_2O_3)	.26
Lime (CaO)...	.25
Magnesia (MgO)...	.40
Water and undetermined...	12.68
	100.00

The silica is obtained from the siftings of the quartz flux. The mixture is rammed into the converter shell for a thickness of 18 inches at the bottom and 12 inches on top, leaving a clear space of from 2 feet 3 inches below to 2 feet 9 inches above for a height of about 4 feet. The lining is carefully dried and heated; moisture left in would lead to serious explosions. The molten matte is run in to the hot converter: when a sufficient quantity has been added the flow is stopped, the slag launder shifted, and air at a pressure of about 8lb. to the inch forced in. A vigorous action at once goes on, the iron is oxidised, and at once combines with the silica of the lining forming a slag which floats on the surface. Sulphur dioxide escapes in large quantities, and is carried away through a hood leading to a set of flue dust chambers connected with the main stack at No. 1 plant. Small portions of molten material are rejected from the vessel, but the workmen pay no more heed to these than to the sparks from a blacksmith's anvil. The curved back of the converter tends to prevent the ejection of matte and slag. A large supply of air at high pressure may cause more oxide of iron to separate out than can unite with silica in the time allowed. The slag would contain much magnetic oxide of iron, and become pasty. The accumulating volume of gas under this may cause a boil over, such as terminated Holloway's experiment. The blowing at Mount Lyell is done intermittently. The converter is gradually tilted so that the air passes through the molten matte still unacted on. Any copper noses formed by the chilling of the metal or matte by the cold blast are punched off as the operation proceeds. As soon as the formation of slag has ceased, which is visible by a change of color in the flame to a clear blue, the blast is stopped, the converter tilted, and the slag poured off. Any white metal which runs out at the same time is broken up and returned to the remelting furnace; the slags which are the richest in copper of the series being returned to the blast furnaces. A second lot of matte may be run into the converter, and worked up as before, the slag being poured when the iron is oxidised, and blowing continued until blister copper is produced. During the first blow the flame is greenish blue, while during the last stage it is yellow or clear blue. The workmen can tell by the flame and the solid particles ejected to what stage the conversion has proceeded. If mistaken as to the final change the first few pounds poured out will show whether matte remains. Should matte run out the vessel is tilted back, and a short blow is given.



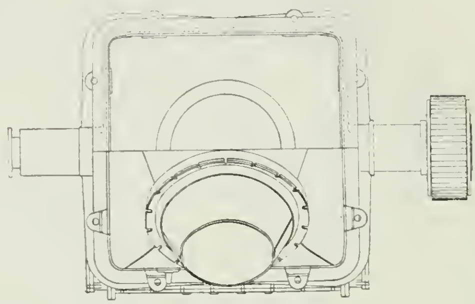
Cross Section of Converter Plant.

On the other hand, the copper may be over blown or contain cuprous oxide. One of the overseers of this department, not knowing of the inter-reaction of cuprous oxide and cuprous sulphide, endeavered to correct an over-blown charge by running a little more matte into the molten copper. The reaction was so violent that the whole contents of the vessel were suddenly thrown out as a mass of froth, so that this interesting reaction is not likely to be repeated in the same way.

The copper is poured into a series of moulds carried on a car which runs on a track directly below the converter. Mould after mould is filled with molten copper, which solidifies almost instantly. The car is run back over a water sump, and the copper slabs tipped out into the water below. They are thus not only rapidly cooled, but acquire the fine red shade which enhances their appearance. This is due to a film of cuprous oxide. The reactions which take place in the converter have already been indicated. They are similar to those in the blast furnace up to the separation of pure cuprous sulphide (Cu_2S). The silica in this case is supplied by the lining of the converter. This is a somewhat expensive way of supplying it. If the lining were a siliceous copper ore a small saving would result, but the extra cost of working in an unsuitable material would more than counterbalance this. The lining is very thick at first, and the charge is correspondingly small, while at the finish there is a danger of the protecting coat being eaten through. The latter risk is minimised in the Stalmann converter, for the blast chest surrounds the front and two sides. The blast may be turned on at the front, so that when that portion of the lining becomes thin the front tuyeres may be shut off, and the side ones opened. The number of blows that a converter will stand at Mount Lyell is from five to seven. Attempts have been made in other places to supply silica by other means. Holloway blew dust through the tuyeres with the air. Quartz has been added to the top of the matte, but none of these methods were successful. It would appear that with the present types of converters the only successful method for supplying the necessary silica is by lining the vessels.

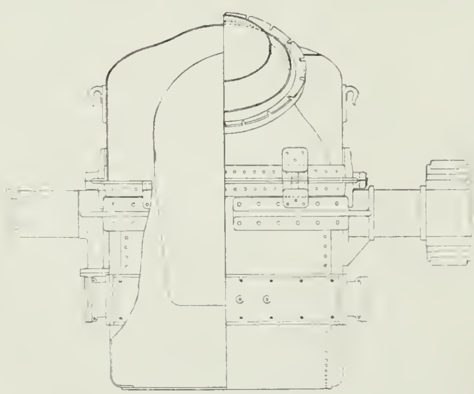
The blister copper is poured into cakes 16 inches by 24 inches by 2½ inches thick. Each weighs about 2cwt. or 10 go to the ton. A lot of 500 cakes, or 50 tons,constitutes one parcel, which is specially marked and specially sampled. The crude bars are trimmed up and weighed. Each cake goes to a drill press, and assay samples are taken by drilling two ⅜-inch holes through the cake. The cakes are shipped to the Baltimore Copper Smelting and Rolling Company in Maryland on a toll arrangement, the products being turned over to the company. The refined product, as ingots, wire and sheet copper, branded B.E.R., is sold in the open market, the silver and gold produced being sold to the U.S. mints. The flat shape of the ingots was decided on because it was found that liquation of the gold and silver occurred to such an extent in ordinary pigs that it was well nigh impossible to obtain accurate assay samples. Silver values would differ by anything from 2 to 100oz. in a 200oz. copper from drill holes bar a few inches apart. The variation was found to be much less with flat cakes, but even in this case a systematic method of sampling is adopted. A template, the size of a copper cake, is mapped out into 150 holes, so spaced as to symmetrically divide each cake. The first cake is marked with two holes, the next with two more, and so on until the seventy-fifth cake has the last two of the template. Each cake is then bored with two three-eighth inch holes, so that the borings from 75 bars represent a sample as if taken from all over one cake, and will be accurate provided that the distribution of precious metals is about the same in each. The borings from a 50 ton parcel are assayed, as before stated, in one lot. The method of assaying the samples is after the plan adopted by Ledoux and Co., New York. This well-known firm does the check assaying for the Mount Lyell Company. Since details are given with great clearness, I will quote in extenso the practice followed in this somewhat difficult problem. The borings are put through a mill and ground as finely as possible. The fine borings are gone over with a magnet to remove any small particles of metallic iron that may have been derived from the mill or drill, and mixed thoroughly.

Copper.—The sample is divided repeatedly on a split sampler until about 20-22grms. remain. This operation is conducted with



Converter.—Plan.

great care, so as to include a proper proportion of fine and coarse borings. The small sample is then placed on the balance, and its weight accurately adjusted to 20grms., which is transferred to a litre flask and treated with a mixture of 200 c.c. water, 20 c.c. of sulphuric acid (Sp. Gr. 1.84), and 60 c.c. nitric acid (Sp. Gr. 1.42). The flask is placed on the steam bath for about an hour, at the end of which the copper will be dissolved. Standard salt solution is now

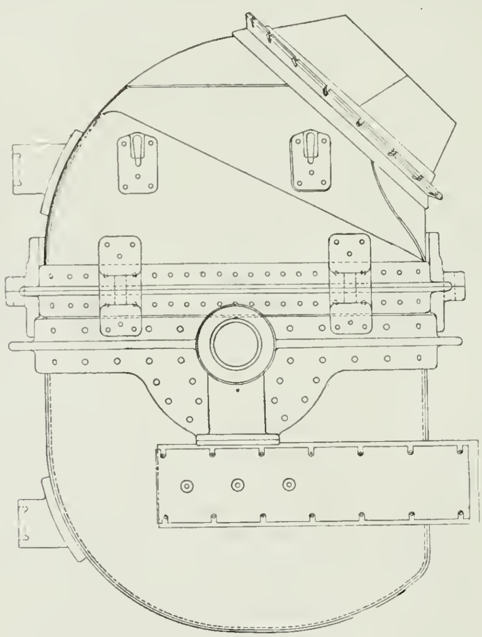


Converter.—Front View.

added to the flask in sufficient amount to precipitate the silver, but not much more. About 100oz. of silver is allowed for in the Mount Lyell bars. After adding the salt solution the flask is transferred to a hot plate, and the contents boiled until the red fumes are expelled. The boiling coagulates the silver chloride. The flask is allowed to cool, and the temperature of the room is noted. Then sufficient cold water is added to the flask to bring the copper solution exactly to the room temperature. Finally the dilution to exactly one litre is completed with water at room temperature, taking care to keep the solution in agitation during dilution to avoid the contraction, which always occurs when water is added to strong copper solutions, so that it forms a separate layer in the flask. After mixing the contents of the flask a portion of the liquid is poured through a dry filter, and the filtrate caught in a dry beaker. From this clear liquid two portions of 50 c.c., each equivalent to 1grm. of the borings are tapped into glasses, and the solution diluted to 125 c.c. The copper is plated down from this on a weighed platinum cylinder; no further addition of acid is necessary. The current employed is 0.3 amperes, and the cylinders have a surface of 10 square inches. The determinations agree to within 0.05 per cent. The apparatus used for tapping 50 c.c. from the liquid is of special construction, so that it measures exactly one-twentieth of the contents of the litre flask. The measuring apparatus, as well as the flasks, are checked from time to time by running through determinations of copper wire of known purity. This precaution is necessary as the glass used in the litre flasks may change in volume to a slight extent after long use.

Silver.—The sample is divided in the same way as for copper. The division is carried so that the small sample aggregates a little more than 3 A.T. This is divided into three accurately weighed 1 A.T. portions, each of which is transferred to a large beaker provided with a cover glass. Over each assay ton is poured 100 c.c. of water, and to this is added 50 c.c. of nitric acid, 1.42 sp. gr. After the strong action is over, another portion of 50 c.c. of nitric acid is added, making 100 c.c. in all, which is sufficient to dissolve all the copper. Boil the liquid to expel nitrous fumes, and remove from the heat. Wash down the cover glass, and add enough standard salt solution to precipitate the silver (only a slight excess is added, the contents being approximately known). Stir well, and allow the beakers to stand over night. In the morning filter off the chloride through double filter papers. Wash with cold water until the copper is practically all removed and wipe out the beakers carefully with a piece of moistened filter paper, which is added to the filter. The wet filter papers are transferred to a 2½ inch scorifier lined with a disc of sheet lead, which weighs about 8grms. The object is to prevent the absorption of water from the filter paper by the scorifier. The scorifiers are transferred to an oven which is heated to about 300deg. C. This burns the paper in a short time. When the charring is complete 20grms. of fine test lead is added to each scorifier, together with 1½grm. more or less of borax glass. The scorification is then made so as to yield a lead button weighing about 7grms. This is effected when the lead ring is about ½-inch in diameter. The slag is saved. The buttons are cupelled in small cupels at the lowest possible temperature. The cupel bottoms are saved, and mixed with the slags. The beads are weighed. If the difference between the maximum and the minimum does not exceed 0.75oz. per ton the beads are united and parted. The gold found is subtracted from the silver. This gold figure is lower by about 0.20oz. than the gold found in the special assay, which will be described later. The cupel bottoms and slag are mixed, fluxed with litharge and glass and a reducing agent, and the lead button is cupelled. The weight of the silver bead is divided by three, since a single fusion is made of all the cupel bottoms and slag. This weight is added as a correction to the average of the other figures. The correction usually amounts to 1.4 to 1.7oz. per ton.

GOLD.—The material is sampled as for copper down to 1 A.T., which is divided into four equal portions of ¼ A.T. These ¼ A.T. portions are placed in 3-inch Bartlett pattern scorifiers, together with 90 grms. of lead. Over the lead is put a cover of about ½ grm. of silica and borax glass. The scorifications are conducted until the litharge closes over and the pouring is done hot. The buttons are freed from slag; the slag is saved. The buttons are then each made up to 65 grms. weight with lead, and ½ grm. silica, and again scorified in 3-inch Bartlett scorifiers. This scorification is kept fairly hot during the whole process until the litharge covers over. The buttons are freed from slag as before, and the slag saved. Then they are united two and two. Two buttons representing ½ A.T. are made up to 90 grms. with lead and ½ grm. silica,

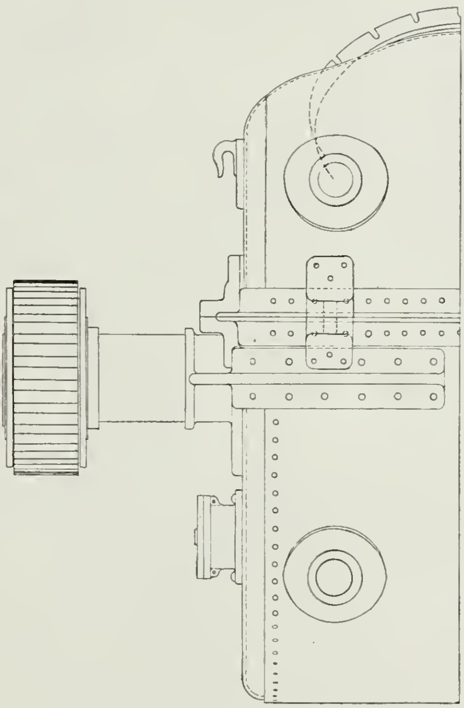


Converter.— Side View.

and again scorified. The slag is saved, and the buttons, which weigh about 35 grms. each, are cupelled. The cupel bottoms are saved, and mixed with the slags. The beads, each representing ½ A.T. of the borings, are parted with very dilute nitric acid (Sp. Gr. 1.078). After boiling, the acid is poured off and washed once with cold water and boiled with nitric acid, Sp. Gr. 1.32. The gold is then washed twice with water, dried, annealed and weighed. The gold determination should agree to within 0.02oz. per ton. The slag and cupel bottoms are fluxed, and the lead button cupelled and parted. The gold correction thus obtained is added to the average of the first figures. The scorifiers used in this assay are of the Bartlett pattern, a broad shallow shape, which allows of scorifying 100 grms. of lead down to 30-35 grms. The shape of the scorifiers is quite important. Assays are made over again if they differ:

More than 0.10 per cent. copper.
More than 0.75oz. silver.
More than 0.02oz. of gold.

It is necessary to state that the assay ton indicated by Ledoux and Co. is the gramme assay ton on 2000lb., or 29.166 grammes. The assay ton used at Mount Lyell is the grain assay ton on 2240lb. It is unfortunate that the same term indicates four different weights. It is also necessary to point out that the assays for gold are done



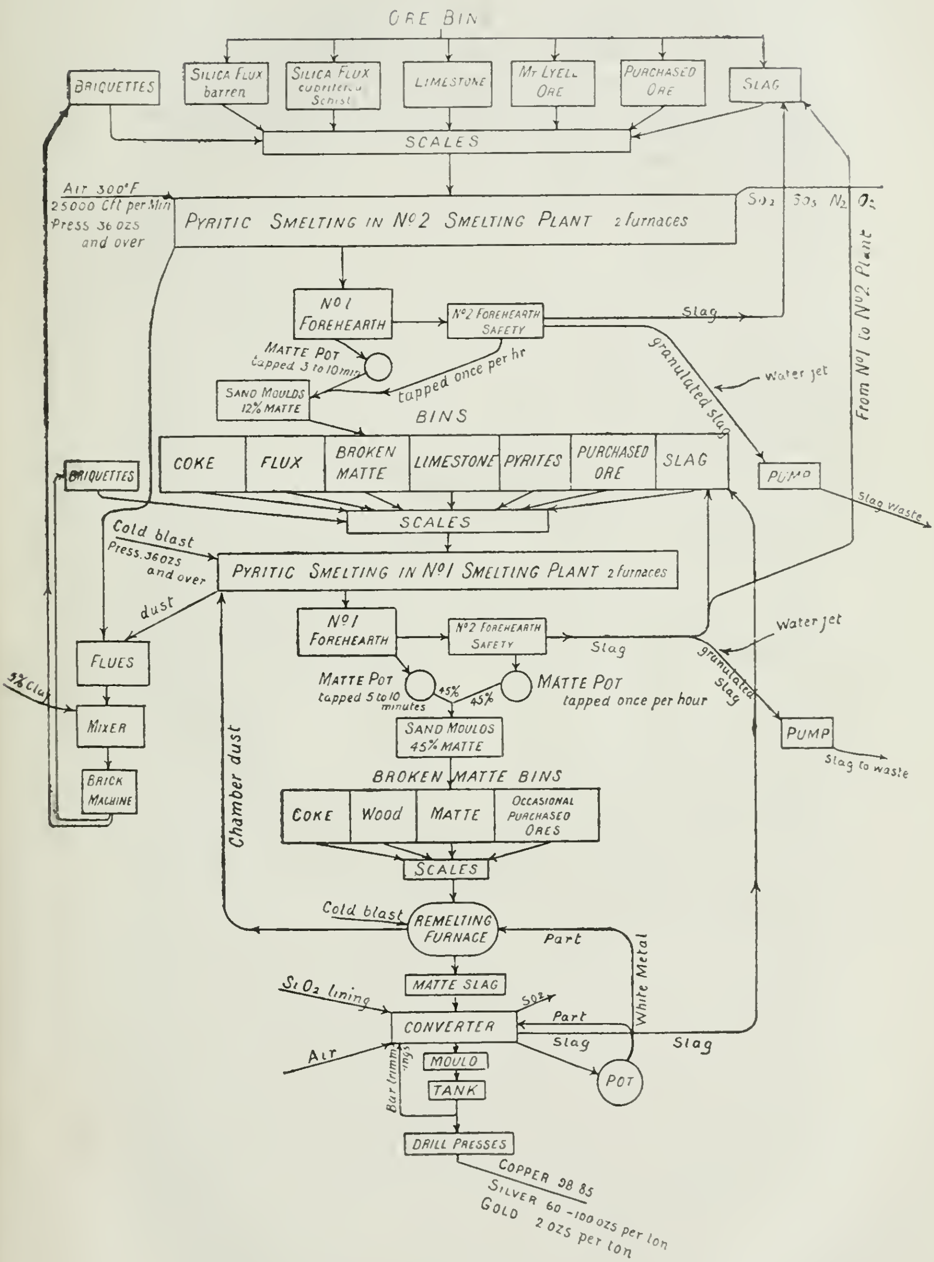
Converter—Section.

on such samples that the results would be of little value on the ordinary assay balance, weighing down to 0.001 grain, or even 0.0005 grains. The balance used must be sensitive for a difference of 0.02oz. per ton to 0.0001 grains on half an assay ton sample. The American balance makers supply button balances, reading to $\frac{1}{200}$ of a milligramme, or 0.000077 grains.

The ordinary assays of ore containing from 2 to 3 per cent. of copper are scorified for gold and silver, one-fifth of an assay ton being taken and 1000 grains of lead free from silver and gold. The scorified button is cupelled and weighed and parted. From 3 to 5 are run down for one gold button. The first matte is treated in a similar manner. For converter matte one-tenth of an assay ton is taken, five being run down together. Copper in mattes is estimated by precipitation of the metal on aluminium, the separated metal being dissolved, thrown down as subiodide, the liberated iodine being determined in the usual way. For the electro deposition of copper in the blister copper assays, gravity crow foot cells are used for generating the current, the E.M.F. being 5 volts.

For slags the ore is fritted with carbonate of soda, on a sheet of platinum foil turned up at the edges. The direct decomposition with acid is not looked upon as reliable. The fritted material is oxidised and evaporated to dryness after the addition of hydrochloric acid. A few drops of sulphuric acid are added. The residue after filtration and washing is weighed as silica plus barium sulphate. This is again fused with soda, when soluble sodium silicate and sodium sulphate and insoluble barium carbonate form. The former two are washed out with water, the barium carbonate dissolved in hydrochloric acid and barium sulphate precipitated. This is weighed, the difference being silica. Ferric oxide and alumina are weighed together. The ferric oxide is then dissolved, reduced with zinc, and estimated with potassium permanganate, the alumina being given by difference. Calcium is precipitated as oxalate, ignited and weighed as lime.

The percentage of copper extracted in the three treatments, i.e., first smelting, matte concentration, and converting from the ore, may be put down as 85, the silver 92, and the gold shows a variable plus over 100 per cent. The last is due to the introduction of minute quantities with the fluxes, and the very minute mechanical losses sustained in the gold assay, particularly in the parting, which are greater than the loss of gold in the blast furnace operations. The copper ores dealt with run very little over 2 per cent., so that a recovery of 85 per cent. on three successive treatments means only a loss of 0.3 per cent. of copper, a vastly different return to 85 per cent. on a 20 per cent. ore in a single smelting, when the slags with the same percentage loss, as in the previous case, would be richer than the Mount Lyell ore. No article on Mount Lyell would be complete without attention being drawn to the unparalleled record of this mine. Had not scientific and technical skill been brought into action, the mine would have fitfully struggled on in a small way, but without any profit being made. The result of management has been that £917,511 19s. 8d. has been paid in dividends since March, 1897. The mine has been opened up, and a magnificent plant and railway paid for. Unfortunately, the grade of the ore has declined, but even this has brought out still more strongly the extending resources of the management, for while the cost of mining has necessarily gone up, as ore has to be won from lower levels, the cost of smelting and converting has come down from the low cost of 22s. 0.03d. per ton to the unbeaten record of 15s. 6.03d., and the total costs from £1 5s. 8.3d. to 19s. 8.21d. per ton. Contrast this with Dr. Peters' estimate, an estimate certainly understated on the methods he recommended, at £1 16s. per ton. This last difference is due to management, and management alone. When one contrasts the judicious expenditure of shareholders' money and the conservation of their interests at this mine with the reckless prodigality and the blundering bungles on the comparatively rich North Mount Lyell mine, it is surprising that


ORE BIN
BRIQUETTES
SILICA FLUX barren
LIMESTONE
MT LYELL ORE
PURCHASED ORE
SLAG
SCALES
Air 300°F
25000 Cft per Min
Press 36 ozs and over
PYRITIC SMELTING IN Nº2 SMELTING PLANT 2 furnaces
No1 FOREHEARTH
Nº2 FOREHEARTH SAFETY
Slag
MATTE POT tapped 3 to 10 min
SAND MOULDS 12% MATTE
tapped once per hr
granulated slag
Water jet
From Nº1 to Nº2 Plant
BINS
COKE
FLUX
BROKEN MATTE
LIMESTONE
PYRITES
PURCHASED ORE
SLAG
PUMP
Slag Waste
BRIQUETTES
SCALES
Cold blast
Press. 36 ozs and over
PYRITIC SMELTING IN Nº1 SMELTING PLANT 2 furnaces
dust
Nº1 FOREHEARTH
Nº2 FOREHEARTH SAFETY
Slag
Water jet
granulated slag
FLUES
5% Clay
MIXER
BRICK MACHINE
Chamber dust
MATTE POT tapped 5 to 10 minutes
45%
45%
MATTE POT tapped once per hour
PUMP
Slag to waste
SAND MOULDS 45% MATTE
BROKEN MATTE BINS
COKE
Wood
MATTE
OCCASIONAL PURCHASED ORES
SCALES
Cold blast
REMELTING FURNACE
Part
White Metal
MATTE SLAG
SiO2 lining
SO2
CONVERTER
Part
Slag
Slag
Air
MOULD
TANK
POT
Bar trimmings
DRILL PRESSES
COPPER 98 85
SILVER 60 - 100 ozs per ton
GOLD 2 ozs per ton

shareholders in the latter had not their eyes fully opened to their only hope of salvation.

In concluding this sketch of the Mount Lyell mine, I must express my acknowledgments to the general manager, Mr. Robt. Sticht, and his staff. No question was left unanswered, while every facility was given for acquiring information at the mine and works. Much of the information given has been taken from a paper written by the company's officers, entitled "Operations of the Mount Lyell Company for the Year 1901-2."

The North Mount Lyell.

The North Lyell mine is about three-quarters of a mile along the Lyell ridge, north of the Mount Lyell. The isolated knob is as characteristic as the famous Crotty's iron blow. The ore from this mine is quite familiar to me, for years ago, before the present company was floated, Mr. J. P. Madden sent down many parcels of ore, and since that time he has very kindly presented the School (Bairnsdale School of Mines) with many bags of typical ore. The grade of the ore, its character and methods for profitable treatment, are therefore not matters of mere conjecture on my part.

The English company was floated in 1897. Out of 500,000 shares 395,000 were held by the vendors in full payment of the purchase money, leaving 105,000 available for working capital. Of these 45,000 were subscribed for by the directors and their friends, 35,000 were reserved for future issue, and 25,000 offered to the British public. In other words, the placing of one-twentieth of the mine in this manner constituted a successful float. After the flotation a rich deposit of bornite was struck in some surface excavations, and the value of the property enormously enhanced. The valuation of the ore was published in 1897 at a mere million pounds sterling, while only twelve months afterwards the chairman of directors stated that a safe estimate to be £5,726,387. Such colossal figures stimulated the North Lyell directors to enter upon an independent career, and apparently to endeavor to outrival the Mount Lyell Co. The result has been most disastrous. More than £20,000 was spent on a special steamer, which was sold soon afterwards, more than £30,000 on useless furnaces, more than £10,000 on a useless concentrating plant, and more than a quarter of a million pounds on an unnecessary railway, wharf and jetty, and probably another one or two hundred thousand pounds just as wantonly and lavishly expended, and for which nothing is to be seen. A mine which was floated for so many shares and so little cash at its command must have been a good one to stand such extravagance. The area enclosed and worked is not large, only about 30 acres. The main ore deposits, as at Mount Lyell, are at or near the contact of the schists and conglomerates. The main ore body, which was struck on the surface, lies nearly in the centre of the property, and at an elevation of about 1800 feet above sea-level. This consists of quartzites, carrying bornite and some chalcocite. This ore body has now been surrounded by tunnels down as far as 330 feet, and various connections have been made right up to the surface, so that the contents and values are fairly accurately known. This body is from 110 to 200 feet long and from 50 to 110 feet in width. Mainly from this 120,000 tons of ore of an average value of 9 per cent. have been broken out and sold. A large amount has been removed from this by an open cut.

The contact schists are bodies of metalliferous schists or tongues, some 50 feet in length and 20 feet in width, which protrude from the contact rock into the schists. These are persistent as to depth, but have not been exploited to any extent so far.

The third deposit is known as the eastern ore body, and is practically all the company had to show when the mine was floated. This has been but little exploited, and appears to consist of crushed rock with boulders of rich ore interspersed through it. It may be a surface deposit of secondary origin. The work done on the mine consists of No. 1 tunnel, which was driven N.E. for 400 feet, passing through the eastern ore body at about 150 feet. A crosscut was put in west, then turned south-west, and finally in the opposite direction to that which it started at, in order to reach the rich ore body. No. 2 tunnel was started 330 feet below the outcrop, and driven for 860 feet in a north-westerly direction, when the main ore body was cut. By a circumferenting drive, the area was found to be lenticular in shape, 150 feet being the length and 110 feet the greatest width. Stoping has been carried on from this level as well as that above. A winze was sunk in country rock from this level, and was down 110 feet at the date of my visit, but I did not have an opportunity of going down this. The ore struck at this level is reported to be poorer than that above. The removal of such large bodies of ore is not an easy task. A wide stope is taken out, pigsty frames are put in, and the space filled with waste rock from the surface. Fortunately the country stands well, and allows of large roofs of unsupported rock. From the open cut a great deal of the rich ore sold was extracted down to 100 feet in depth. The bottom and sides of the unworked portion do not seem to be as rich as the ore underground. Certainly nothing like the original values. It is a comparatively easy task to fix the grade and quantity of ore available above No. 2 level. So much has been extracted and sold that the block has been very well sampled. Probably about 200,000 tons of six per cent. ore might be obtained. Mr. Muir says certainly 100,000 tons of 10 per cent. ore. It is quite evident the millions of tons mentioned in the sanguine annual reports have vanished into thin air. I am indebted to Mr. Muir, the mine manager, for showing me through the mine. It is gratifying to note that there is no evidence of wasteful expenditure on this one essential part of the company's property. It is ancient history now of how Mr. Muir protested against much of what he was ordered to do, but in vain. According to official returns, the ore sold was as follows:—

1900	1,782 tons	...	22.46 per cent. copper.
	9,461 tons	...	13.88 per cent. copper.
1901	12,494 tons	...	17.33 per cent. copper.
	44,249 tons	...	7.53 per cent. copper.
1902	17,970 tons	...	6.5 per cent. copper.
	1,698 tons	...	15. per cent. copper.
	11,000 tons	...	8. per cent. copper.

The bulk of the ore went to Mount Lyell; the balance was sent to England. The latter had to be bagged, packed on horses over a fearful road, handled and rehandled, and shipped to England. The profits must have been most meagre. In other words, nearly £600,000 worth of copper has been taken out of the mine, and the company has not started yet. In 1900 Mr. J. S. MacArthur, one of the original directors, supplied a report on the property, and the method of dealing with the ore. Mr. MacArthur recommended smelting in large reverberatory furnaces, and the direct conversion of the matte produced by bessemerising. This report was no doubt based on inflated ore values, also on the published records of the work done with similar furnaces in America. But Mr. MacArthur does not seem to have considered the extra cost of fuel in Tasmania, or the extra amount required to melt such slags as he proposed. Mr. L. C. Trent was appointed general manager, and proceeded to carry out Mr. MacArthur's scheme. A site was selected at Crotty, over eleven miles from the mine, on the King River. Four great reverberatory furnaces, each 36 feet by 15 feet, with a hearth four feet deep, were erected. Each furnace had its own huge stack and fireplace for consuming a maximum of fuel for a minimum of heat. With coke at 5s. instead of £2 per ton, there might be some hope of success with such reverberatory furnaces, provided they had a freely flowing slag. But at Crotty, about 200 tons of coal were consumed per week, and a small amount of matte was produced at a fabulous cost. Mr. Trent then demanded richer ore from the mine, but to supply this a large amount of seconds would have had to be stacked. To get over this a concentrating plant, costing about £12,000, was erected. Now, of all ores and gangue treated by concentration this was about the least suitable. The brittleness and low specific gravity of the ore, the hardness and toughness of the gangue, should have shown the merest novice that trouble would occur in crushing, and further trouble in concentrating. The plant erected cannot be condemned as a concentrating plant, but as one wholly unsuitable for North Lyell ore. And the folly of directors allowing such a plant to be erected can only be excused on the grounds of sheer ignorance.

The design of the plant is on the catalogue system of Fraser and Chalmers, and consists of a grizzley which screens out the fine ores, a breaker which crushes the coarse. The crushed ore passes to an elevator. After crushing finer in a Blake crusher the fines are passed through a grizzley, the coarse going on to a sorting table. Thence the material is elevated to a storage bin, and passes to a trommel. The coarse ore from the trommel goes to a set of rolls, the fines to a pit. The ore from the rolls goes to a topping jig, thence to fine rolls and to a series of three trommels followed by corresponding jigs. The fines from the pit are also delivered into the same series of trommels and jigs. The tails from the jigs pass into two Chilian mills for further trituration. The product from this mill is passed through a spitzkasten, the sands going to a table of the Wilfley type, the slimes going to 18 Frue vanners, each having belts 6 feet in width. Mr. H. Mackay has charge of this plant. I am much indebted to him for his courtesy and hospitality. The return obtained by this method of treatment, I was informed, was 65 per cent., which was very good considering the nature of the ore. On a low grade ore by rushing a large amount through even with a waste like this, the operation may be commercially the most profitable. The rolls used wear out very quickly, but for such soft mineral no other pulveriser could very well be applied. The attempt to slime everything at the finish in order to get good work out of the vanners shows a woeful state of affairs. Mr. Trent plunged more deeply when he found that smelting would not do, for suspecting huge losses with a concentrating plant, he recommended the addition of a lixiviating plant. He could not decide on what system to adopt—fortunately for the shareholders. The absurdity of sliming nearly all his material for the Frue vanners makes it appear that a large portion of the plant was erected for the installation of these machines, and the suggested after treatment by wet method shows that the directors must have had more faith in their experts than fitness for their positions. After the failure of these schemes, Mr. Trent departed. The next move was to erect blast furnaces, and start smelting on ordinary lines. Thanks to the skill of Mr. H. H. Lewis, the company was given the first glimmer of hope, since the so-called metallurgical work started. Four blast furnaces were erected by Mr. Lewis. Each of these is 9 feet by 3 feet, by 12 feet in height, and provided with six tuyeres, back and front. The furnaces are steel jacketed. The blast pressure is about 12oz., and the charges are so adjusted that a high grade matte containing 45 per cent. of copper is made in one operation. The slags are granulated by a jet of water and run to waste. At the date of my visit, all four furnaces were running smoothly, and turning out 56 tons of matte, or a total of 100 tons of copper per week. The slags were running on the average:—Silica (SiO_2), 42.0; alumina (Al_2O_3), 7.0; ferrous oxide (FeO), 23.0; lime (CaO), 23.0; magnesia (Mgo), 3.; copper, 0.4. Unfortunately, North Lyell ore does not contain enough sulphur or iron for matte, and also contains a high percentage of silica, thus necessitating the addition of a large amount of fluxes. The general composition of the good ore may be stated:—Silica (SiO_2), 71; alumina (Al_2O_3), 9: iron, 4.3: copper, 9.2; sulphur, 6.5.

In order to supply the iron and sulphur required for a 45 per cent. copper matte pyrites from the South Lyell mine is fed in with the ores and fluxes. I was shown over this mine by Mr. J. Ryan, the popular manager, and also supplied with much information about the field. The South Mount Lyell mine adjoins the Mount Lyell. A shaft was sunk in the former, and a body of pyrites struck at a depth of 505 feet, and passed through at 630 feet. This mass is lenticular in shape, being about 230 feet long by 93 feet wide at the centre. It is similar in composition to the low grade pyrites of Mount Lyell, but contains less gold. By itself, so far as proved, it is not payable. The floor of this deposit is about on a level with No. 8 level of Mount Lyell, but it does not seem as if there was any direct connection between them. It is clear that this property should be further prospected, preferably with the diamond drill, of which too little use has been made on this field. One ton of South Lyell pyrites are used with two tons of North Lyell ore, just the reverse of pyritic smelting. Part of the sulphur distils off, and burns in this case as with Mount Lyell pyrites, most of the remainder going to form matte. The iron-stone necessary has to be brought all the way from Zeehan, and costs 20s. per ton. A cheaper local material was used, but the extra cost of that from the Manganese Hill at Zeehan is more than counter-balanced by the better driving of the furnace. The limestone is obtained from a quarry along the railway line. A system of electrical transmission is used throughout. A compound engine, 250 horse-power, drives the dynamo, which supplies the current to motors which drive the concentrating plant, motors which drive two small Baker's and two large Roots' blowers. This magnificent plant is under the charge of the electrical engineer, Mr. J. H. Newby. Everything works with the greatest smoothness, and the confidence in the plant is complete when motors driving the blowers are not even duplicated.

Whatever faults might be found with the system of treatment adopted, none of this reflects in the slightest degree on the staff at the mine. They are zealous and eager to obtain the best possible results, and, under favorable conditions, the work done would bring them into prominence. As it is, shareholders are clamoring for profits, and any work which does not bring dividends is condemned, while even bad work on a good mine will give a manager undeserved credit. Taking a wider view of the position, it is clear that the task set the present expert staff is to show how copper or matte may be produced at Crotty for more than it is worth. I cannot pretend, and do not pretend, to judge the whole value of this mine, and the work, from a rapid examination, but the position is made clear by the amount of material taken out of the upper levels, and the manner in which it was taken out. Ores containing six per cent. of copper may pay to concentrate even if 50 per cent. of the values are obtained, but such siliceous ores will never pay to smelt at Crotty under existing conditions. The final outlook for shareholders from this point of view is a hopeless one.

It is quite possible, and I believe feasible, that many of the low grade siliceous ores could be dealt with successfully by wet methods. Mr. MacArthur, during his last visit, erected a small muffle furnace, and did some experimental work on ton parcels of North Lyell ores. Mr. Mackay informed me that with salt roasting a perfect extraction was obtained. I can also state from experiments carried on at our school on similar ores that good extractions may be obtained at far less cost than by smelting. This work, which should have been tried first, instead of last, inspired the directors with no confidence. They had been so long chasing the spot where the rainbow rested that this was regarded as another ignis fatuus.

The vital question of amalgamation of the two companies, which is now on the point of fortunate consummation, awaiting, as it does, only the ratification of the shareholders interested, has been raised since the commencement of this series of articles. On this subject, before visiting both mines, I could not form any opinion, while on the field the matter was only mentioned by a few who were opposed to it. On due consideration of the whole question, only one conclusion can be arrived at, and that is in favor of amalgamation. In our younger days, we all heard of the fable of the cat and the fox; of how the cat had only one way of escape, but the fox had a thousand, yet when the critical time came the cat, by the help of her single shift, ran up a tree, and sat securely among the top branches, from whence she beheld her neighbor, reynard, who had not been able to get out of sight, overtaken with his thousand tricks, and torn into as many pieces by the dogs which had surrounded him. While this fable may not be exactly apposite to the Mount Lyell and North Mount Lyell companies, yet the clear road to success was followed with unerring singleness of aim in the one case, and many uncertain tracks were dodged about on the other, all leading to failure. Had either company a clear future before it, there would be no amalgamation; absorption would take place in that case. If the Mount Lyell costs come down at the same rate as they have done during the past year or two, and even small developments take place, it is quite possible that many years may elapse before work ceases to be absolutely profitable. On the other hand, if the North Lyell costs of production of copper were reduced to anything like those at Mount Lyell, large profits would be made, and there would be no necessity to amalgamate. The ore in both mines has declined in value at a depth. That at the North Lyell is three times as rich in copper as the Mount Lyell, but as it cannot be worked at a profit, while the latter can, the balance is in favor of Mount Lyell. On the same lines, a reef 10 feet wide, going 1oz. of gold per ton, may be more profitable than a 1-inch reef going 10oz. per ton. At present two per cent. ore can be mined and worked at a profit at Mount Lyell, while it takes at least eight per cent. at present to pay the cost of smelting and realisation at North Lyell; and even in a limited time, say, two years, the supply of this ore would be exhausted and not a farthing would be given back to shareholders. It is evident, therefore, that the present smelting methods must be susperseded by something simpler. Assuming the North Lyell goes in for pyritic smelting, then it would mean that from two to three tons of South Lyell pyrites would have to be mixed with one ton of North Lyell ore. But the South Lyell ore has to be stoped and mined, and transported by rail. The total cost of this would not be less than 18s. per ton, while its value in copper would only be equal to the amount lost in the slags. The mining and transport of North Lyell ore would not cost less than 18s. per ton, so that the ore and pyritic flux delivered at the furnaces would cost not less than 60s. per ton of North Lyell ore smelted. Assuming that the work from this point can be carried on as cheaply as at Mount Lyell at present, say, 15s. 6d. per ton, the cost would be on the pyritic ore—for comparison, say, $2\frac{1}{2}$ tons—or about 40s. for every ton of North Lyell ore. The bare costs would therefore be 100s. per ton. Now, with copper at £60 per ton, one per cent., or one unit, is worth £$\frac{60}{100}$, or 12s. Dividing this number into 100, the profitable percentage would be given as 8.3. The gold and silver values are small, and are neglected in this calculation. Before a plant capable of doing such work is erected the company would have to find some hundreds of thousands of pounds, and even then would be no better off than at present. On the other hand, assuming the ore is dealt with by the Mount Lyell smelters, and assuming that the Mount Lyell pyrites fed in will just clear all expenses of mining and smelting—this, by the way, can be done for a great deal longer time than by working for profit, quarrying barren fluxes and feeding those in—then the only charge against the North Lyell ore will be that of mining and transport. Assuming this to be done for 15s. per ton, it is easy enough to see that profit may be made on any North Lyell ore carrying over $1\frac{1}{2}$ per cent. of copper, or, allowing for losses and extra costs, two per cent. In other words, were 5 per cent. ore sent down the profit made would be 36s. per ton, with copper at £60; 6 per cent. would give 48s. per ton; and 10 per cent. would give 96s. per ton profit. This wonderful result is due to the fact, readily enough seen by those who read the preceding articles, that the siliceous ores in themselves form a perfect fluxing mixture with the pyritic ores of Lyell. No expensive carbonaceous fuel is required; no addition, except in very small quantities, of any outside fluxes.

Since the foregoing was written, the two companies have amalgamated. The Crotty works have been dismantled, and the whole of the work is carried on at Mt. Lyell, under the control of Mr. Sticht.

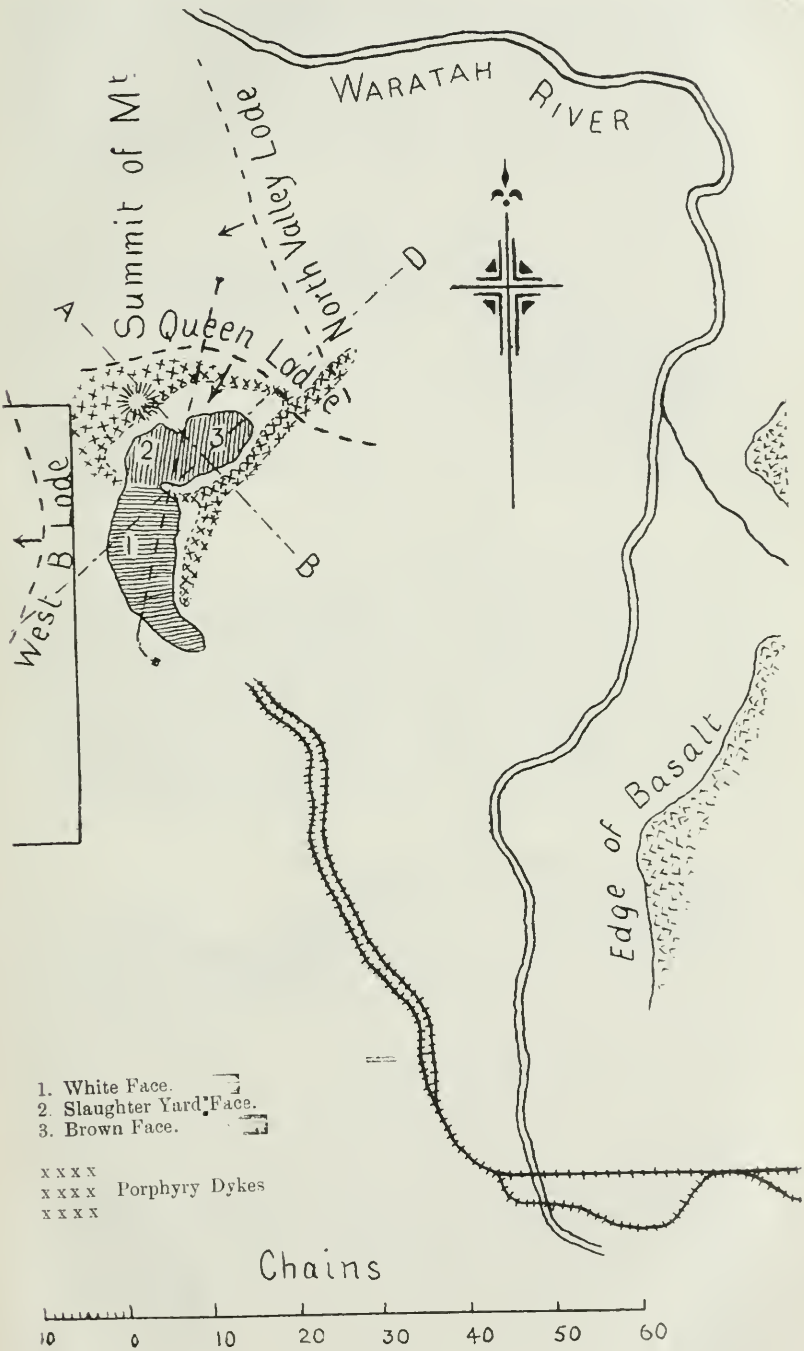
Mount Bischoff.

There is no need to more than indicate the oft-told tale of the discovery of Mount Bischoff as a tin mine. Mr. James Smith, better known as Philosopher Smith, prospecting alone in the heart of the hills, discovered a black heavy mineral. This was tested by Dr. Walker at Table Cape. The doctor smelted it in a crucible, a shining white metal was obtained, but the sanguine expectations of the onlookers were quenched on being told that the metal was not silver. The smelter placed the button between his teeth, heard the crack of the crystals across each other, and pronounced the metal to be tin. This was on the 4th December, 1871, or over 30 years ago. Since that time the Philosopher saw over £2,000,000 worth of tin extracted from the mine and over £1,000,000 paid in dividends, yet by the irony of fate never received one. Mount Bischoff itself was discovered before the Philosopher panned off the tinstone, for the surveyors, more than a generation before, had actually planted a trig. station there.

In 1872 the Philosopher and Mr. Crosby, who subsequently became the first manager, with a party of workmen, opened up a track to the mine, which is about 50 miles distant from Burnie, and in December, 1872, mining was started. In 1873 the present company was formed in 12,000 shares of £5 each, £1500, and 4400 paid-up shares going to the Philosopher for his two 80-acre sections. It may be said in passing that those two sections included practically everything that was worth getting. One pound per share was called up, it being considered that £15,000 would be more than ample to open up the mine and put it on the dividend list. Even after this lapse of time the balance-sheet of the company states: 12,500 shares at £5 per share less uncalled 7600 shares at £4 per share, so that Philosopher Smith's shares are the only paid-ups. The hopeful shareholders forgot about the troubles of transport, the terrible roads, and the everlasting rains. Little wonder was it that Mr. Crosby's health failed, and Mr. H. W. Ferd. Kayser took his place on the 16th November, 1875. The masterful methods of the new manager, his extensive grasp of the whole situation, and the vigorous policy he recommended puzzled some and alarmed others. His work was severely and adversely criticised, and it is said that at this darkest and most trying period the Philosopher cut himself adrift from the company, and sacrificed his scrip. Mr. Kayser's work proved to be right; the mine developed in a most marvellous way, the situation was saved, and Mr. Kayser's reputation in tin has even since been held by the shareholders on a higher plane than Bischoff itself. It was not until February, 1878, that the first dividend of £1 per share was paid, while over £100,000 was spent before it could be declared. The banks were evidently good in those days.

The mount itself is 2598 feet in height, but is only a few hundred feet above the table land at its south-eastern extremity, on which the township of Waratah is situated. The latter place is connected with the Emu Bay railway, while the company's steam tram line runs from the terminus of the line right up to the mine. On the east, north and west sides of the mount the slope is steep. The workings are on the south-eastern slope of the mount, and a few hundred feet below the summit. The mount itself is composed of porphyry, as are a number of dykes which branch out from it, and which may be traced for considerable distances. Adjoining this rock are a series of sedimentary rocks, having a general strike from N. 15 E. to N. 45 E. These slates are much indurated and near the contact are transformed into porcellanite. The workings at Mount Bischoff include a series of open cuts or quarries, extending for about half a mile in length, and from five to ten chains in width. A vast amount of material has been removed, and it is somewhat difficult to obtain a clear conception of the nature of the deposit. In the following notes, in addition to any observations made, information is embodied supplied by the general manager, Mr. Kayser; Mr. Jones, the Government Surveyor; Mr. Sheedy, the surface manager; and Mr. Bradford, who is employed on the mine, and who has written a paper on the mount for the Australian Institute of Mining Engineers.

The upper portion of the deposit was an alluvial one, and some of the adjoining claims furnished rich tin ore from very shallow ground. It is not easy to understand, with the present configuration of the country, how a river or stream could have run over Mount Bischoff, but when one considers that the basalt which flowed in such extensive sheets over this part of Tasmania must have filled up all the existing creeks and gullies, and made a tableland of all the country it passed over, then it will be readily understood that the Waratah River as such did not exist, but that it is since those times that the present stream has carved out its channel. The river that ran even in those times must have passed the mount, and its bed has been plugged up with basalt, so that there should be a deep lead under this rock. This, up to the present, has never been satisfactorily explored. Practically the whole of the alluvial deposits have been removed from the mount. One face now being worked at the northern end is about 50 yards long, and has a face about 50 feet high. This, however, will soon cut out as work is carried on towards the slope of the hill. The material consists of boulders, clay and tin oxide, some of the lumps of the latter being up to a hundredweight in weight. In most cases the boulders forming the wash are stanniferous, and the whole lot goes through the battery. A rather interesting feature of this formation is a small patch of lignite, while just below it is a corresponding patch of pyrites, formed no doubt by the reducing action of the organic matter on solutions of ferrous sulphate. The pyritic matter is most carefully relegated to the dump. For seven years from the starting of the mine only such alluvial deposits were worked. The formations underlying the alluvial are of great interest, and it is a matter of regret that they have not been worked out in detail. Starting from the south end of the workings an alluvial deposit, known as the White face, has been worked out for a distance of 300 yards across and 200 yards north and south. On crossing a porphyritic dyke the Slaughter-yard face is reached, then the Brown face. These two constituted the main sources of supply for many years past. The workings show that this ore body is funnel shaped, the funnel itself being surrounded by a dyke which has emanated from the main mass of porphyry at the mount. There is a layer of slate of varying thickness between the dyke and the sides of the funnel. Through the Brown face deposit


WARATAH RIVER
Summit of Mt
North Valley Lode
Queen Lode
West B Lode
A
B
D
1
2
3
Edge of Basalt
1. White Face.
2. Slaughter Yard Face.
3. Brown Face.
x x x x
x x x x Porphyry Dykes
x x x x
Chains
10 0 10 20 30 40 50 60

run decomposing stanniferous dykes, which appear to be very basic. These, no doubt, have been charged with pyrites. These basic dykes do not seem to be connected with the main mass of porphyritic material. If they are older then it would seem as if they came from below highly charged with tin stone, while at a later period the more acid and less stanniferous dykes were forced upwards. The Brown face deposit measured about 700 feet by 400 feet, with an angle slope of about 55 degrees. Besides the decomposed dykes

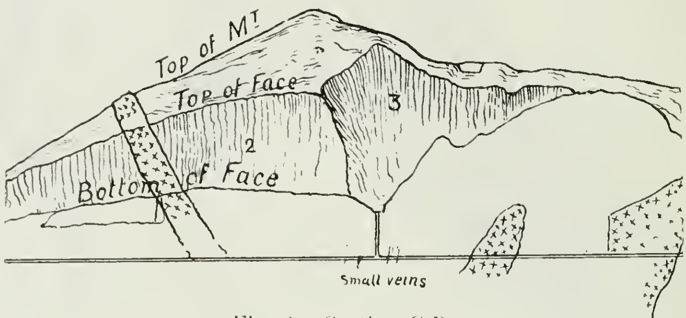


Fig. 2.—Section C.D.

already mentioned this funnel is filled with various minerals arranged in horizontal layers. These minerals are for the most part loosely arranged, and may be dug out; in fact, the whole appears in section very must like a sandy drift arranged in layers. On examination a layer of fine quartz crystals, no larger in diameter than a pin, and each perfectly crystallised, may be seen; then layers of black and green tourmaline; in other places layers of talc, overlying layers of jet black crystallised cassiterite, in some places so thick that they could be shovelled up almost pure.

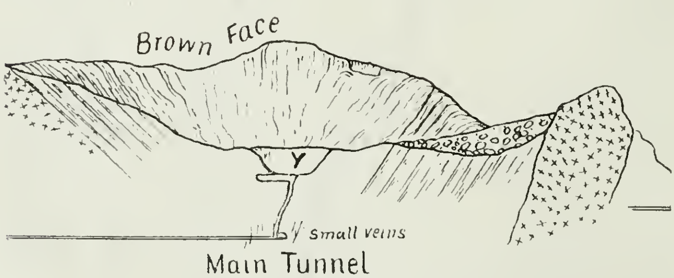


Fig. 3.—Section A.B.

The tin crystals in the whole mine are usually small tetragonal pyramids, about one-eighth of an inch in diameter. These are sometimes isolated, but more commonly appear on the surface or in cavities of nuggetty masses of oxide. Some pockets in this drift-like material were extraordinarily rich, one of 150 cubic yards, yielding £60,000 worth of tin, or at the rate of £400 per cubic yard.

Just before leaving the mine I was shown a thin seam of anthracitic looking coal on the hanging-wall of one of the basic dykes referred to. Extensive deposits of sulphur were also obtained from the Slaughter-yard deposit. In addition to this most unique deposit, which will be discussed later on, there are two well-defined lodes on the property. The Queen lode lies to the north of the Brown face; has a course east and west and underlies flatly to the south. This lode is a true fissure, having well-defined walls, and cutting through both slate and porphyry. Its width varied from one to two feet, and it was worked for a length of 700 feet for 15 to 20 per cent. of tin. In places there was a solid vein of cassiterite a foot in thickness. The dressed tin stone from this lode is eminently adapted for beds for the jigging sieves, since it is found to wear better than any other variety. Another lode on the north, known as the North Valley lode, is from one to five feet in width, and runs in a north-westerly direction, with an underlie to the west. It traverses slate country. This lode is rich in tin ore, but carries pyrrhotite, arsenical and iron pyrites. Owing to this fact it is not at present being worked. On the west side of the mount, near Tinstone Creek, where the Philosopher first got his samples of tin, lies the West Bischoff lode. This has been traced for more than half a mile, running in a northerly direction,

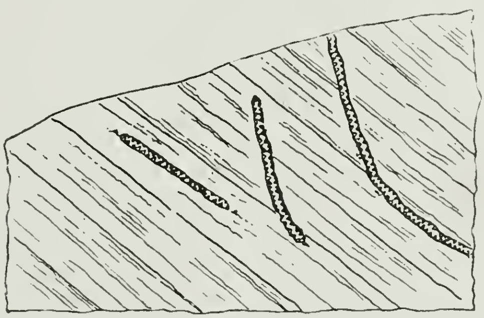


Fig. 4.—Seams of Cassiterite in Slate.

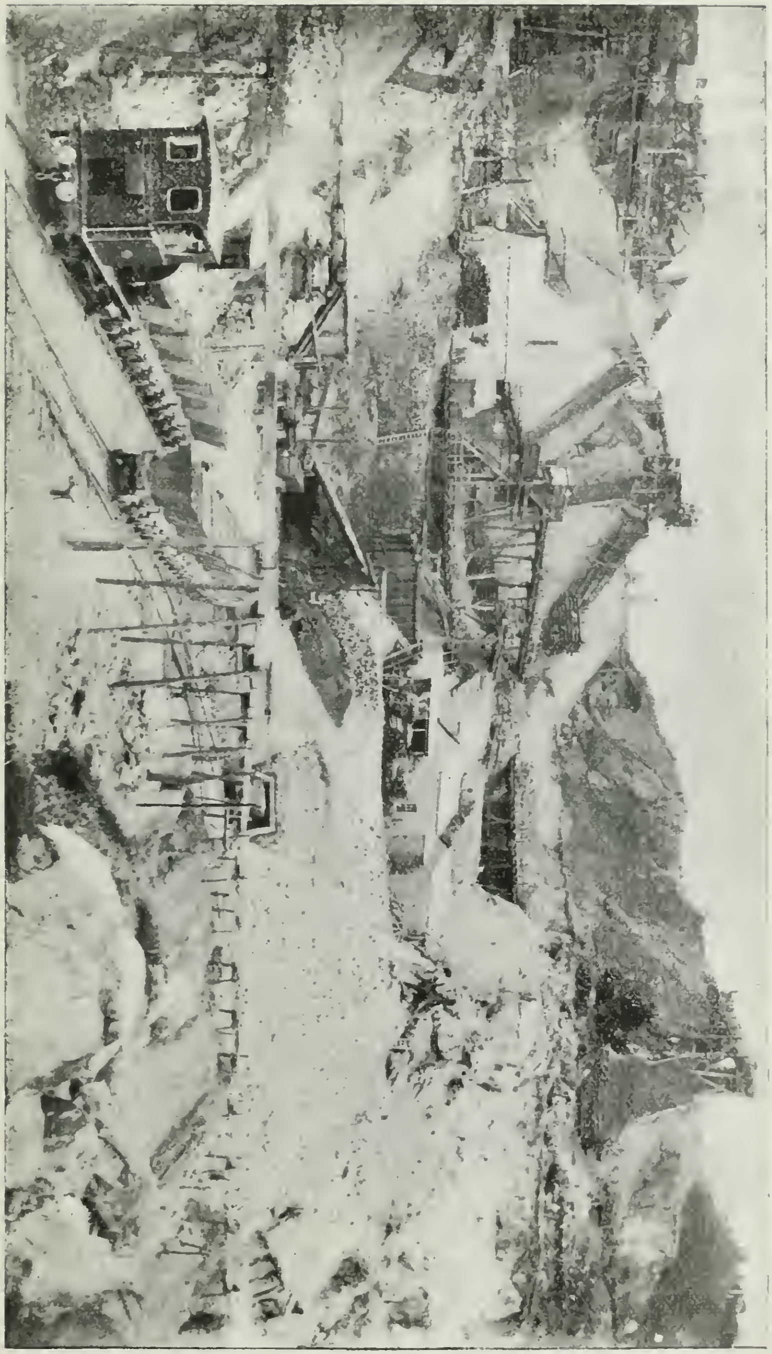
and underlying to the west. From the upper levels splendid yields were obtained, but the presence of pyrites frightened people. This lode is now being worked by the West Bischoff Tin Mining Co. While it is not safe to speculate too much upon the origin of such ores, yet if a correct explanation or true theory of their occurrence is propounded, it may be of service in guiding working operations, and be of importance when dealing with new deposits. In the first place a coarse grained rock, such as Mount Bischoff is largely composed of, is held to have been formed under great pressure, so that there must have been hundreds of feet of overlying rocks, probably similar to the slates and sandstones that surround it now. The various rock minerals crystallised out under great pressure and at a very high temperature. The heat applied was not a dry heat, but the material was mixed with vapors of metallic compounds as well as water vapor. These vapors were prevented from escaping by the enclosing magma, which in its turn was kept under by the overlying rocks. In places the overlying strata gave way, and sheets of igneous rock would protrude. Sometimes the surface rock would fissure, and through this fissure would be forced in a few moments the liquid rock from below. This material, in cooling so rapidly, would differ from the main mass in that the various crystals would not have time to separate out. It is probable that under conditions of high temperature and great pressure the tin would exist as to fluoride, and on a decrease of temperature and pressure would react with water to form new compounds; in other words, stannic oxyfluoride and water would produce stannic oxide and hydrofluoric acid.

$$Sn_2 F_4 O_2 + 2H_2O = 2SnO_2 + 4HF.$$

The hydrofluoric acid thus liberated, in its turn might attack silicates, converting orthoclase felspar, if iron were present, into tourmaline; in the presence of lime it would form fluor spar; while any silicate of alumina, such as exists in clay or slates, would be transformed into topaz. Another element which is invariably associated with tin ore is boron. It is likely that it also plays an important part in bringing about the solution and deposition of stannic oxide. At Mount Bischoff there is abundance of the green and black tourmaline, while beautiful radiating bunches of topaz crystals may be obtained from the southern end of the workings. I was also shown a sample of fluor spar.

It would thus appear that the present mountain was formed under a mass of slate or some other overlying rock that it is possible for sheets of intrusive rock, even at that time to have reached the surface, but that all the rock at present showing was a great distance below it. It is plain even now that many of those intrusive masses never reached a level as high as the present surface, for dykes have been struck by a low level tunnel, which do not appear on the surface at all. It also appears to me that the ring-like dyke, enclosing the Brown face deposit, had reached a point below the surface, which admitted of ready fracture, and that the various tin laden vapors had free access to the bulk of the material in the funnel-shaped opening through which they were escaping. This material was to a great extent redissolved and crystallised or altered and crystallised. On the eastern side of the dyke, where a short tunnel has been put it, some very fine seams of excellently-crystallised cassiterite stand out prominently. These would represent the deposits left by the vapors or solutions escaping readily through the slates. The fact that the Queen lode cuts through both slate and porphyry makes it evident that fissures formed subsequent to the main porphyritic intrusion were filled in the same manner. In such a case as this the time that elapsed between the first heating and the final cooling down may have been equal to or greater than that which takes place even now in our modern volcanoes. A few centuries is but a few days for geologic action.

Subsequently the old fissured and cracked surface was the weakest and most easily worn surface. The very mountain top became the bed of a stream in bygone years. As this wore its way down in successive ages it actually flowed over almost the whole length of the famous deposits, thus sluicing and wearing away the lighter materials and allowing the heavier cassiterite to drop down. The outlet of this stream has already been indicated. Thus it was that the mine which was worked for seven years as an alluvial mine, and from which masses of almost pure oxide, weighing as much as a ton, were obtained, was situated right over the top of perhaps the richest, and most easily won, lode ore in the world. The



The White Face.—Ore Shoots and Bins for Loading the Trucks.

ground is opened up by stripping off the peaty surface. The water-worn boulders which underlie this are usually stanniferous, so they go to the battery; the clayey material is sluiced away, and the semi-washed product sent to the mill. In cases it is sent direct. The ground is then opened up on the benching system up to the walls, but as these have a slope steeper than the angle of repose the ore is knocked down. To adopt the benching system here would be like building a staircase without nailing it to any supports. There would be a danger of the whole lot crumpling up at any time. Bars are driven in above, ropes are hung over to the men, so that in case of any slip in the ground they are always available. Men pick down large quantities of the fairly loose material, which runs down to the bottom of the bench. It is there trucked, hauled up an incline, and sent down to the stonebreakers should this be necessary. It is not apparent why the low tunnel beneath this level should not have been made use of. In the Brown face and the Slaughter-yard face the material is so soft that almost the whole of it may be shovelled. The decomposed gossany dykes are also soft, while the cassiterite is coarse, gritty and crystalline. The hard white dykes enclosing the ground are being worked at the southern and western extremities, but, judging from the comparatively small amount of material removed from them as compared with the huge amounts taken from the softer material, it is evident they were rather looked upon as diluting agencies. Prospecting tunnels have been put in. One has been driven longitudinally under the main workings. This has only revealed small veins of pyritous tinstone. The western crosscut, driven under the summit of the mount, revealed nothing of importance, the porphyry, of which the mass of the mount is made up, proving barren where cut through.

The Queen lode will soon produce pyritic materials with the tinstone, while the North Valley lode is unworked because the pyrites are there. From all this it is quite evident that so long as the soft, easily-worked formations last, the yields can be equalised, but when these are done, and that will not be very long, a much more difficult problem will be presented, and one which is of vast importance to the whole of Tasmania and of vital importance to Mount Bischoff. That is the treatment of hard rock carrying only a small percentage of tin and a cheap, effective treatment of tin stone carrying pyrites.

The diagrams given to illustrate this article are approximately to scale.

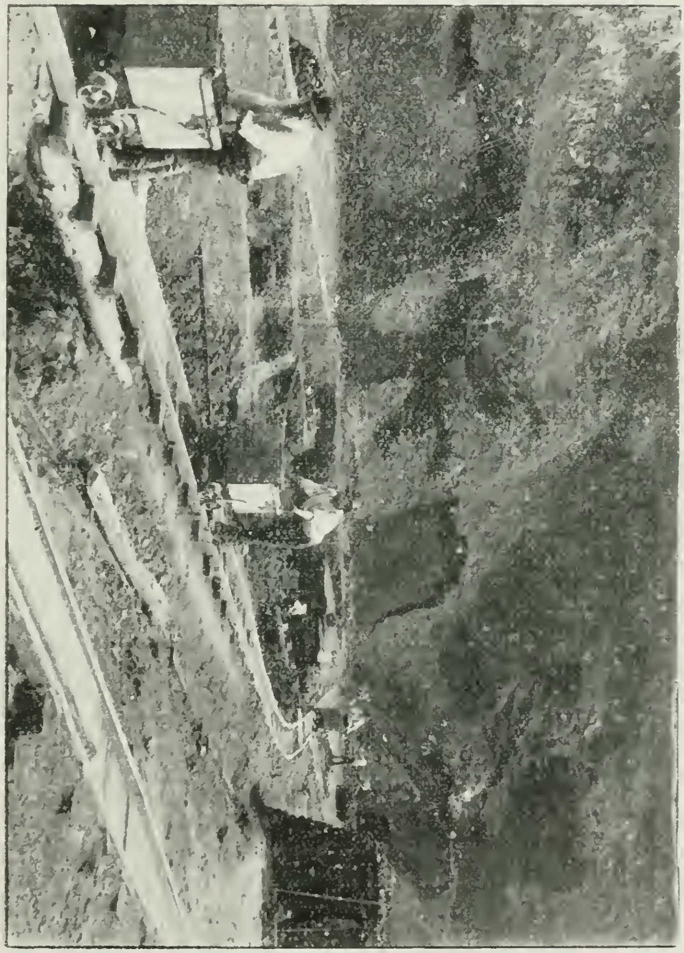
No. 1 shows a plan of the Mount Bischoff workings, the Queen lode and the North Valley lode, as well as the West Bischoff lode. The tramway from the mine to the works is also shown.

Fig. 2 gives a longitudinal section through the main workings. The whole of the ore towards the mount has been practically removed, and it is only the lower portions of the deposits 2 and 3 which remain.

Fig. 3 gives a section from the mount across the Brown face. The stoping sides of this appear much steeper than is shown in the figure. The lowest excavation or bench marked Y is no great distance above the main tunnel driven under the formations. The soft, rich formation, dies away before this is reached, so that the amount available is limited.

Fig. 4 shows the small seams of cassiterite in porcellanite as exposed in a small cut on the eastern side of the porphyry dyke.

Mt. Bischoff was worked for many years for alluvial tin. During that time, as may readily be understood, many tons of tinstone specimens were exposed. In December, 1876, a five head battery was started, and during that month 187 tons of tin oxide were produced. This amount was increased to 250 tons in August, 1877. With a mine developing so rapidly additional machinery became



The Brown Face.—Trucking Ore.

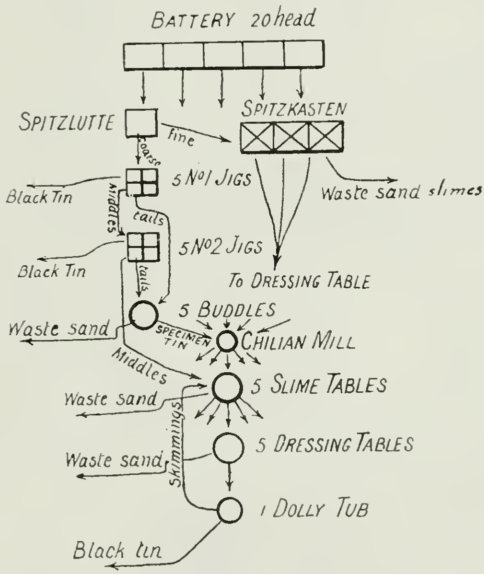
necessary, and another 15-head was added in 1879, while in the early eighties this was increased to 75-head. While there are many points which critics have been severe upon with regard to this plant, such criticisms have no weight when considered apart from local conditions. The weight of the stamps and the fact that there are plants of the same type as the main one, erected to deal with the sand which escapes from the first, do not seem to accord with our modern notions. The explanation given by the manager puts the matter in a very different light. The stampers fully weighted are really 8cwt., and this was certainly a full weight stamp twenty years ago. Owing to the shortness of water in the summer half shoes only are cast, so that the whole plant can be kept going. A great deal of the material is soft and is discharged by mere agitation. With regard to retreatment, it is explained that if the top sheds were doing all the work there would be no necessity to retreat the tailings, but owing to the water becoming thick, due to the presence of clayey matter and oxide of iron, a portion of the finer tin ore would float away if such treatment were repeated at the upper sheds. The water is used as often as possible, but it was not considered advisable to have finality of treatment here, but deemed a better plan to allow this escaping material to accumulate and re-treat it with clean water at the lower sheds. The nature of the ore also has to be carefully considered—probably the bulk of the material does not need much crushing at all, as delivered in the trucks it is a clayey or sandy material. A great deal of the gossany material from the Brown face is semi-soft. Some hard material in the shape of stanniferous boulders in the wash and from the Queen lode, and faces of hard porphyry give enough rock to facilitate the escape of the clayey material from the stamper boxes. The high duty per stamp, 4½ tons per 24 hours, is due to this, and also to the coarseness of the screens, which are woven wire 14 mesh.

The row of waterwheels is not looked upon with favor by the modern hydraulic engineer, but their existence is justified on the ground of simplicity and suitability, the quantity of water available being large and the fall comparatively small. The waste water from one wheel is made use of wherever possible to drive a wheel below.

While it is an easy matter to criticise severely and adversely, it is doubtful whether any other class of machinery would have dealt so successfully with the large body of ore, over six million tons, treated, and over sixty-one thousand tons of oxide won. So far as Australasian practice is concerned, there is no doubt that the Bischoff mill was far in advance of anything of its kind at the time of its erection. Since that time minor details have been perfected, and men who have grown up on the works have become so accustomed to each machine that the best possible work is got out of it.

So far as mere theoretical considerations go the winning of tin stone or cassiterite from its gangue should not be a difficult one, but even now, after centuries of experience, it is doubtful if as good a percentage extraction is obtained as with most other metals. We only know what is won; but what is lost is seldom if ever accurately known—though often put down, knowingly or unknowingly. Suppose an ore runs 1 per cent. of oxide: this amounts to 22.4lb. per ton. A return, let us say, of 16.8lb. is obtained, or 75 per cent. This leaves 5.6lb. in the tailings—which is only one-quarter, or .25 per cent. — which appears low when expressed this way. Should this amount exist as specimen tin, i.e., grains of sand with visible adhering grains of oxide, then the ordinary test of crushing and vanning will show a large proportion of it, but should it pass away as slime, then beyond a certain limit no expert will save it on his shovel. He may consider that because he gets exceedingly fine slime in a careful test that he can save any tin oxide, no matter how fine; but this is utterly fallacious. Now since no satisfactory method for assaying small quantities of tin in large quantities of sandy material is adopted at dressing establishments, any positive statement as to losses is not to be relied upon.

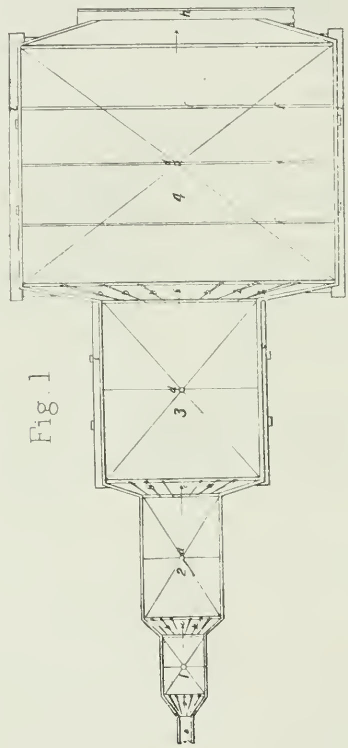
It is often stated that since the tailings from any particular mill cannot be profitably worked that it shows there was no loss in the original treatment. Of course it only shows that the method of treatment adopted the second time, or the knowledge of the user of it was not sufficient. Should a profitable return be obtained with even cruder appliances than those used the first time



Plant Diagram.

it shows there is something radically wrong in the design or working of the original plant.

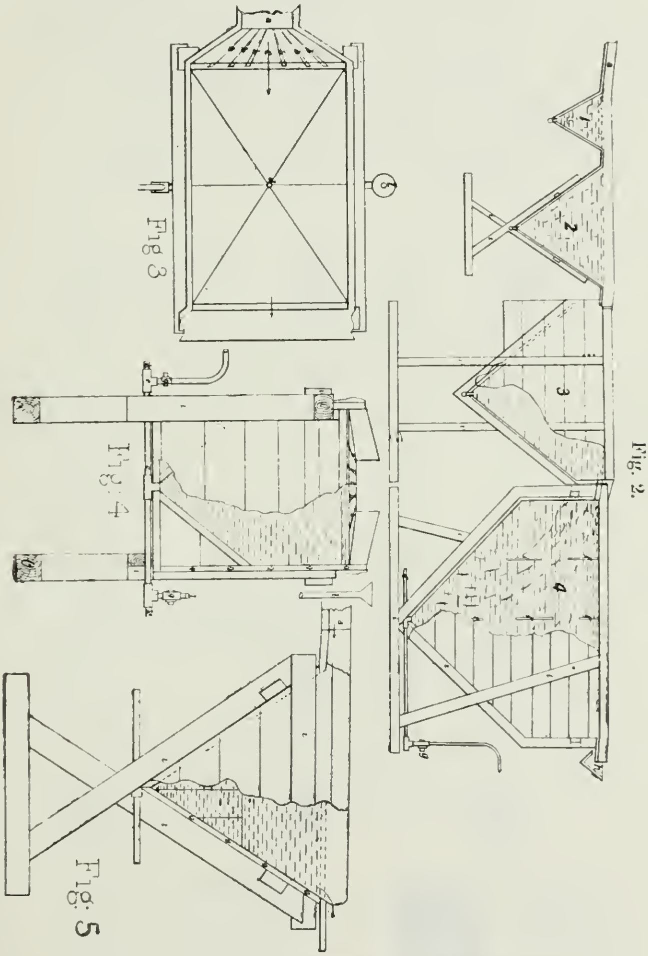
Cassiterite, or tin stone, as is well known, has a formula SnO_2, and when pure contains 78.6 of tin and 21.4 of oxygen. If pure it should be white or a light grey. The dark color of most of the specimens is due to oxide of iron. From a great number of assays quoted by the metallurgist for Mt. Bischoff it would appear that the results run from 72 to 73 per cent. of metallic tin—what the metal admixed with the tin is is not stated. The oxide has a high specific gravity, viz., 6.8 to 7.1, on account of which it may be concentrated so readily, but on the other hand it is as hard as quartz and much more brittle, so that in pulverising it, much is reduced to an impalpable powder or slime. Were the oxide disseminated in particles invisible to the naked eye through a hard rock it would be a very difficult matter to save even as much as 50 per cent. of it. When it is in coarse lumps, if those lumps can be freed from gangue without much pulverisation, most machines can deal effectively with it. The finer it becomes the more difficulty there is in saving it, so that a stage is at last reached at which it would be about as easy to pan off gold leaf from grit as tin slimes from sand. At Mt. Bischoff they have been fortunate in being able to save the greater portion of the oxide in a coarse state, and their efforts have been directed to saving as much as possible in a coarse condition, only when compelled to, sliming the sand to obtain the balance. One drawback to Bischoff dressing is



Kayser's Patent Ore Dresser.—Plan Sheet II.

the large amount of clayey material which tends to remove by mechanical action particles of even fine clean tin ore. As showing the relative yields in coarse and fine the jigs save from 60 to 75 per cent. of the total, the buddles 4.75 per cent., and the slime tables 15 to 20 per cent.

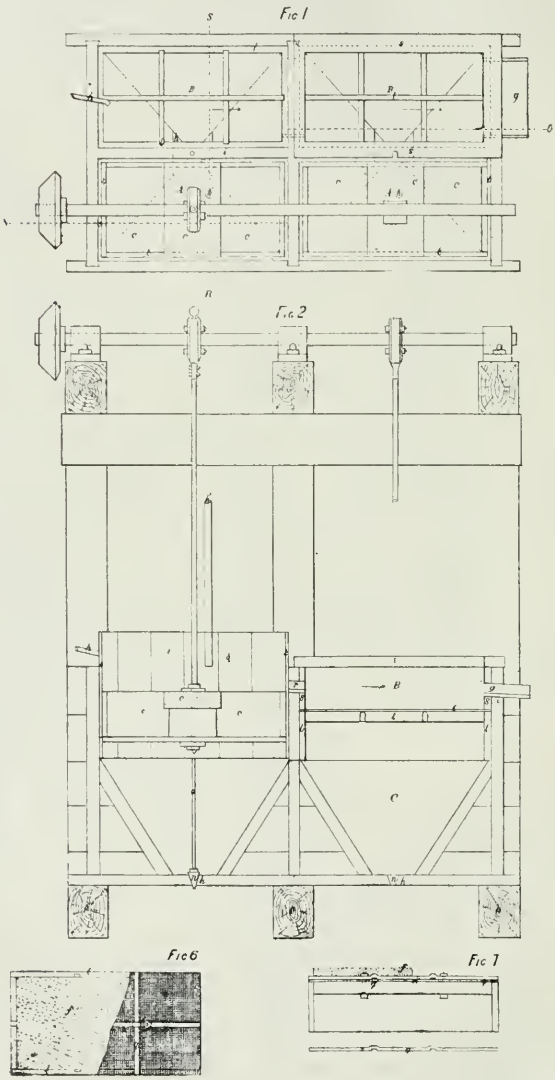
The ore as broken at the mine is roughly classified, all the coarse material going to a couple of stone breakers on the mine itself. The whole of the ore is transported in trucks along a steam tram to the works. These are situated on what must have been at one time the base of a waterfall, the creek falling rapidly below them. The high table land covered with basalt terminates abruptly on the cliff high up above the works. Occasional boulders of basalt become detached and threaten the works below. Beneath the basalt is a yellow clay, and much difficulty was experienced in building huge retaining walls and making floors and buildings on a site which must have been a tangled mass of great stones and scrub. The result is that all materials flow by gravity, and there is no



Kayser's Patent Ore Dresser.—Plan Sheet II.

re-elevation of anything except the final product—the dressed oxide of tin. The plant is divided into two sections, one of 15 and the other of 60 head. The 15-head battery seemed to be made use of for dealing with rich ore, the 60-head for the great bulk of the material.

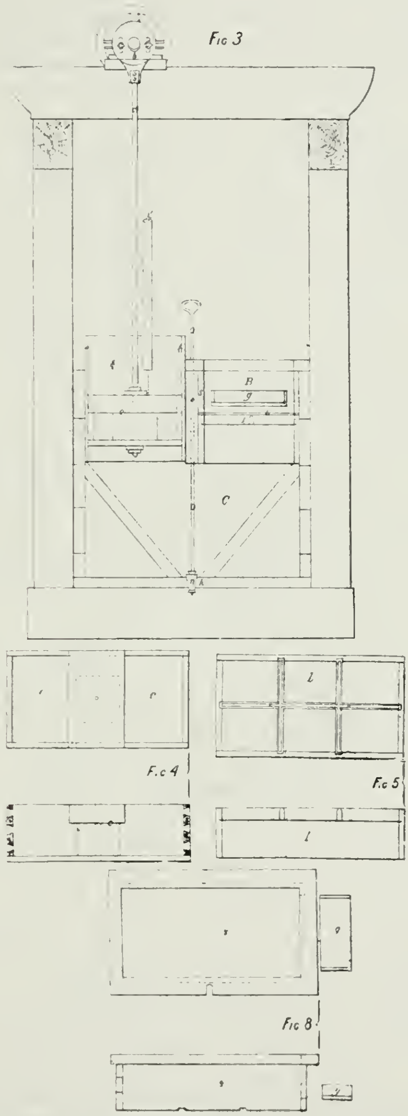
The boxes are of the ordinary Australian type, woven screens having 14 holes per linear inch being used. The false bottoms, or dies, are almost on a level with the bottom of the screens, thus giving a high discharge. The crushed material after leaving the battery passed into a small Spitzlutten, having an upward flow to wash out the fines. The coarser products were delivered at the



Kayser's Patent Ore Dresser.—Plan Sheet III.

lower end of the box, the fine slimes overflowing to a slime settling box. The coarser products were treated in a jig of the Hartz type, the details of which are shown on Sheet 3. The coarse tin saved from the first compartment was absolutely clean, assaying 73 per cent. metallic tin. From the second compartment the coarser grains, still containing oxide of tin as specimens, pass into a second jig, the tails from each passing on to a buddle.

The jigs are divided into four compartments, two for the

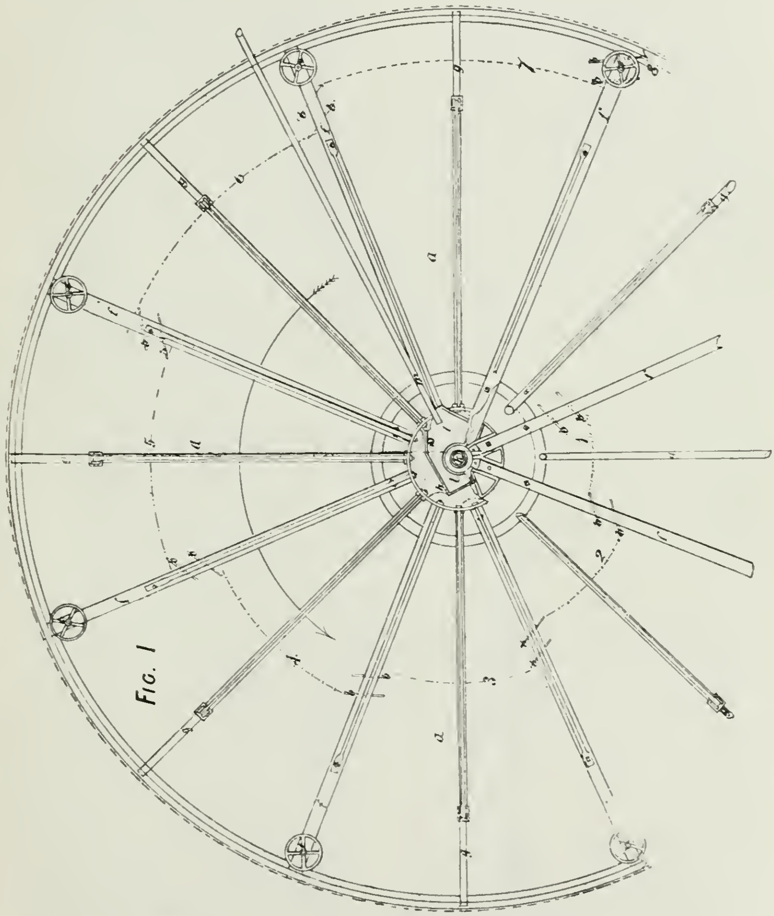


Kayser's Patent Ore Dresser.—Plan Sheet III.

plungers, whose stroke is given by an eccentric, and two compartments parallel to these, which are used for concentrating the ore. The first plunger compartment and the first concentrating compartment are open to each other below, but the former is closed horizontally above by the plunger and its compartment; the concentrating compartment is divided off horizontally by a screen, on which rests crushed tin ore of larger size than will go through the mesh. The whole vessel is filled with water, the crushed ore is run over the bedding of tin oxide on the sieve, a rapid and short throw is given by an eccentric to the plunger. This has the effect of setting up a pulsating movement in the water, part of it being forced up through the screen slightly lifting the bed; the heavy crushed mineral finds its way between the particles of oxide, while the lighter material is carried upwards and swept over the second concentrating compartment of the jig. The heavy material at the next stroke finds its way through the mesh of the screen, and drops down into a hopper shaped recess below, from which it may be drawn off by raising a plug. Mr. Kayser has introduced a frame for stiffening the screen, and yet not constricting its area, which is a great improvement on the usual type.

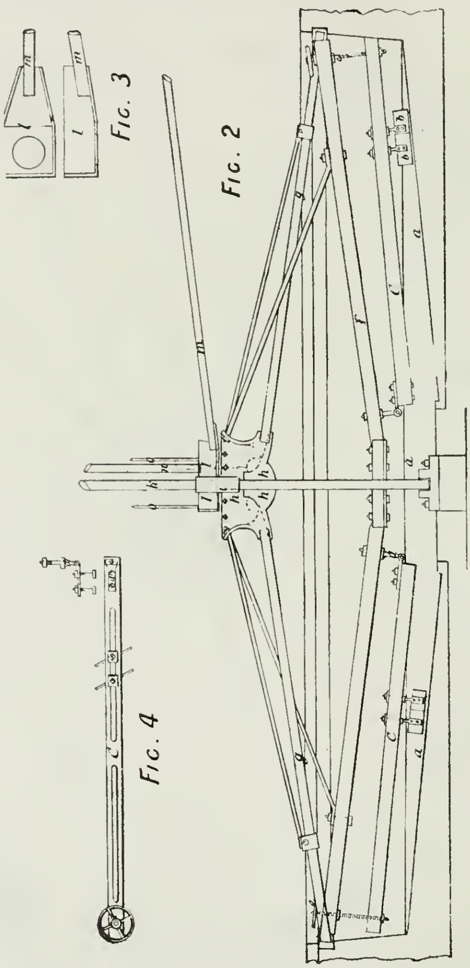
The coarse waste tailings from the jigs pass on to a buddle. This is shown in plan and section on sheet IV., and consists of a circular concave table 18 feet in diameter. Pulp is delivered evenly on to the outside edge of this, and runs towards the centre, where the light worthless material escapes. The heavier material settles on the surface of the buddle. In order to preserve an even surface, and to assist in removing the worthless sand, a series of scrapers are provided. These are suspended from eight radial arms, which are attached near the centre of the buddle to carrying arms by means of a hinged joint or piece of steel spring, and at the other end are suspended from the arm by a screw fitted with a hand wheel, so that they may be elevated or depressed. There are a pair of scrapers placed close to each other on every arm. These are all set with their forward edges projecting inwards, and are arranged systematically from the outside on the first arm to the inside on the last one, so that in a complete revolution of the arms the whole surface of the sand is scraped. A vertical spindle, resting in a footstep, carries the arms and tie rods for their support, also the radial pipes spaced between these for supplying the buddle with an even feed. The arms revolve about six times per minute. The sand, according to its composition, accumulates on the sloping surface, and the workman in charge gives a turn to each wheel as it revolves slowly past him. This gives a higher surface for the sand to concentrate upon, and the process is repeated until a layer several inches in thickness is formed. The buddle is then stopped, and the accumulated material dug out. A section of it is kept as a tell-tale, for the fineness of the laminations will at once show how the raising wheel has been manipulated.

The coarse sand from the buddles is ground in a Chilian mill, or practically reduced to slime. It then passes on to a slime table shown on sheet V. On this the material is concentrated to a certain extent. The concentrates are further enriched by a repetition of the process on a similar machine. It then passes on to a final machine of the same type known as the cleaning table. On this only the heads are kept, the tails and middle products being swept away. The slime table consists of a convex table 16 feet in diameter, having a fall of 1¼ inches per foot. The tables are substantially built. The under part is a framing of sixteen struts, joining below on a vertical spindle, and supporting at their other extremities the joists to which the flooring boards of the buddle are attached. These flooring boards are put on in sixteen sections, each board forming a tangent to the circle that a series of them surround. A second flooring above breaks joint with the under set, and gives a surface which may be planed off very accurately. In order to give a more perfectly even surface, the tables are now covered with a composition. A slow motion is given to the table by means of a worm wheel. The diluted slimes are fed on by a fixed pipe over one section of the table. If the



Kayser's Patent Ore Dresser.—Plan Sheet IV.

table were at rest the waste material would simply run off below the point it had reached the table, while the heavy slimes would lie on the surface near where they entered. Now, if the table slowly revolves the concentrates will be carried round beyond the reach of the water, which, as before, would sweep the slimes away. These would preserve a straight course down the surface of the table, as if it were at rest, but at the same time would be carried round a distance equal to that which the table would revolve, while the particle was moving from the inner to the outer circumference. With reference to some fixed plane below, say, the ground, the path of the light particle would have been a curve. The heavier products still clinging to the surface of the table are treated with an additional supply of clean water, which further frees them from gangue, and finally concentrated material is all swept off the surface of the buddle as it revolves around to meet a



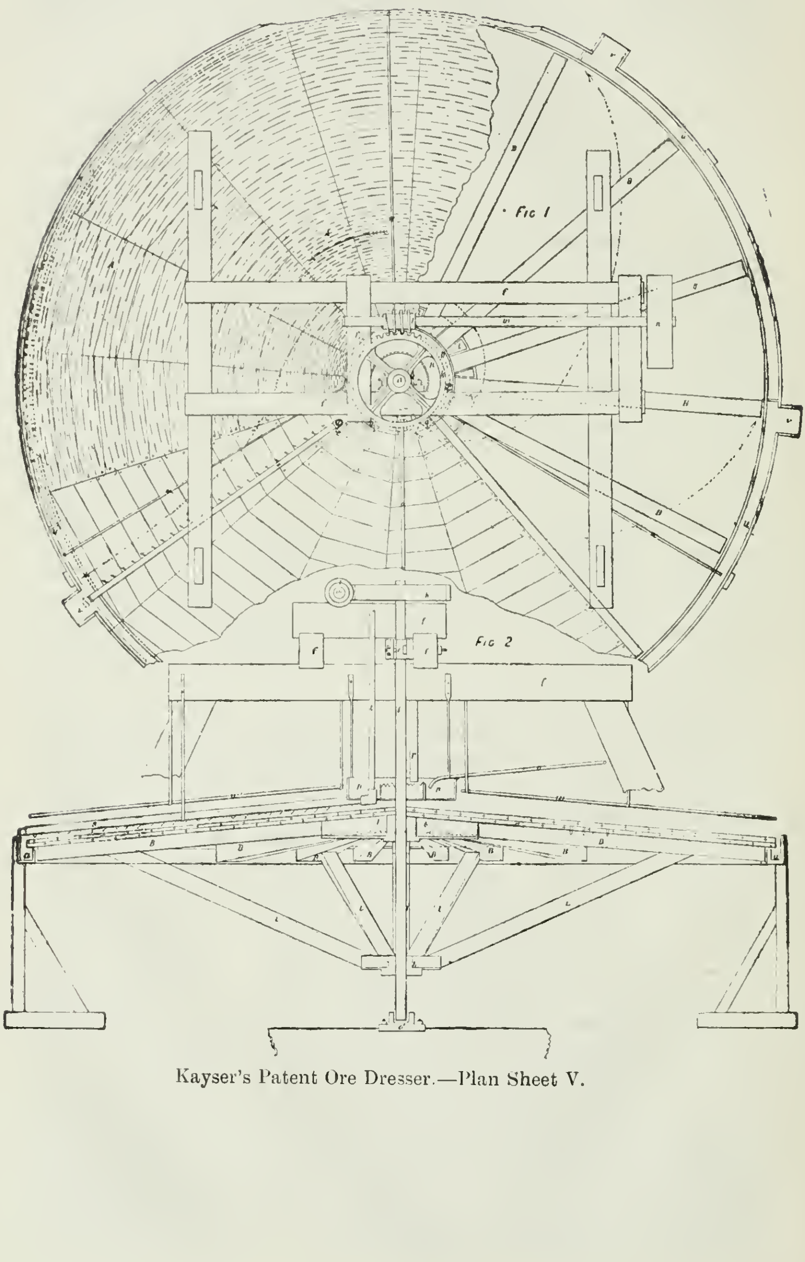
Kayser's Patent Ore Dresser.—Plan Sheet IV.

jet under high pressure. The concentrates so obtained are fed on to a cleaning table, where much the same process is repeated. However, in this case, after the concentrates are brought round on the table, for a certain distance, a jet of water is so directed on them that the middle product is swept away, and the rich tin slime alone saved. It would seem as if there were some loss in this operation since it would be hard to cut out the whole of the oxides of iron and pyrites, and leave the tin oxide.

The slimes from the battery and from all overflows where they are not specially treated are run into spitzkasten, sheet II. The fine material is thus sized, the coarse particles settling in box No. 1, where the stream is rapid. The coarse material is drawn off, and run on to a slime table. The current, owing to the larger area of cross section of No. 2 box, is much slower than in No. 1, consequently a further settlement of slimes will take place. These are drawn off, and treated on a separate slime table, similarly with No. 3 and No. 4, while the issuing water flows away almost clear. The method of classifying by size before classifying by weight is very successful here, for tin slimes are saved which take hours to settle in still water. But after all efforts to catch this floating material there must be a considerable portion which eventually escapes. The dressed material is still not clean enough, and the final operation of tossing has to be gone through. This is done in the dolly tub, an ordinary strongly made tub. The ore is gradually fed in, water being present, and the whole is tossed or worked round with a shovel after the manner of the alluvial miner, when puddling. So long as the shovel is kept going, some hundredweights of oxide may be kept in suspension. Tossing is kept up for some time, after which the grains are allowed to settle, this being facilitated by tapping on the outside with a hammer, the water being kept in a state of vibration. This operation at Bischoff is performed mechanically. The coarse and heavy ore settles to the bottom, leaving sand of lower specific gravity above. The water is drained off, the top layers scraped off, and saved for redressing and the concentrated tin ore bagged up. A very ingenious method of elevating the slimes and intermediate products requiring redressing by means of a jet of water under considerable pressure is made use of. This has the advantage of giving an even feed at a trifling cost.

The upper portion of the Mount Bischoff plant may be said to to consist of crushing appliances, jigs, buddles, Chilian mills, slime tables, and dressing tables, with the tossing tub to give a finished product. More than a mile further down the creek are the ringtail works, Mr. H. Fooks being in charge of these. The sand coming down the creek is impounded in two dams, No. 1, the upper, and No. 2, the lower. No. 1 is allowed to slowly fill. The sand in this is run into No. 2, and No. 1 allowed to fill again. No. 2 affords a sufficient supply to last for some time. The material is run on to the buddles described, there being six of these. Each three supplies concentrates for a Chilian mill. The slimes from the Chilian mill are gradually enriched by being passed successively over five dressing tables, the product from each one being lifted automatically by a jet elevator on to the next. The product from the last one goes to the cleaning table, thence to the dolly tub.

In spite of the large rainfall, owing to elevated position of the works, the natural water supply is unreliable. A series of reservoirs has been built, guaranteeing a fairly constant supply. It would seem as if more power would be available by making use of the fall down the creek. It must be many hundreds of feet to the mile. In that case any surplus of water must be relied upon to shift the tailings to the ringtail works.



Kayser's Patent Ore Dresser.—Plan Sheet V.

There is a well-equipped engineering shop in connection with the works, also a foundry, so that all the work requisite can be done at short notice. The whole of the extensive works are under the charge of Mr. Quinton, the acting manager, and it was a pleasure to see such a large, almost automatic plant working so smoothly. It is hardly necessary to say that the spirit of the designer is still paramount.

An interesting comparison between the noted tin mine in Cornwall, the Dalcoath, and Mount Bischoff for 1901 may be made. Both mines were producing about the same grade of ore, viz., 2 per cent. Bischoff is now nearer 1 per cent.:—

DALCOATH.	£	s.	d.	MOUNT BISCHOFF.	£	s.	d.
Authorised capital	£350,000	0	0	.. ..	£60,000	0	0
Paid up	303,667	18	11	.. ..	29,000	0	0
Balance from June 30th, 1901	£750	12	0	.. ..	£46,803	0	2
Profit half-year, December 31st, 1901	16,273	18	8	.. ..	31,671	0	2
				Interest ..	378	0	8
Total	£17,024	10	8	.. ..	£78,852	1	0
Less written off as under:—							
Depreciation of machinery and plant	£2,333	13	2	Plant ..	£3,585	9	1
Loose plant	445	2	9	Races and dams ..	411	10	9
Development of mine	6,153	7	3	Horses, &c.	66	2	3
	£8,932	3	2		£4,063	2	1
Difference	£8,092	7	6	..	£74,789	11	11
Less dividend 6d., 2½ per cent.	7,481	6	3				
Balance forward	£611	1	3				

Do. Dividends.		No.
6,000	..	292
6,000	..	293
6,000	..	294
6,000	..	295
4,500	..	296
4,500	..	297

	£	s.	d.
Dividend = 114.14%	33,000		
	£41,789	11	11
Income tax	1,650	0	0
	£40,139	11	11

July 1st to December 31st—
632 tons 14cwt., valued
at £120 per ton for tin

DALCOATH.	£	s.	d.	MOUNT BISCHOFF.	£	s.	d.
Black tin sold, 1033 tons 13 cwt., value	£71,133	14	1		£55,800	0	0

There is certainly a vast difference in the returns. Mount Bischoff made a profit of £31,670 on an output of £55,800, while Dalcoath only made £16,270 on £71,133. The comparison is interesting, so far as figures go, but the difference in cost of production would be found in underground work and hauling, as against surface quarrying. The hardness of the rock is also an important factor in milling. The duty per stamp of the Bischoff plant would only be a little more than half of what it is now were hard rock alone being crushed. Independent of this the cost of production is remarkably low, as is attested by the following figures:—

	s.	d.
Mining, including new works and maintenance..	2	8.221
Filling, hauling, and emptying trucks	0	6.026
Crushing, dressing, and maintenance of plant	0	9.600
Slime sheds	0	1.228
Ringtail sheds	0	2.072
Management and supervision	0	6.370
Plant, including all machinery	0	3.075
Development and progressive works	0	2.063
Waterworks	0	.305
Ore bagging	0	.374
Sundries	0	.868
Stores	0	5.226
	5	9.428

The plant diagram gives an approximate arrangement for each 20-head of stamps at Mount Bischoff. With the falling off in the grade of the ore and the alteration in character, the relative number of concentrators need not be as great as indicated.

The Mt. Bischoff Company is the fortunate possessor of tin smelting works which deal with the whole of the ore produced from the mine and most of that produced in Tasmania. The works are situated in Launceston because the only saving in freight effected by having the works on the mine would be 30 per cent. on the ore, while the carriage of coal and materials would amount to more than the total weight of the ore. At Launceston the works are centrally situated for the whole of the tin districts of Tasmania, and the ores from the various mines may be economically smelted by judiciously blending them.

As pointed out in a former article, the tin smelter requires the oxide of tin as pure as can be obtained. The great weight of the oxide (sp. gr. 7) fortunately favors the tin dresser; yet the weight of the metal, only having a sp. gr. of 7.3, is against the metallurgist who desires a metal much heavier than his slag. The oxide therefore is packed into proportionately less space than the metal itself, and probably this is in some way connected with its hardness. The metal melts at a very low temperature—230 C, very much below—contrary to generally received opinion—the melting point of lead. When melted it is very fluid and will run like water into any crevice. At a very high temperature it will boil and take fire, the oxide forming passing away as a white powder. It alloys with almost every other metal, and if only metal, sulphur, or arsenic, is present when in the smelting furnace, the reduced tin will take them nearly all up and be rendered impure. Not only are foreign metals objectionable, but owing to the fact that oxide of tin acts both as an acid and a base, any silica present when heated with it will form a silicate of tin, which on smelting with charcoal or coal alone is not reduced to metal Similarly if a base such as soda is present it will unite with it to form a stannate which is also not reduced. A corresponding action is said to take place with lime.

It will thus be seen that if silica is present it combines with part of the oxide of tin to form a slag. The problem at once arises of what material should the furnace be constructed. If firebricks are used there must be a loss through the oxide attacking them; if dolomite bricks were used they also would be attacked, and probably would not be hard enough to stand the mechanical operations in the furnace. The use of a more neutral compound, such as bauxite, naturally suggests itself. So far as I have been able to learn this material has not yet been used, but fire bricks, most of which go to form tin slags, are solely relied upon. I am quite sure that at such progressive works as are at Launceston, a trial would be made of these bricks if they were manufactured from the vast deposits of bauxite in New South Wales. The texture, shape and edges of such bricks, would for mechanical reasons have to be perfect.

Given a fairly coarse pure oxide of tin the metallurgy of it is so simple that even the Chinese are able to extract more than 90 per cent. of the metal with their crude furnaces. The Romans also were able to recover a high percentage of metal by simply making a small hole in the ground, filling this with charcoal, heating with the aid of a bellows, and then feeding in alternate layers of tin ore and charcoal. The natural outcome of this was to build a short shaft furnace, and these most primitive furnaces are in use at the present day as they were in use in Cornwall three hundred years ago. Even a century ago in Cornwall, these shaft furnaces—6 feet high, 2 feet square at the top and fourteen inches at the bottom, the bellows nozzle or tuyere being 10 inches from the bottom—turned out about a ton of tin in 24 hours. The smelter even in those times had to return 70 per cent. of tin from all the oxide brought to him. The Chinese also in the Malay Peninsula are able to obtain high yields from stream tin by smelting in small circular furnaces, made of clay held together by bamboo staves and hoops—in fact an ordinary barrel well lined with clay and stood on a tripod, would serve well enough to smelt pure ores in. The blowing is generally done by means of a crude force pump, the barrel of which is made from a hollow bamboo stem. A long piston, fitted with flap valves, worked by coolies supplies the draught. This method of smelting gives a very pure tin, for the stream tin as a rule is not contaminated with arsenic, sulphur, and such noxious materials. The temperature attained is not so high as to readily assist in the reduction of iron, while the charcoal fuel used introduces but little impurity into the reduced tin, while the metal, when once reduced, flows away rapidly. The lining of the furnace does not seem to be attacked at the same rate that the bottoms of the reverberatory furnaces are, consequently less slag is formed.

Shaft furnaces could not be used on very fine ores or slimes on account of the loss in dusting, so that as mining progresses and alluvial mining gives way to lode mining and reduction of large quantities of hard rock for small quantities of tinstone the reverberatory furnace must take the place of the simpler shaft furnace. The reverberatory furnace for tin smelting originated in Cornwall, and was much the same in construction as at the present. The object of it was to have one compartment in which the tin oxide and charcoal or coal were mixed, and another for burning fuel to supply heat to the former. The furnaces used by the Mt. Bischoff Company are of the Cornish type. There are six of these, each pair being connected to one stack—Mr. Latta would prefer a separate stack for each furnace. The external measurements of the hearth are 12 feet 9 inches by 15 feet, the fireplace projecting 6 feet beyond this and being 6 feet 6 inches in external width. The interior measurement of the hearth, which is oval, is 13 feet 3 inches in length by 9 feet 8 inches in width, and of the fireplace 4 feet 6 inches in width and 4 feet along the length of the furnace. The ash pit is open from the end, while the fuel is fed in from the side. The fire bridge is 2 feet wide. There are two feeding and rabbling doors on one side, the discharge hole being on the opposite side. The arch, which is only 18 inches above the floor, the fireplace, and the side walls, are built permanently, but the hearth is built so that it may be replaced many times during the life of the furnace. The whole of the parts in contact with the flame are built of the very best English fire bricks, the outside being of ordinary brick, the furnace being held together by means of the usual iron rails and tie rods. For the bottoms of the furnace, longitudinal girders are put in. On these rest a number of cross girders, all being of railway rails packed close together. A thin bedding is laid down, and on this the bottom, consisting of single fire brick on edge, is packed in. The slightest crevice left would allow the molten liquid tin to stream through, so that special precautions have to be taken to have all the joints tight: their faces should fit like sheets of glass against each other. When they are all laid down the surface should have an even slope with a 5-inch fall towards the tap-hole. These bottoms only last three months.

About a couple of feet below the floor is an inverted brick arch floor designed to catch any tin which may escape. The method made use of at the Straits for catching the drip tin is a vault 8 feet deep under the hearth. This is filled with water so that metal is granulated in dropping through. Escape pipes 18 inches in diameter are provided for any sudden outburst of steam, but it is stated explosions never occur if the vault is kept full of water. The water is pumped out once per week and the granulated tin collected. The products of combustion pass through a short lateral flue from the end of the furnace into a stack 7 feet 6 inches square external and 4 feet 6 inches square internal dimensions at the base.

The Mt. Bischoff ores contain notable quantities of oxide of iron, and if smelted by themselves would produce a ferriferous tin. The alluvial ores are somewhat silicious, and if these were smelted alone would give a silicate of tin, but by a judicious blending of the two the silica and the oxide of iron combine, giving a high-class tin and better yield than either would produce. The furnace is first heated; then a mixture of 50cwt. ore and 10cwt. small coal is thrown in and the air excluded. The first reaction will consist in the distillation of the coal, leaving the fixed carbon, since the temperature will not be high enough for the reaction.

$$SnO_2 + C = Sn + CO_2.$$

At a higher temperature this reaction is probably brought about in two stages—first the contact of the carbon and oxide of tin give

$$SnO_2 + 2C = Sn + 2CO,$$

then

$$2CO + SnO_2 = Sn + 2CO_2.$$

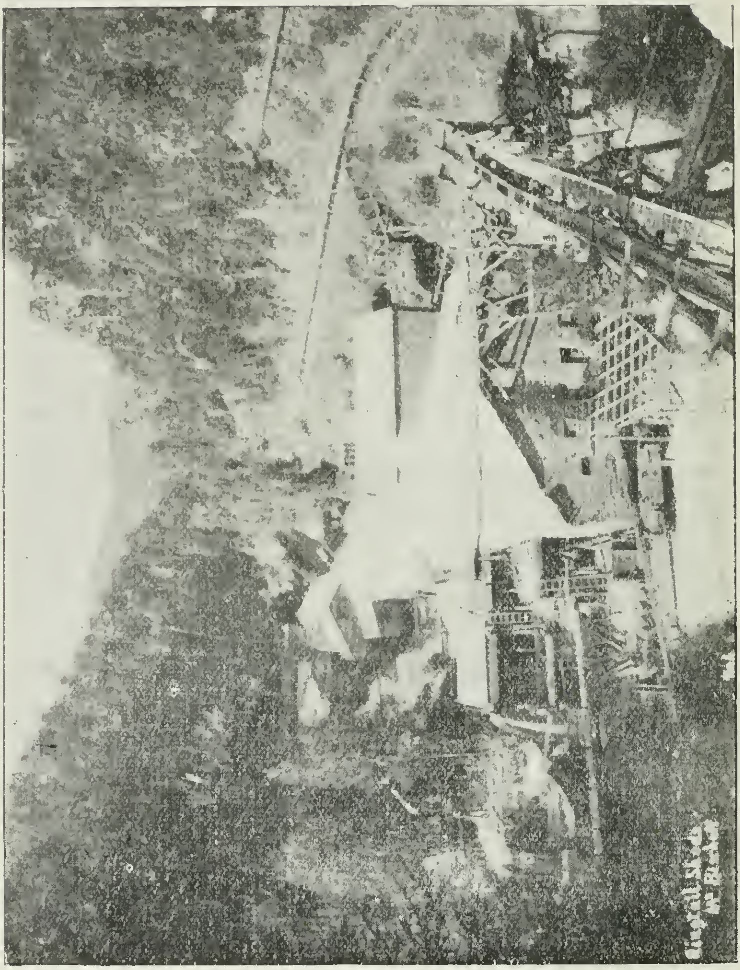
The mixture is rabbled several times, and in about eight hours the action is complete, the metal sinking and a slag rising to the surface. During that time the temperature is maintained by supplying fuel to the fireplace. The total coal used is about equal to the weight of ore for pure ores.

The melted charge is run into the well or float and allowed to cool slightly; the slags are skimmed off. The furnace is heated strongly for another hour, and the slags still remaining there are melted, run out, and reserved for further treatment. Another charge is immediately fed in through the side rabbling doors, the skimmings from the refining of the previous charge are thrown on top, and the doors luted up with clay and the process repeated. About 40 tons per week are put through each furnace.

The tin is ladled from the float after standing in it for about an hour into the refining kettle. This is simply an iron pot about 5 feet in diameter, standing between a pair of furnaces, any unfused residue in the bottom of the float going back to the furnaces with the next charge. As soon as the kettle is full, green billets of wood attached to an iron frame known as a cage are lowered into the molten tin. A heavy weight attached to the upright arm of the cage causes the wood to sink when lowered with a block and tackle. A vigorous boiling at once ensues, due to the steam and gases evolved from the wood. When the action slackens, fresh pieces of wood are lowered in and the process continued until the tin is considered to be refined. The dross is skimmed off from time to time. The time taken is from two to four hours.

What the action of these gases is has not been satisfactorily proved. Most text books on the subject state that an oxidising action goes on and that the various metals are oxidised and float to the top. Mr. Latta, the metallurgist, holds that no oxidation takes place, but that various impurities held by the tin are mechanically removed by the vigorous churning action which goes on. It certainly seems as if his opinion is the correct one, for at the temperature of the molten tin it does not seem as if water vapor could be decomposed. If it were decomposed the hydrogen and oxygen liberated would be in a balanced condition, which is not favorable at that temperature to either exerting any chemical action on outside substances. In fact, with the distillation of the wood the action would appear to be more of a reducing than oxidising one.

After refining the wood is withdrawn, the surface skimmed, and the metal ladled out into moulds, each holding 75lb., any residue in the bottom of the kettle being put to one side for retreatment. The tin so obtained is very pure, assaying 99.8 per cent. metallic tin. The purity of the metal is tested not by analysis, but by casting a bar 2 inches square and 6 inches long. This is nicked across the centre, and the ends doubled round until they meet. Should the metal be pure, the arched surface will show a smooth silky lustre. If the grain is broken in the slightest, or if it is



The Ringtail Sheds, Mount Bischoff.

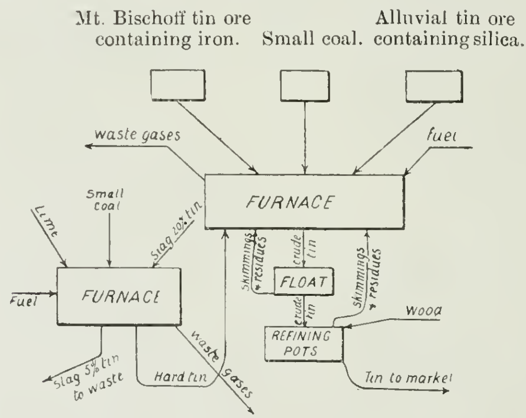
at all rough and shows small pitted holes, the metal is not pure, and would be condemned. It only requires a minute proportion of impurities to spoil the best tin. For instance, 1-10.h per cent. of lead, or 2¼lb. per ton, would ruin tin for market purposes: also a trace of arsenic. It is utterly absurd that the tin smelter should have to waste his time in order to produce a pure metal which is seldom used in a pure state. Nearly all the tin produced goes to form alloys, or is used for the preparation of tin salts used in dyeing. If used for tinning iron, there surely can be no objection to having a trace of iron there; if for bronze, a small amount of copper would not hurt it, while if used for solder or making salts the lead already present would not be objectionable in either case. In the former it would only need a little less than usually required, while in the latter it either would not dissolve or could be got rid of readily. It would seem that any excuse is good enough for the buyer to take a large slice off the legitimate price of the metal and the remedy should lie in turning out a finished article instead of sacrificing the raw material. Some elements such as sulphur and arsenic are highly objectionable, and in the case of these it is essential that they should be got rid of by some preliminary process before the ore goes to the smelting furnace, for once they enter the tin there is no process that, up to the present, has been discovered to remove them.

The slags formed only bear a small proportion to the amount of tin reduced, consist of silicate of tin, and contain numerous small shots of tin and an alloy of iron and tin—the bulk of the material being locked up as silicate. This runs up to 20 per cent. of tin. These are broken down to about 2 inch gauge, and a charge of about 25cwt. is mixed with about 5cwt. of small coal and 1cwt of lime, and if the slags do not contain oxide of iron either scrap iron or iron ore is added. This is kept at a high temperature for about 10 hours. The slags formed are complex mixtures, but assuming the bisilicate is present the action of the iron and lime might be represented as follows:—

$$SnSiO_3 + CaO = SnO + CaSiO_3,$$
$$\text{and } SnO + C = Sn + CO,$$
$$\text{or } SnO + Fe = Sn + FeO,$$
$$\text{and } FeO + SiO_2 = FeSiO_3.$$

Should there be an excess of iron, and this is usually the case, then an alloy of tin and iron, known as hardhead, forms. With an excess of lime there would be a danger of forming stannates, from which the tin would not be reduced by charcoal alone. The slags tapped from this operation run about 5 per cent. of tin, but it is not profitable to work these over again, although they could be reduced to 2 per cent. The metal that runs out is largely alloyed with iron, and this is thrown on the ore charge, the iron being oxidised and then removed by the silica present. The total loss of tin in a 72 per cent. ore is from 1.6 to 2 per cent. The company buys large quantities of oxide, there being a deduction of 2 per cent. for loss on 72 per cent. ore and over, 3 per cent. on 65 per cent., and so on, the percentage loss necessarily increasing with the admixtures in ores—a 45 per cent. ore having 7 per cent. deducted. No base metal ores are taken, for the company has had a high reputation for its tin, and obtains the highest market price. The cost of treatment is slightly under £2 per ton.

The methods of assay adopted are first to obtain an even sample by running a sampling rod through each ore bag—the total samples being mixed and ground. An assay sample is taken, treated with hydrochloric acid if it contains iron or oxides of iron, the residue is then mixed with four to five times its weight of potassium cyanide and run down in the usual way. Mr. Latta emphasises the fact that pure potassium cyanide must be used—the ordinary 100 per cent. cyanide so commonly sold nowadays is made up of impure potassium cyanide mixed with cyanide of sodium. The latter, on account of the lower equivalent of the metal to cyanogen, serving to make a material which contains as much cyanogen in 100 parts as if it were pure potassium cyanide. This material often contains finely divided iron, which, of course, enters the tin. Many pots also are too porous for tin assays, and the highly fluid metal often finds its way into the material of the bottom of the crucible. The methods recommended by Hoffman, to heat the crucible until white fumes just rise, is the only guide one can get with regard to temperature. A very low temperature is as bad as a very high one.



Even with a cyanide assay the percentage of tin present should be high.

The method of assaying tin slags is to fuse the finely powdered slag with caustic soda in a silver crucible. Dissolve in hydrochloric acid. Evaporate to dryness to render silica insoluble, the tin being again dissolved in hydrochloric, the silica filtered off, and the tin precipitated with sulphuretted hydrogen and converted into stannic oxide and weighed as such. Mr. Latta states that the tin does not volatilise under these circumstances as chloride, the excess of salt formed preventing it.

It is not out of place to draw attention to the need there is for some speedy wet way method for the determination of tin in ores. The first trouble is to obtain a solution. The method with carbonate of soda and sulphur very seldom gives a complete solution with one fusion. The same may be said about caustic soda, and the fluorides—the last compounds being objectionable in any case. The reduction with hydrogen is never satisfactory, and in that case it is better to mix the ore with an equal bulk of finely divided charcoal. This gives a better chance for the hydrogen to ac., but even this in the presence of much silica is not to be relied on. Neither is the method of dropping the ore into fused potassium cyanide, then decanting the clear solution off and dissolving the reduced tin. I found the most speedy method of obtaining a solution was to take about three or four grammes of sodium peroxide, place in a small nickel dish about 2 inches in diameter, mix about half a gramme of finely powdered oxide of tin, by means of a stout platinum wire. Seize the dish in an iron tongs and melt at a dull red heat over a bunsen burner, giving a circular motion to the dish to prevent settlement. Heat in this way for about five minutes, then give a stronger heat—dull red for about three minutes. Wash out with water, dissolve in HCl. taking precautions to prevent spitting. It will be found a complete solu ion is obtained. The stannic chloride was reduced with metallic iron, any arsenic, antimony, or copper being precipitated, the solution filtered and evaporated to dryness to remove silica, and the tin estimated as before as oxide. A volumetric method is needed. Another method of decomposition is to fuse some metallic sodium and drop the ore containing oxide of tin into this. The tin is instantly reduced, and alloys with excess of sodium, but the action is somewhat vigorous. Slags may be attacked by powdering them finely and heating with strong hydrochloric acid in a sealed tube heated up to 200deg. C. The tin as silicate or metal passes wholly into solution. A plan of the plant operating at the Mt. Bischoff Smelting Works, Launceston, is given with this paper.

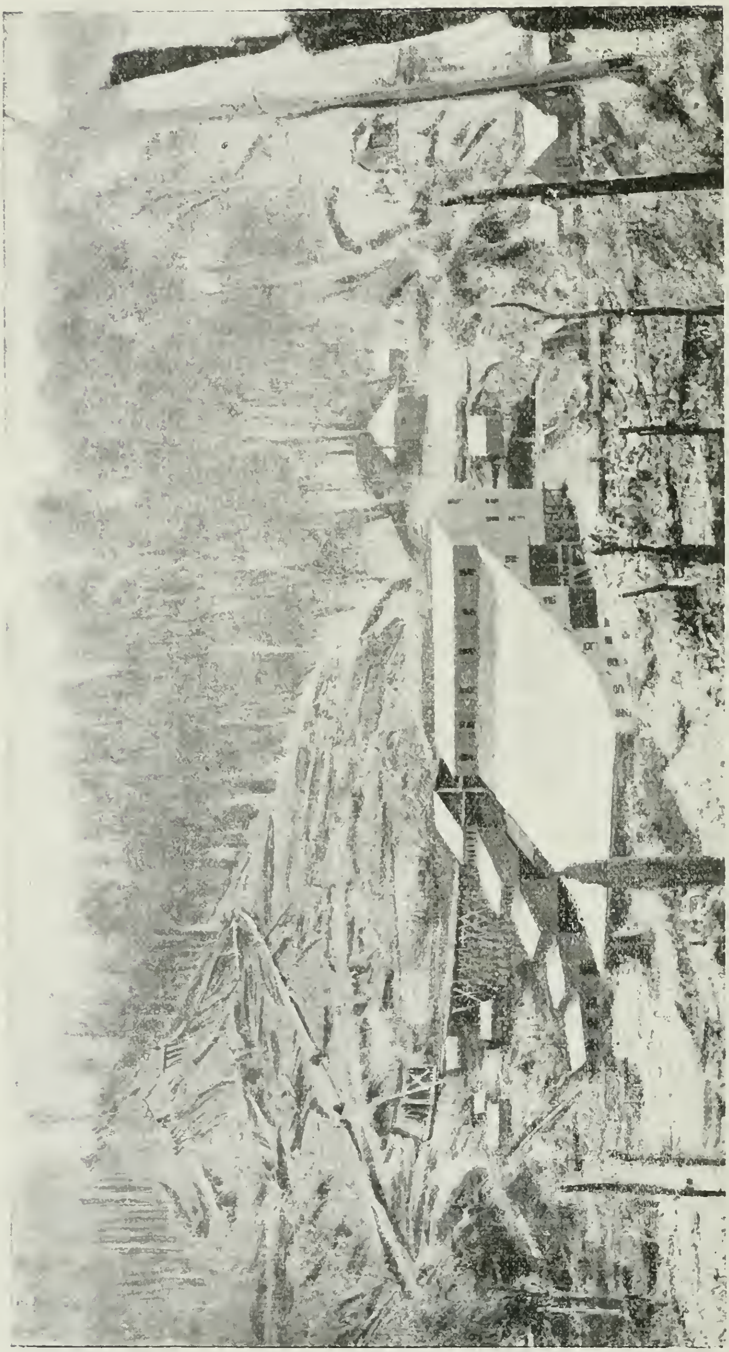
The Anchor Tin Mine.

As an example of a low grade tin proposition successfully worked I have selected the Anchor mine. This is situated at Lottah, one of those picturesque spots on the East Coast, between St. Helens and Scotsdale. Not far away are the well-known alluvial mines, the Briseis, Brothers, Pioneer, all of which are producing large quantities of tin. While in the same run of country as the Anchor are the Australian, Liberator, and other low-grade propositions, which will probably be worked for centuries. On perusing various reports on the subject the origin of these stanniferous rocks is differently accounted for by the various geologists for the department—and also Prof. Ulrich. The last-named pronounces it to be a stockwork in the granitic mass, or mineralised zones or bodies in the country rock which have no boundaries in depth and longitudinal extent, and may have none laterally. In other words, much the same as a blotch in the granite which carries tin, while the surrounding granite is barren. Both Mr. Montgomery and the late Mr. Thureau looked upon these formations as stanniferous dykes, which burst through the granite, but while there is a marked difference in their appearance to the surrounding rock, they do not possess the characteristics of invading dykes. Mr. Twelvetrees holds they are due to alteration in the original granite owing to the ascent of vapors finding their way through certain fissured portions of the granite. These vapors or solutions contained fluorine, which has led to the alteration of so many of the original granite minerals. The granite may be more silicified, secondary talc abundant, chlorite also present, while fluor spar and even fluor apatite is abundant in some places. The changes are made evident by the more or less alteration of the original granite into greisen, and it is noteworthy where this has taken place that cassiterite is more abundant. As Mr. Twelvetrees points out, the irregularity of the outline of the stanniferous rock and the presence of floors as well as the structure from side to side, is altogether against the theory of a dyke formation. I am not, however, prepared to accept Mr. Twelvetrees' conclusions in full, for it is hardly likely that such tin-bearing solutions ever got to the surface that existed when such action was in progress. Condensation and deposition would go on when temperature and pressure were lessened. It is also just as likely that below a certain limit no deposition ever went on to any great extent. In other words, assuming that such emanations occurred from some saturated region below, and that solutions and vapors arose, they would only do so through small fissures, but on rising to levels where the pressure was less would force their way laterally into every opening, altering the rock as they went. They need not necessarily rise to the surface, nor need there be any great lateral extension of their action below the particular level acted upon. In this way stockworks could be accounted for, and it would appear that so far as terms are concerned such a description fits these deposits better than any other. Such speculations are of very little use except as guide to future operations. If these were to extend down indefinitely of the same richness as at the surface the engineer's task would be a happy one, for he could frame his estimates with certainty. But it would be as well to remember that similar formations in other places have given way to barren granite at a depth, and not build too much hope upon their probable vertical extension. In the case of the Blue Tier deposits, the difference of elevation between the various mines which are all working on the same class of material is over 1000 feet, so that if the various levels correspond to those which existed when the tin was deposited the problem of working down to the bottom of the stockworks may be left to future generations. There is such a large extent of country here, and if it can be shown that such ore as exists can be worked profitably, a large amount of capital will be available. So far as surface work goes the field has been very well prospected. The Anchor mine was worked for more than 20 years. Up to the end of 1892, 30,734 tons were crushed for 288 tons of tin ore, or 0.937 per cent. For 21 months ending June, 1901, 111,167 tons were crushed for 434 tons tin ore, or 0.39 per cent. Corresponding amounts have been treated up to date, so that it will be seen that a large amount of material must have been available.

The workings are on the side of a hill some 700 to 800 feet below the roadway through Lottah township. The whole of the material is quarried out in benches, and everything goes to the mill. The lowest excavation starts from the working level, and is carried on into the face of the hill for about 120 yards by about 40 yards in width, the height being from 50 to 60 feet. The second face starts immediately over this higher up the hill, and is about 40 yards by 50 yards by 50 feet high. The third face is over this again, and is about 100 yards by 30 yards wide and runs up to the slope of the hill at the surface. More faces may be got above this, so that the amount of material available is colossal. During previous times much of the stuff must have been picked, for bare boulders stick out on part of the workings, while pot holes were put down and patches rooted out of the working floors; but under the present management, the evidences of erratic and unsystematic methods of opening up are being quietly effaced, so that in a very short time an even grade ore will be produced at a minimum of cost.

The whole of the material removed might be taken from a granite quarry so far as appearances go. Hundreds of tons could be looked over without seeing a specimen as good as some of the poorest at Mount Bischoff. The rock is a little softer than granite, but appears to have gained in toughness what it lost in hardness. It is porphyritic in appearance, quartz crystals through a felspathic matrix, most of it stained a light green with chlorite or rendered greasy looking by talc. White mica occurs in patches, the material being greisenised, and here the best tin occurs. The oxide of tin is present, as a rule, in fine crystals sparingly disseminated through the rock. A trace of copper pyrites also is present, as well as traces of phosphate of cerium and other elements of that group.

The ordinary mining operations do not need much description. The benches are continually being pushed backwards, each floor maintaining its level. As each face becomes inconveniently high another bench is opened up higher up the hill, and work pushed on as before. Holes are jumped, charges put in, and the material



Anchor Mine.—Ore Dressing Sheds

shot away from each face on to the floor below. Large stones are popped, smaller ones broken with hammers, and the whole of the material trucked away, rails being laid to any point of the workings. All the men were well distributed, and there was no wasted effort in unnecessarily handling materials; in fact, it was a case of navvying rather than mining, and left little, if any, room for lowering costs. The broken ore is trucked in a horse tram to the breakers, of which there are two No. 4 Gates. The material drops into a hopper of small size above each crusher, is worked into the breakers, and drops automatically into a hopper, thence to trucks below. These are run down a tram line to the battery hoppers.

The crushing plant consists of 100 head of stampers, in two sections of 50 stamps each, both sets being in different sheds and not in line. One plant was designed and sent out from England; the other, which is an exact copy of it, was built in Launceston by W. H. Knight. Each five-head battery has the usual Victorian style of box, the dies being about half an inch below the level of the screens. The latter are punched, having 100 holes per square inch. Each stamp, weighing 8cwt., is run at 96 blows per minute with a 7-inch drop. These are fitted with a key and cotter tappets. Each five head is belt-driven from a countershaft, which runs the whole length of the building, so that it may be disconnected without interfering with the others. A Challenge feeder of the ordinary type supplies each five-head with broken ore from a 300-ton bin arranged behind the line of stampers. The feeders require very little attention, and work in a satisfactory way.

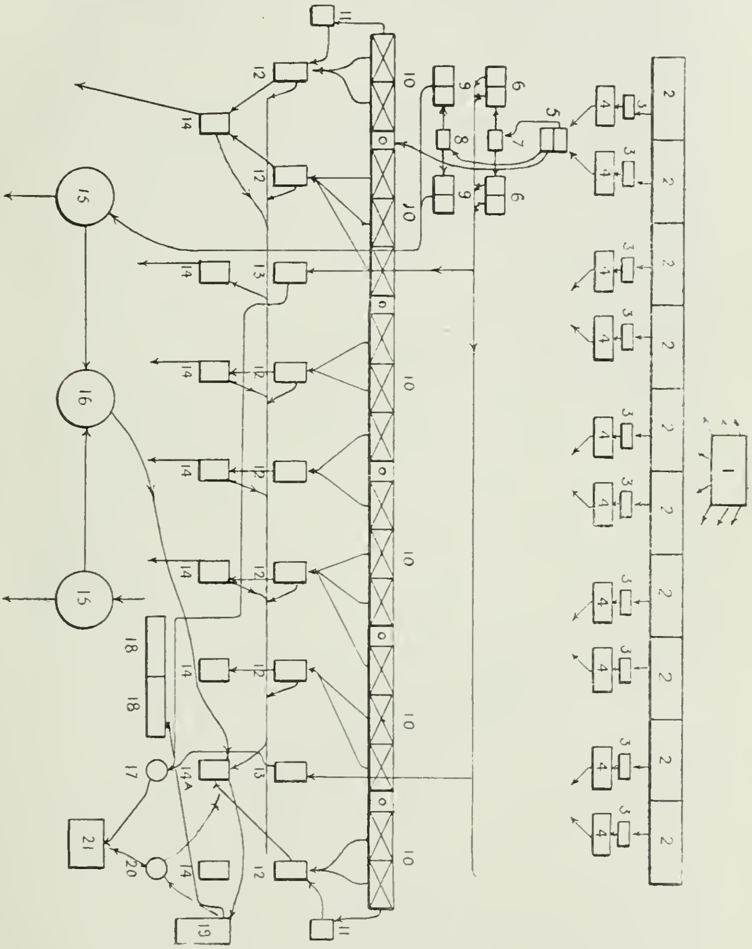
Water is fed into the box, each stamp being supplied separately. This arrangement is good, for it tends to give a more even distribution of sand within the box. The issuing pulp is led away from 10-head by a launder to a hydraulic separator, the coarse sand going to a pair of jigs, each having two compartments and run at 170 strokes per minute, with a 3-16th inch throw. The finer sands are led away from the bottom of another classifier to a pair of jigs running 170 strokes per minute and having one-eighth inch throw. The concentrates from these jigs and also from those from each 25-head of stamps are run on to a Frue vanner provided with a corrugated belt. The heads from these go automatically to a settler, the heavy portion going to a hydraulic separator, the dressed tin oxide from this going to the ore bin. The tails from the jigs pass on to a buddle, one for every 25 stamps. This buddle is 16 feet in diameter, with a slope of 1 inch to the foot, and revolves five times per minute. The surface is stirred by a pair of rakes and a pair of brushes made of native grass sweep round after these. The whole framework may be raised or lowered by a hand wheel working a screw which presses on one end of a lever, the other being under the framework of the buddle. From this buddle the tails run to waste. The tin oxide caught on these buddles differs from that at Mount Bischoff in not being encased in grains of sand, or specimen tin, as it is called. The heads from two of these buddles go to a fine buddle; the heads and middles from this go to a dressing vanner, the tails to waste.

The overflow from the classifiers above the jigs containing the sand and slimes passes into a series of pointed boxes, which are arranged parallel to the line of stampers from side to side of the building. There are sixteen of these boxes, each 25 head supplying eight. The overflow from each end of the series is led into another settling box, the overflow from this passing to waste, while the settled slimes, together with those from the spitzkasten, are fed on to seven Frue vanners. The vanners are supplied with plain belts, and run at 160 shakes per minute. The tails from these slime vanners pass on to seven vanners lower down, while the tails from the latter go to waste. The heads from the vanners go to a dressing vanner, where they are cleaned up, the tails pass over a second dressing vanner, the heads from this pass over the slime dressing vanner again, the tails going to waste. The coarse tin is cleaned in a hydraulic separator, which is a simple, yet effective contrivance. A circular tub 20 inches in diameter and 26 inches high is partitioned off horizontally by a diaphragm made of perforated plate with ½-inch holes spaced ½-inch apart. On this lies a brass screen, No. 12, and over this a layer of hessian. The screen is 12 inches below the top of the box; an overflow launder leads out some 4 inches below the top. A pipe let in near the bottom of the tub supplies a stream of water under a 12 feet head. The cleaner is filled with oxide, water turned on, and any light material is carried away. The cleaned tin oxide goes to the ore bin, the tails to a settler. From this they go to the tossing tub, or, if not clean enough, to the hand-buddle. The overflow from the settler goes to the tail race.

The slime concentrates from the dressing vanner go to the hand buddle, which is a rectangular inclined plane; the concentrates are fed in at the upper end, and swept down by an even stream of clean water. These are continually brushed upwards with a light broom; the heads settle near the top, and are cleaned in a tossing tub. The tub in this case is placed at an angle against the wall. The tin is fed in first, a hose is turned on, and it is puddled with the shovel, then allowed to settle, the usual knocks being given. The top is scraped off, and in course of time the pure tin oxide obtained. About half a ton is dressed in two hours. The middles and tails from both these appliances are settled, then retreated on the square buddle. Nothing is allowed to escape until it is redressed.

The final product assays over 70 per cent. of tin. The plant is driven automatically by two Pelton wheels 5 feet in diameter, which drive each 50 head of stamps and two Pelton wheels 18 inches in diameter, which drive the concentrating plant. The supply pipe is 16 inches in diameter, the head of water being 382 feet, the gauge pressure being 160lb. per square inch.

This mine and plant are exceedingly interesting, since about the cheapest mining, milling, and concentration in Australia are done on a fairly large scale. The tonnage for the fortnight preceding my visit was 5142, working six days per week. The amount of oxide of tin won was 11 tons, or .21 per cent., a little more than 1-5th per cent., or 4.7lb. of tin oxide per ton of ore, having a net value of less than 3s. 6d. Yet this wonderfully low return—in terms of gold, about 20gr. p[illegible] t[illegible] showed a profit of more than £100 per fortnight. The whole of the work was done for less than 2s. 9d. per ton. It must be remembered that although the rock is quarried from open cut, it is a hard, tough granite, and that the stampers are not of such weight best capable of dealing with large quantites, and that Frue vanners are most delicate machines, which can only be worked properly by those paying a lot of attention to them. All these drawbacks serve to make the work done on this mine all the more creditable and noteworthy. Only five men are employed per shift for the 100 head. These include one shift boss, two vanner men, and two feeders. On the day shift two dressers and five boys extra are employed.



General Arrangement of Plant.—Anchor Mine, Tasmania.

The use of water as a motive power has helped considerably to reduce costs, but with an adequate supply the possibilities of the Blue Tier are enormous. The Mining Department is doing excellent work in compiling accurate information concerning the rainfall and water supply of the watershed in which these mines lie.

The manager of the mine is Mr. Lindesay C. Clark, formerly manager at the Mount Lyell mine. The illustrations show the mine shortly after the 100 head of stampers was erected. The open cuts are now where the logs may be seen lying above the sheds.

The plan of the general arrangement of the works is given for each 50 head.

1. Stonebreaker.
2. Ore bins.
3. Challenge feeders.
4. 5-head stamps.
5. Hydraulic classifier.
6. Coarse jigs.
9. Fine jigs.
10. Spitzkasten.
11. Overflow settler from Spitzkasten.
12. Frue vanners for Spitzkasten slimes.
13. Frue vanners for jig concentrates.
14. Frue vanners for tails from 12.

14A. Frue vanner for dressing heads from 12 and 14.

15. Coarse buddles.
16. Fine buddle.
17. Hydraulic cleaner.
18. Settlers.
19. Hand buddle.
20. Tossing tub.
21. Ore bin.

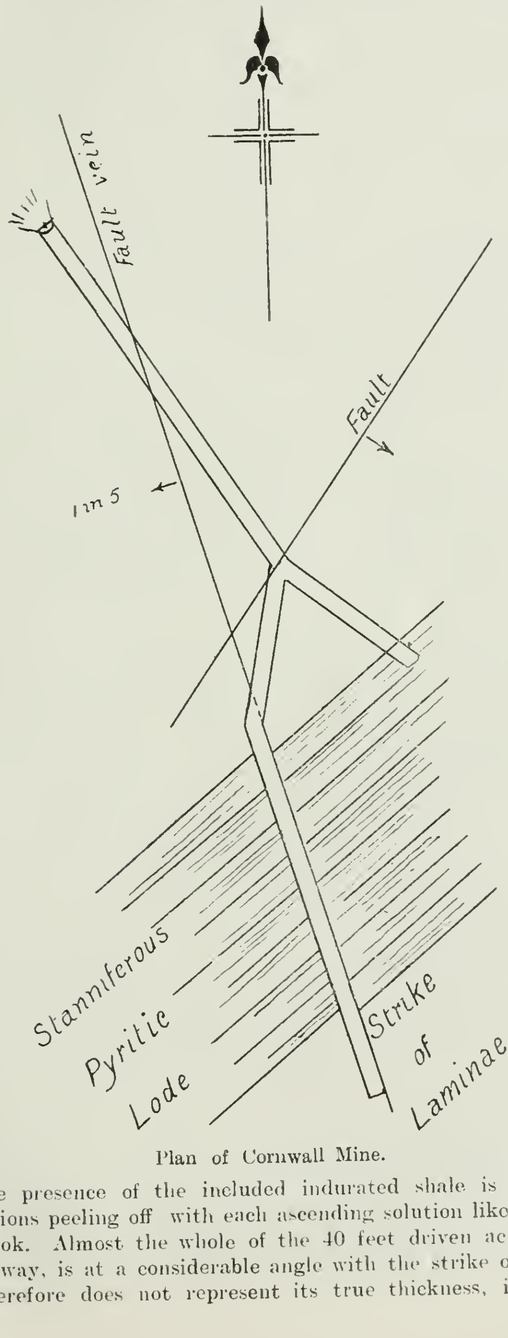
The "Cornwall" and "Mayne's"

As typical of two types of tin ore bodies, the present article is devoted to the property known as the Cornwall, and to that known as Mayne's. They are both easily reached from Zeehan, being from seven to eight miles distant. The former is on the Ring River, a tributary of the Pieman. It may be readily reached by taking the North-East Dundas tram for about eight miles and then striking across a high ridge—the Commonwealth Hill—for about a couple of miles, the country crossed over being serpentine, granite, and a much indurated slate; in the last the tin lodes occur. Stream tin mining has been carried on for years in the river, and its branching tributaries. It is somewhat surprising to see the small prospect that is payable. The oxide is worth about 8d. per lb., so that a yield of 1lb. to the dish, which is not uncommon, means an equivalent of 4gr. of gold. Were it not for lost time and the discomforts of life in this rainy region, many of these creeks would attract a great many more "tin scratchers," as they are locally called. On the north-east coast of Tasmania, where tin is more widely distributed, one regrets to see the industry mainly in the hands of the Chinese. In prodigal times the dogs were allowed to eat the crumbs that fell from the master's table, but in the lean years any country that has not the leavings of more prosperous years to fall back upon, will need a workhouse or its equivalent. It may be said there are no leavings when these human scavengers get to work. The good and the bad fields are despoiled—not only the crumbs but the loaves themselves are taken. A few local storekeepers benefit in the meantime, but all at the expense of posterity. To judge from the sleek appearance of John on the East Coast, his endeavor to appear on equal terms with those whom he lives amongst, his lines must have fallen in pleasant places. It may be the remembrance of him in Victoria which has led the Zeehan miners to politely request any Oriental arrival to take his departure by the next train—and, marvellous to relate, this people who are in this respect a law unto themselves, have been obeyed. The result is that no Zeehan streams are worked by Chinese, as they undoubtedly would otherwise be.

In the vicinity of the Cornwall mine many huge masses of oxide of tin were found. Some of these are said to have weighed close upon a ton. Efforts were made to discover the matrix which yielded such masses, and several lodes have been opened up. The most peculiar of these is the Cornwall. The outcrop is near the base of the Commonwealth Hill, which rises to a height of over 1000 feet. The lode, for the various exposures, will probably be found to be on the same ore body, shows out prominently in section by a cutting across its outcrop; is between 20 and 30 feet from wall to wall. This consists of a banded structure, the seams of which stand out in ribbon-like structure. It is only oxidised in part, the balance consisting mainly of pyrrhotite or magnetic pyrites. A small vein, only 2 or 3 inches wide, of arsenical pyrites of secondary origin, cuts through the centre of the lode. A tunnel has been driven on the hanging wall for about 40 feet. For about 20 feet it is in a gossan carrying oxide of tin, but beyond this the sulphides make again. Some of the gossan is very rich, but of course no great quantity has been opened up.

A shaft seems to have been sunk near the open cut, but work appears to have been discontinued there on account of the appearance of dense sulphide ore. A short distance below the upper level, between 20 and 30 feet, a lower prospecting tunnel has been put in. This is shown approximately on the plan. After driving 40 feet a vein, said by Mr. Waller, the Assistant Government Geologist, to consist of decomposed axinite, was cut. At 86 feet a fault was met with. The course of the tunnel was then altered, when it met the lode almost at its intersection with the axinite vein. It was driven across at an angle for about 40 feet, being in dense solid sulphides all the way. A small branch tunnel was driven from the bend in the tunnel. This struck the lode at a distance of about 20 feet.

The axinite vein clearly represents a fault plane, which was subsequent to the formation of the lode, for there has been a throw towards the south, showing that the lode after its formation was broken across and the southern portion has slid vertically downwards, thereby causing the lode to first appear in the hanging wall of the fault and to continue on the footwall when driven through. Mr. Waller, in his report, has put the main facts before the reader very lucidly. He has also had the advantage of studying in detail similar occurences in the district; but if his conclusions are correct, then the owners of the mine must believe that the axinite vein bears much the same relation to their lode as the indicators do to the gold reefs at Ballarat. In other words, Mr. Waller believes that the tin-bearing solutions with the pyrites came up through the fissure now filled with axinite, and that such solutions spread out wherever they could along the most porous and soluble beds of shale, the lode itself being looked upon as being formed through metasomatic replacement of a finely laminated shale—the laminations being due to the structure of the original shale. Should this be correct, one could not hope for any great lateral extension along the shales, and in all probability the richest part of the lode would be that cut through. Further prospecting should, according to this view, be rather along the line of axinite itself than in the line of the ore bodies. The other possibility, that the tin-bearing solutions ascended the channels in which they subsequently solidified, appears to me to be a more reasonable supposition, for it appeared to me that the fault plane represented by the axinite vein was formed after both pyrites and tin oxide were deposited. The banded structure is due to much the same cause as the laminations in our reefs. A fissure may be rapidly filled and the material which filled it will be uniform from side to side. Should any further movement take place solutions will be forced along the line of greatest weakness, which will be one or other of the walls. These solutions in their turn deposit their metals, which may be different from those originally deposited. Generally also a small amount of the wall of the first filling will be included between that and the second; similarly another film of slate or graphitic material or fine pyrites between the second and third, and so on until action ceases. It has always appeared to me that the presence of gold or other minerals in one particular lamination could only thus be accounted for. Similarly for this lode, it was not suddenly formed, but layer after layer was filled up, each layer slightly different from its neighbor,



Plan of Cornwall Mine.

and the presence of the included indurated shale is due to its laminations peeling off with each ascending solution like the leaves of a book. Almost the whole of the 40 feet driven across, which by the way, is at a considerable angle with the strike of the lode, and therefore does not represent its true thickness, is in dense pyrrhotite with the seams of black tin oxide showing from top to bottom. Were the gangue quartz or some non-metallic mineral it would be one of the richest tin lodes discovered, but the fact of such heavy sulphides being associated with the oxide of tin at once pulls down the percentage of metal by weight and makes the treatment much more expensive. Picked samples could be selected, which would give almost any return, and various estimates have been given of the value. When the stone is exposed to the weather for any time the surface is tarnished to a dull copper color. This film effectively obscures the richness of the ore.

In order to get an approximate idea of the value of the stone about three or four pounds of chips were collected all over the heap. These were ground and assayed, and gave 2.03 per cent. metallic tin. None of the oxide was visible in the stone—that is, none of the laminated seams apparent in the lode. So that the result is probably below the true value. It certainly looks much richer. An assay even if taken across the lode at present with the greatest care would not do much to develop the mine. The lode should be driven on into the hill, and since every foot of driving would be adding to the backs, the value of the ore, the extent of it, and the machinery required could be decided upon. It is reasonable to suppose that near the upper levels a considerable amount of gossan carrying cassiterite exists, but at present surface prospecting is rendered difficult by the growth of timber and scrub. While a certain portion will, no doubt, be oxidised, it is worthy of note that this form of pyrites—pyrrhotite—is but very slowly oxidised, and in many places throughout Australia I have seen it unaltered even on the tops of hills, only a few feet below the surface.

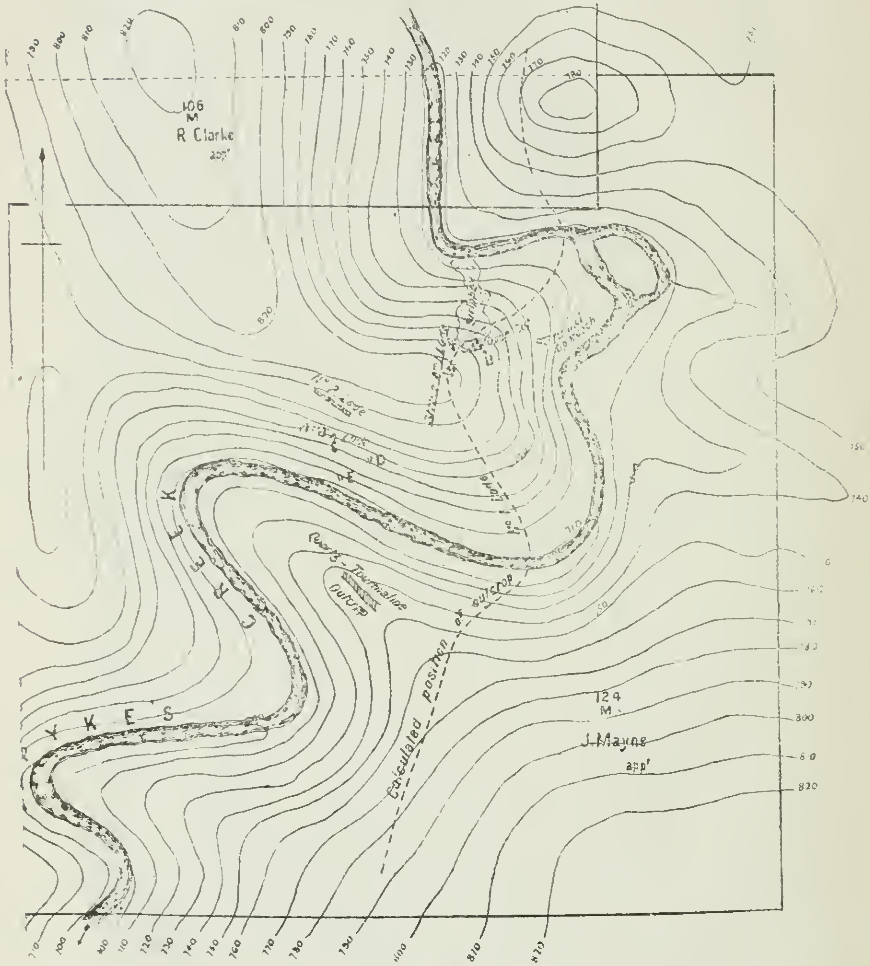
The treatment of such an ore as this may very glibly be got over by those who have not tried it in actual practice. Crush it, concentrate it, get the tin from the heads, roast the middles, and let the tailings go is the general advice given. Such a method would mean failure for this mine. Crush, roast and concentrate would be nearer the mark where all the material is, practically speaking, a concentrate extremely difficult to detach from the oxide of tin. With most Cornish ores even when pyrites are present the physical nature of the ore is widely different. Even at Mount Bischoff the treatment of the sulphides is a much simpler task than they would be on this mine, and so far, although an elaborate plant costing some thousands was put up years ago, near Mount Bischoff, this branch of the work is severely left alone. This is only mentioned to indicate a very important piece of work remains to be done in Tasmania, and if the Cornwall Company successfully leads the way a new era will commence in tin mining. There is no reason why it should not be done, and assuming certain costs:—

	Per ton.
Mining and development	15/-
Breaking and pulverising	3/-
Roasting	8/-
Dressing	2/-
Incidental expenses	2/-
	£1 10/-

With tin at £100 per ton, it would require 1.5 per cent. material to pay expenses. So if this mine should open up even as well as at present a handsome profit could be made with tin at its present price. Of course, after a time working costs would decrease, but it would not be safe to assume a lower estimate for a start than that given—and it would be still more unsafe to erect machinery before the mine is opened up.

The other mine referred to is of a widely different character. A large extent of country, comprising Mount Agnew and Mount Heemskirk, is composed of granite, and is tin-bearing throughout. The country is covered with button grass to a great extent, and the ground for the most part shallow. With the discovery of tin many years ago many limited liability companies were floated, and the results to the unfortunate shareholders were such that Heemskirk got a name as unenviable as Chillagoe. In reality the field never seems to have been properly tested; most of the money seems to have gone in the erection of batteries and appliances to save the tin which was not even prospected for. In one case the battery site was actually on a stanniferous lode which was never tested, while nearly a generation afterwards men were busy carrying material from the opposite hillside—not 200 yards away—and obtaining highly payable results of tin. About seven or eight miles on the road from Zeehan to Trial Harbor lived Mr. John Mayne, who was able by farming, dairying, combined with a little tin scratching, to make a living for himself and family. He knew what tin was, for he was surrounded by it, and had worked at Mount Bischoff for some years. His boys wanted a holiday, so he gave a provisional promise that if they could show him a claim yielding a pound of tin to the dish, they should have it, as well as a cash bonus. The boys never went more than 300 yards from the house in which they had lived for years, got a prospect from the base of an uprooted tree, and earned their reward. The father and sons started to work, pegging out a comparatively small area, for they then thought that in such a patchy place they would soon work it out. As time went on discoveries were made which caused the fortunate owner to form a different opinion. A small pot hole on the side of the hill, only a short distance from his back door, gave nearly £1000 worth of metal. A couple of men could have easily taken all the stuff out in a day. Lode after lode was opened, giving pounds weight of oxide to the dish, and without having done as much sinking as a wombat would have gone through in the time upwards of 30 tons of oxide were extracted by the most primitive means. For this surface show the fortunate owner wanted £10,000 cash and a large interest, and, according to report, he has since got it. There are other blocks beside the 18 acres from which the tin was won, but those had not been prospected to any extent at the date of my visit.

The area pegged out includes a section of Pykes Creek, and the lodes in it are in an indurated slate from 10 to 20 chains from the contact with the main granite mass forming Mount Agnew and Mount Heemskirk. The slates are converted into porcellanite, while dykes of green tourmaline run in a direction approximately parallel to the line of contact. These dykes are more or less stanniferous. The most extraordinary part about these deposits lies in the appearance of the oxide. In no case is it black, but of a dirty grey color, resembling cement. On first looking at the decomposed material forming the lode, even an experienced tin miner would pronounce it barren, but Mr. Mayne, dish in hand, most eager that no false impression should be created, advises you to pan it off and "to look at the gravity of it." Sure enough there seems to be as much oxide as dirt, and yet it is difficult to tell



Sketch Plan of Mayne's Mine.

which is which—probably even the "gravity" will not preven the slimed oxide from floating away in large quantities.

So little work had been done on the property that it was not safe to judge whether the extremely rich prospects obtained were due to the material of the lode or decomposition and wearing down of the surface; probably they are due to both. Neither was it evident which way the lodes were running, except perhaps the one marked No. 3 lode on Mr. Waller's plan. This appeared to continue across the creek for a distance of about seven chains. There also appeared to be other lodes, one below and one above, parallel to this, all trending in a westerly direction, with an underlie to the north. The hole from which the rich samples were obtained yielding many tons of oxide are marked A and B. The bearing of this lode could not be stated with precision. If as given in the plan it will cut the east and west lodes. Some of the oxide occurs in nuggets up to 20lb. in weight. These are also grey and most of them have at least on one side a mammillary surface which appears to have been superimposed in layers. The most important feature in connection with such a deposit as this is that while for years the granites themselves have been prospected and worked, it is the contact or rocks near the contact which have yielded the most metal. Mount Bischoff may be taken as an example, and here the same holds. The indurated contact rocks containing such material as tourmaline, topaz or flourine minerals in this extensive area should be carefully prospected.

While at Zeehan Mr. H. Castle showed me some splendid samples of rolled boulders containing oxide of tin and black tourmaline which were put to one side when sluicing for tin on the Stanley River. Five hundred tons of this rich material are said to be stacked. No doubt undiscovered lodes exist in that district also. A considerable quantity of monazite or phosphate of cerium is said to accompany the tin.

I desire, in concluding this article, to express my indebtedness to Mr. F. Borley and others, who spared no pains to give me accurate information, also to draw attention to the excellent reports on the Heemskirk tin ore deposits by Mr. Waller. There will be a great future for this district when every man has the technical knowledge necessary to enable him to deal with his own product. It is most regrettable that rich tin ores should be sacrificed because they contain a little pyrites. The need of a School of Mines at Zeehan on the lines previously suggested becomes more pronounced than ever.

Since writing about the troubles connected with the treatment of oxide of tin associated with sulphides of other metals, Mr. John Ditchburn, the manager of the Mount Bischoff West Tin Mining Company has supplied me with the following details:—The ore from the mine is crushed in a ball mill, the coarser materials passing to a coarse jig, thence to a dressing jig; the finer material goes to a fine jig. The overflow from the classifier passes to spitzkasten, and thence on to a pair of Wilfley tables, followed by a settling tank, the slime from which goes on to a Phoenix weir table. The concentrates obtained are roasted at Merton's furnaces in Melbourne, the product then being sold to the Smelting Works, Sydney. The assay of the concentrates reaches 70 per cent. and over, and what is more satisfactory still is that the deductions made are very small indeed. For instance, with stream tin 70 per cent. at 24s. 9d. per unit, the Mount Bischoff West Company was paid for 68.6 per cent., only 1½ per cent. being deducted, the rate being 24s. 6d. per unit, or a deduction of only 3d. per unit. The cost of treatment also is put down at £1 per ton. It is further satisfactory to note that the company is working at a profit, which is all the more creditable after tackling and being successful with what other companies have failed over. The directors of the company are Messrs. A. T. Robl, H. Rosales, T. D. Merton, W. R. Warren, and Colonel Elliston.

The Tasmanian Smelting Company.

The Huntington-Heberlein Process.

The works of this company are situated in the centre of the rich silver lead ores of Zeehan. The site was chosen about a mile from the town and along the Zeehan to Strahan railway. The works have branch sidings which connect with the main line. A quarry with dark blue limestone and another alongside containing siliceous flux serves further to explain the selection of the site, which from these points of view is a most suitable one. A great outcrop of manganic ironstone deposit, containing a little silver, is held by the company under lease from the Government, and serves to supply the necessary iron flux. From this deposit also the ironstone for the North Mount Lyell smelters was drawn.

The ore as delivered at these works, which were erected for the treatment of purchased ores, passes over a Howe's weighbridge, which automatically records the weights; thence it passes to the sampling works. Sulphide ores are crushed in a stonebreaker of the Blake type, having jaws 15 inches by 9 inches. It then passes through two sets of Reliance rolls, the crushed product passing through screens. Two elevators carry it to the upper levels; one portion is taken for sampling, the balance going by gravity into the ore bins. The sample is taken so that 1-32nd, 1-64th, or 1-126th part of the stream passing may be deflected. This is discharged in iron buckets and goes to a drying room. The whole of this is ground still finer, and then quartered down in the usual way. For the oxidised ores from 1-5th to 1-10th is taken for sampling after the material has been crushed. This also is broken down fine and quartered, the final sample for assay being chosen from this.

The next operation differs from that carried on in most smelting works. Instead of picking out the sulphide ore and roasting it separately, a mixture is fed into the roasting furnace, which serves as a basic flux when the material has to be smelted. The ore is not all in a fine state of division, but most of the coarse grains of galena decrepitate and become powder on being subjected to heat.

The roasting furnaces consist of two circular mechanical and two hand reverberatories. The former have a circular bed 15 feet in diameter, which revolves. The rabbling is done by two fixed arms being provided with blades so placed as to deflect the ore towards the circumference. A small strip on the outer edge is worked off by the outer blade. This has been heated to bright redness. It falls down a chute into a vessel prepared to receive it. From a roasting point of view, very little can be said in favor of these furnaces—the depth of the charge is too great and the rabbling is insufficient to give anything approaching a good roast. Arsenic and sulphide of arsenic distil off readily enough from arsenical pyrites and sulphur from ordinary pyrites. The upper layers may become partly roasted, but galena, blende, and such compounds as are only roasted with difficulty cannot be roasted unless kept in a furnace such as this for a prohibitive time. Fortunately any crude rough roast will do on account of the subsequent operations.

The reverberatory furnaces are of the usual type seen at smelting works. They have nothing to be said in their favor since they are clumsy, cumbrous and ineffective, and there is little wonder that the cost of obtaining a good roast is excessive. Where furnaces have to be hand-rabbled there is a limit to the width and length over which effective work may be done. This is invariably exceeded on smelting establishments, while the height from floor to crown is ridiculously in excess of what it need be. The amount of sulphur which is left after a so-called roast in such furnaces is often so great that the roasting need hardly have been done at all. In cases where plenty of oxidised ores existed, the smelter was not troubled to any great extent with his sulphide ores, but when lead had to be produced from a galena intermixed with blende, the roasting trouble cropped up, followed by others, such as an excessive quantity of low grade matte, an almost infusible slag and a pasty matte. In order to remove the objectionable sulphur, better furnaces might be designed or advantage taken of certain chemical reactions. Well-designed hand reverberatory furnaces do not seem to exist outside the range of the metallurgy of gold. The reaction mainly relied upon was due to the interaction between sulphate of lead and silica at a high temperature, sulphur dioxide and oxygen or sulphur trioxide being evolved, and silicate of lead forming. This has also the advantage of fritting or sintering the various materials in the furnace, and renders them suitable for feeding into the blast furnace. Should reducing gases be present the sulphate of lead is converted back again into sulphide, with which silica will not react.

In the treatment of concentrates, such as are produced at Broken Hill and also to a lesser extent in Tasmania, trouble has arisen on account of the presence of blende. Roasting converts portion of this into oxide, but a considerable quantity is oxidised to sulphate. Lead sulphate is also produced from galena to the extent of 30 per cent. The result is that when these are fed into smelting furnaces where reducing agents are in excess these sulphates are converted back again into sulphides, the operation reminding one of the gallant Duke of York, who marched his men right up to the top of a hill to march them down again. When sintering with silica is adopted the sulphur may be eliminated from the sulphate of lead, but sulphate of zinc is not decomposed wholly at a white heat, and even in the presence of silica will not behave like the lead. This no doubt is largely due to the infusibility of the original products and also to the fact that silicate of zinc in itself is infusible. When the sulphate is fed into the furnace it mainly becomes sulphide, and as such is highly objectionable. It passes partly into the matte, and partly into the slag, rendering both less fusible and more difficult to separate. As oxide of zinc it would be far less objectionable, for in that case none will pass into the matte, while it appears to take the place of some of the lime in a slag, and a suitably proportioned slag can absorb large quantities of it. These points are mentioned as showing only a few of the troubles which arise through not being able to eliminate almost the whole of the sulphur preliminary to lead smelting.

It was reserved for Huntington and Heberlein to point out a new way to eliminate the sulphur and the importance of their discovery cannot be overestimated. For some considerable time it has been known that oxide of lead heated in the presence of lime, soda or the oxides of any of the alkalies, or alkaline earths, will absorb oxygen and combine with the more strongly basic materials forming a plumbate of the corresponding base.

$$CaO + O + PbO = CaPbO_3.$$

The fact that galena in presence of salt in a reverberatory furnace will also form a plumbate has also been known, one of the reactions being—

$$2NaCl + PbS + 5O = Na_2 PbO_3 + SO_2 + 2Cl.$$

But it was not known that galena, heated with lime, would, when a current of air was passed through the heated mixture, be deprived of practically the whole of the sulphur. It would appear that work was first carried out in Italy, for in 1897 the process was patented. Its adoption by large smelting works is only a question of time. The discoverers of the process found that by mixing galena and lime together, and heating to about 700deg. C., and then allowing to cool to a dull red heat that oxygen was given off. The oxygen given off appears to attack the galena, and transform part of it into oxide, sulphur dioxide being evolved at the same time, while a considerable amount of heat is generated. At this point, if air is forced through the mixture, the operation becomes continuous, and the temperature is maintained at such a point that reaction takes place between the sulphides and the sulphates, the oxygen acting on the galena so long as any sulphur remains in the mixture. Concentrated fumes of sulphur dioxide are evolved, the temperature rises, and the mixture gradually fuses to a mass of lead oxide in conjunction with the gangue of the ore treated. The patentees of the process believed the reactions which take place to be according to the following equations, although they admitted that this might not be the correct explanation:—

At 700deg. C.

$$(1)\ CaO + O = CaO_2.$$

At 500deg. C.

$$(2)\ 4CaO_2 + PbS = PbSO_4 + 4\,CaO.$$

Or combining (1) and (2)

$$4CaO + PbS + 4O = PbSO_4 + 4CaO.$$

In other words, they held that lime heated to about 700deg. C. absorbs oxygen from the air, and becomes a dioxide, but that on cooling this oxygen was evolved. If, however, sulphide of lead were present, then as the mixture cooled down the oxygen evolved from the dioxide converted the lead sulphide into lead sulphate, the lime returning to its original state. After this change, it was held that at fusing point

$$(3)\ PbS + PbSO_4 = 2PbO + 2SO_2.$$

As written thus: (3) would be unmeaning, supplying oxygen to the left-hand side would complete the equation.

$$PbS + PbSO_4 + O_2 = 2PbO + 2SO_2.$$

In other words, sulphide of lead in the presence of sulphate and oxygen would give oxide of lead, and sulphur dioxide would be evolved. The reactions, as here given, do not give a satisfactory explanation of this interesting process. In the first place, calcium dioxide does not decompose in the manner stated; that is, it exists at temperatures far below 400 C., and is decomposed on even gently heating it. Certainly it partakes of the properties of the alkalies and alkaline earth metals to form a peroxides when heated in dry air or oxygen. Lead oxide also possesses this property, and as is well known will form a higher oxide when heated in the air at a temperature not exceeding 450deg. C. It is most probable that both oxides when formed unite to form a plumbate. The third reaction, as given, is also unlikely as causing the generation heat and the evolution of sulphur dioxide. Unless the two compounds were in molecular proportions, as indicated by the equation, a considerable quantity of sulphate of lead should be left unchanged. The formation of compounds of calcium, other than the peroxide, have been ignored, and yet it is certain they play an important part in the transformation. The reaction between lime, or even calcium carbonate and sulphides, does not appear to have been considered to any appreciable extent in dealing with smelting. A simple experiment of heating lime and any sulphide in the air will at once show that a sulphite of calcium at once forms, and in many cases where there is an affinity between the oxide of the metal and the alkaline earth, a compound will at once form. For instance, if stibnite is mixed with lime and roasted, calcium antimonite and calcium sulphite will form. On further roasting the higher salts, calcium sulphate and calcium antimoniate, will be formed. When galena is roasted with lime, one of the products formed is calcium sulphate, and a small quantity of calcium plumbate.

The Carmichael-Bradford process, which was announced some few months ago, practically makes use of the same reactions as must occur, though not stated by them, in the Huntington-Heberlein process. The object of this process also was to get rid of the sulphur from zinc, copper, iron or lead sulphides. The raw sulphides are mixed with calcium sulphates, and introduced at about 400deg. C into a converter, and a current of air passed through. The galena and calcium sulphate are said to react, giving calcium sulphide and lead sulphate.

$$CaSO_4 + PbS = CaS + PbSO_4.$$

When air is passed through this mixture the calcium sulphide is said to be oxidised, and this oxidation generates sufficient heat to cinerate the ore. Another reaction also takes place, giving calcium plumbate and oxides of the various other metals. This explanation cannot be looked upon as final, but it is evident a link missed out in the original explanation has been picked up, or the Huntington-Heberlein process starts with raw limestone or lime, and obtains a final product free from sulphur. The other starts with one of the intermediate products, which undoubtedly form in the Huntington-Heberlein process, but which was not indicated by them, and arrives at the same final stage. The original patentees found that ferrous or manganous oxides may be used in place of lime or earthy bases, but they never seemed to have recognised the full importance of the method for sulphides other than galena. No explanation was given why blende loses its sulphur completely, and also becomes transformed into oxide of zinc, yet this result is even more important than the change in pure galena. The fusing of the product is also important, for fine ores may be changed into masses suitable for feeding into a blast furnace, and thus briquetting is avoided. At Broken Hill, where the process was adopted after a trial with 12,000 tons, it was found that better results were obtained than by the old method of roasting and briquetting. The Tasmania Smelting Company was about the first in Australasia that adopted the new process.

The vessel or converter in which the operation takes place is simplicity itself. An inverted sheet iron inverted cone is suspended on trunnions. A perforated plate or colander is placed as a diaphragm across the apex end of the cone, which is about 5 feet 6 inches in diameter and 5 feet deep. The heated ore from the roasters drops into the conical vessels, and when sufficient has been introduced air is turned on underneath the perforated plate, and blown at a pressure of about 17oz. through the mixture. The temperature at once rises, and sulphur dioxide is given off in great abundance, and carried off through a hood. The mixture fuses in course of time, and is transformed into a solid semi-fused mass, which may be tipped out of the converter. The whole operation lasts from two to four hours. Three hours may be taken as the average time required for the conversion. The semi-roasted material on entering the converter may be taken as containing 10 per cent. of sulphur, but the fused mass on leaving only contains 1 per cent.

There are 12 small converters for the two mechanical roasters, and two large ones for the reverberatories. The size of these vessels could no doubt be increased with advantage. The semi-fused lumps from the converters are mixed with the requisite quantity of silica or siliceous ores, and smelted in a blast furnace. There are two of these for lead smelting, each capable of running down 80 tons of ore per day. They measure 2 inches by 120 inches at the plane of the tuyeres, and are higher than most lead smelting furnaces, being 25 feet, with a charge column of 21 feet high. They are provided with double tuyeres, and supplied with air at a pressure of 20oz. The lead and matte are tapped off in the usual way, but the slags are granulated and swept away by a stream of water. The molten slag on falling is struck by a horizontal jet of water, and is at once chilled and broken up into small grains, which are swept down to the slag dump. This innovation, which shows that the managers, Messrs. Kapp and Kunze, have confidence in their work and are certain of the poverty of their slags, also means a great saving in labor. The lead bullion as tapped, also the matte produced, is shipped away for softening and desilverisation or other treatment necessary. A small furnace for the smelting of copper ores was in course of erection at the date of my visit.

A fine Reynolds-Corliss engine of 125 horse-power, another of these being in reserve, serves to drive the two No. 7½ Roots' improved high-pressure blowers. These blowers furnish 87 cubic feet of air at each revolution and are guaranteed to work up to 5lb pressure. In spite of all one hears about the marvellous metallurgical work done in Germany, it is a pleasure to hear that the ores dealt with at Zeehan can be treated more cheaply on the spot than if shipped to Freiberg. Cheap labor in Europe tends to lower costs, and permit of the handling of lower grade ore than can be dealt with at present in Australasia. The detailed treatment of metallurgical products can also be carried out to greater perfection where the supply is constant and the market for the finished article at one's door. The absence of any detailed treatment means a lot of waste on the West Coast. It is also noteworthy that in all the smelting plants very few of the original schemes for improvement have come from Germany. Nearly all labor-saving appliances and progressive schemes are due to Americans. The furnaces and plant at these works are no exception to the rule.

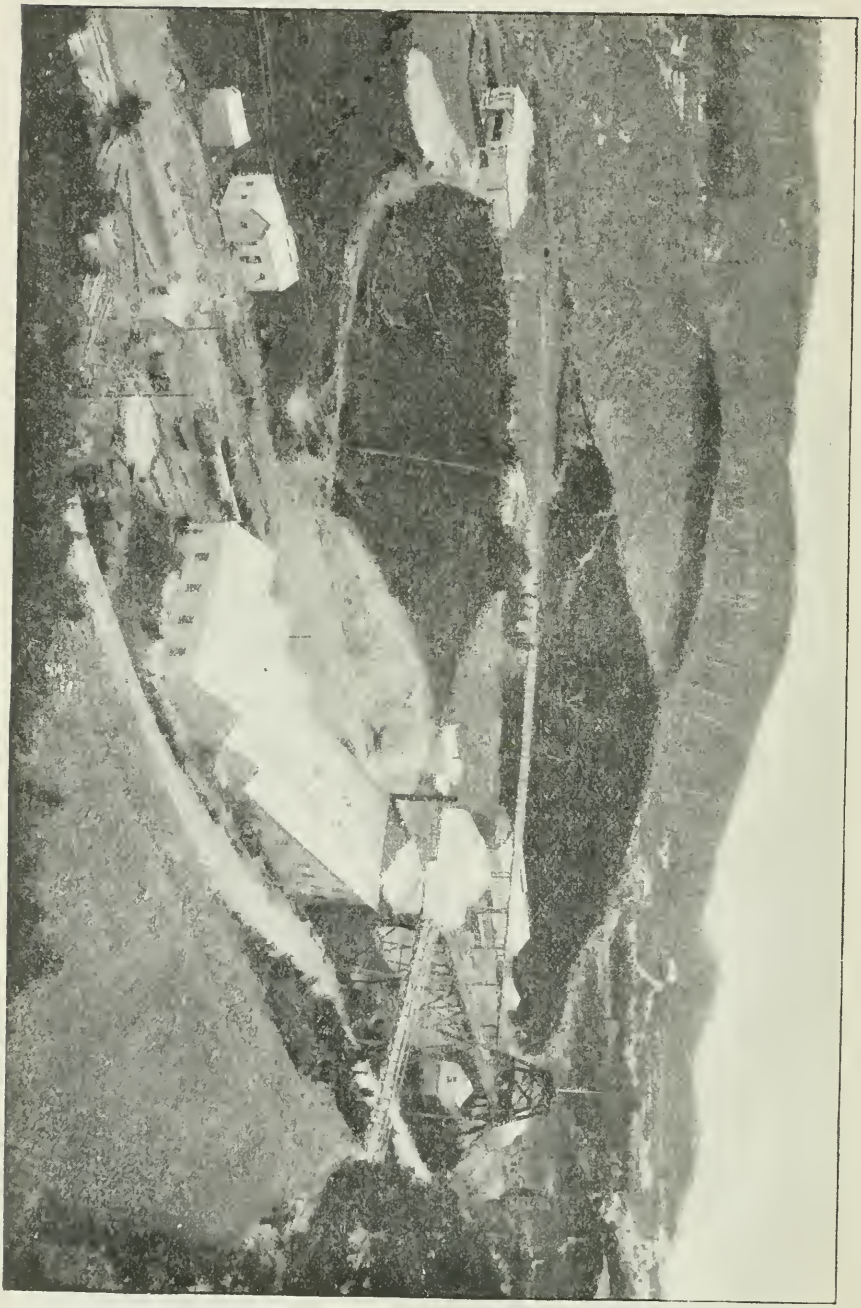
Zeehan-Montana Mine and Plant.

There is no field in Australia more interesting from a geological or mineralogical point of view than Zeehan. The geology of the field has been well described in the reports of Messrs. Montgomery and Twelvetrees, while careful detailed work is now being done by Mr. G. A. Waller. On the whole the lodes may be said to be worked for galena, but blende may largely predominate in one set and carbonate of iron in others—while such uncommon minerals as stannite, with bismuthnite, are worked in the middle of the field. The country in which the lodes lie may be said to consist of silurian sandstones, quartzites, slates and limestones, and a melaphyre, locally termed white rock. This melaphyre or ancient basalt varies in character from a hard and compact to a soft rock. Mr. Twelvetrees holds that this is of the same age as the silurian rock, or, in other words, that this material poured out and overflowed the horizontal silurian strata of the time. Several flows appear to have taken place, thus interbedding some of the sedimentary rock, after the manner of our own basaltic flows which occurred in Tertiary times. Subsequent to the outburst of the melaphyre the rocks were subjected to a pressure from the N.E. and S.W., which resulted in their being folded at right angles to this, or N.W. and S.E. It is easy enough to understand that along a line of contact of two dissimilar rocks that fissures and faults would be of frequent occurrence, and that along these fault planes, metalliferous solutions would flow. The lodes occur in both silurian and melaphyre country, but so far there is not sufficient to warrant any statement about variation in values when a lode passes from white to black rock.

The time at my disposal did not permit of any exhaustive examination of the mines on this field, but from what was seen, it is quite evident that most of the mines are only down to very shallow depths; that for the most part the rich ore was rooted out regardless of development, or, in other words, the goose was killed that laid the golden egg. The tendency has been of late to amalgamate the smaller mines, and provide sufficient capital to systematically open up and develop before troubling about dividends. The policy of letting strips of the leases to tributors on favorable terms has also led to important discoveries, which have benefited both miner and company. Taking all matters into consideration Zeehan has just commenced to be a mining field; the boom days are over, and whatever money is earned must come out of the mines, which no doubt will be worked in a more systematic and progressive way than during the boom period. One is at a loss to understand a system of mining followed at the Western, where No. 7 level is only down 430 feet. No. 1 level was 45 feet; No. 2, 110 feet; No. 3, 170 feet; No. 4, 230 feet; No. 5, 290 feet; and No. 6, 360 feet. Were all these levels on rich ore, very little might be said against them, but when the plans are viewed it may be seen that miles of unproductive crosscuts were driven, and the tracks of lodes followed at such short distances. It is not to be wondered at that this rich mine, which paid for a splendid plant and for all development and distributed very large sums in dividends should now be waiting for capital to develop it.

One of the mines which did not adopt the hand to mouth style of working is the Zeehan-Montana, and its progress and developments has infused new life into some of the other properties. The company started work in 1893, and gradually increased its holding until now upwards of 300 acres are held. The lease includes both slate and white rock, and the lodes are worked in both, though mainly in slate. Through the courtesy of Mr. T. Craze, the manager, I was enabled to inspect the deeper levels, No. 4 and 5. In both those levels a number of lodes have been followed, some of these in the volcanic rock, some in the sedimentary and some in both. The strike varies to either the east or west of north—that is to say, the lodes as a rule are not parallel, but run in converging directions. The underlie is nearly vertical. The lodes themselves are easily followed, for there is a distinct marking off from the country on either side. In places they are filled for from a few inches up to 3 feet with solid galena, but for the most part they contain a gangue which mainly consists of carbonate of iron. So far as I was able to see there was very little zinc blende or pyrites in the material. The country is very much broken and work rendered much more difficult by the presence of numerous faults, which have shifted the lodes for considerable distances. Prospecting is absolutely necessary even along the lines of lode, for blanks may occur for long distances at one level, and yet the next be very productive. Every galena leader must be followed up, otherwise very important enlargements would be missed.

By keeping several levels going, Mr. Craze has been able to turn out good ore all the time and keep his mine well opened up. The utmost care is used from start to finish in keeping the ore clean, and this largely accounts for the good work done at the mill. The solid galena broken from below is carefully hand picked underground, sent to the surface, broken into convenient sizes and bagged. The poorer material containing galena is sent up for concentration. The former amounts to about 125 tons per month, which assays 63.5 lead and 100oz. silver per ton; the latter amounts to nearly 1400 tons, running 7.20 per cent. lead and 12.1oz. silver per ton, and zinc 5 per cent. The lower grade ore is trucked direct from the mine to ore bins holding 90 tons. Thence it passes to a grizzley made of 3-8 inch by 2½ inch bars, spaced 7-8 inch apart. The bars are set at an angle of 50deg. The fines drop through and fall into a chute, down which they are carried to No. 1 trommel by a small stream of water. The coarse material passes into a stone breaker capable of crushing 7½ tons per hour down to 1 inch gauge. The broken ore falls on a shaking table fitted with a perforated plate having 7-8 inch holes. The fine ore drops through this and falls into the same chute as the fines from the grizzley. The coarse ore is delivered on to circular revolving picking table. This table slowly revolves, and the coarse product is evenly delivered on to one portion of it and carried around—any pure pieces of galena are picked out and any pieces of worthless gangue rejected. The material then passes automatically from the table to the coarse rolls. The rolls are 24 inches diameter and 14 inches face. They revolve about 8 times per minute, and



The Montana Mine.

are tightened up in the usual way with set screws and spiral spring. They are set so as to crush the ore to 13 m.m. diameter, the product from the rolls passing into the same chute as the fines from the grizzley. Samples for assay are taken at this point. The whole of the material is now sized by means of trommels. There are five of these. The first has steel punched plates, having holes 13m.m., the second 9m.m., the third 5m.m, the fourth 3m.m., and the fifth 1½m.m. The whole of the ore passes into No. 1 trommel, and all pieces over 13 m.m in diameter are delivered out at one end and pass to the roughing jig. This is after the style of the Hartz jig, with side pockets. The bed of ore is kept 6 inches deep. The plungers are run at 90 revolutions per minute, with 3 inch stroke. The galena works to the bottom of the bed, and travels forward over the cast iron framework which supports the plates until it reaches the side pocket, where it is delivered. Any fine particles of ore which may have got on to this jig pass through the jig plate, the screen having 3m.m. diameter perforated holes. The tailings from the jig pass into an elevator, and are passed back to the rolls—where they are crushed still more finely and pass through No. 1 trommel to be dealt with by other jigs. The first jig products assay 66.5 per cent. lead and 100oz. of silver per ton. The material which passes through the mesh of No. 1 trommel passes to trommel No. 2. The fines pass through the screen as before, the coarse going to a 13m.m. jig. This jig has two compartments, and makes two products, the first being clean ore and the second being marketable—both are bagged. The average assay of this material is 65 per cent. lead and 80.8oz. silver per ton. The tailings go back to a set of fine rolls, where they are crushed and treated with finer jigs. The material which will not go through No. 3 trommel is sent to the 9m.m. jig, which has three compartments. The concentrates from the first compartment assay 69.5 per cent. lead and run 84oz. of silver per ton. The concentrates from the second and third compartments go to the fine rolls for re-crushing.

The coarse material from the next trommel passes to the next jig, whose first compartment product assays 84oz. silver and 66.5 per cent. lead. The second and third compartment materials are returned to the fine rolls. The last trommel separates out ore and gangue particles over 1½m.m. in size. These pass on to the 3m.m. jig. The first concentrates assay 67.5 per cent. lead and 84.8oz. silver per ton, the second and third products also being sent to the finer rolls. The material which passes through the mesh of the trommel, being less than 1½m.m. in diameter, is sent to a V-shaped classifier. Water under pressure is let in from below; the slimes are washed off, and the coarser particles fall to the bottom and are carried away for treatment by a jig which returns a product from the first compartment assaying 62.5 per cent. lead and 70.4oz. silver per ton. The second and third compartment material are re-crushed. The sand which is swept over with the slimes from the hydraulic classifier is dealt with on the fine side of the plant, the treatment of the coarser particles being now complete.

The tails from the 13m.m. jig and the second and third products from the next four jigs, consist of specimen galena and other minerals, as blende, pyrites or carbonate of iron, whose size and weight gives them about the same resistance as the galena specimens. The fine rolls used for recrushing this material are the same size as those used for crushing the coarse stuff, only that they are set closer and driven more slowly. The crushed material passes into a trommel having 4m.m. holes, a second trommel 3m.m. holes, followed by a third with $1\frac{1}{2}$m.m. holes. The coarser material from the first trommel is carried back for re-crushing in the rolls. The coarse material from the second trommel is delivered on to a three compartment jig, the second and third products being returned for recrushing. The assay of the concentrates from the first compartment is 47.5 per cent. lead and 54.4oz. silver per ton—the lower value being said to the presence of zinc blende and other materials finding their way into the products on the fine side. The material between $1\frac{1}{2}$ and 3m.m., which escapes from the third trommel, is likewise treated, the product from the first compart-

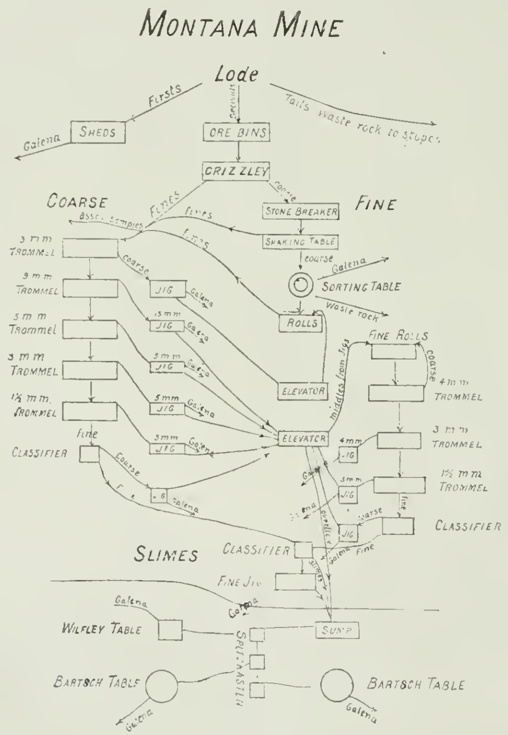


Diagram of Plant.

ment assaying 52.5 per cent. lead and 58.4oz. silver per ton. The material less than $1\frac{1}{2}$m.m. which passes through No. 3 trommel goes to a hydraulic classifier of the same type as that described. The coarser particles go to a 4 compartment jig. The first compartment assays 42 per cent. of lead and 46.4oz. of silver per ton. The product from the other compartments pass along and meet the slimes from the coarse side of the plant. The whole of these flow into a common sump, and are then elevated into a series of spitzkasten. The spitzkastens are 3 feet, 4 feet 6 inches, and 6 feet deep respectively. The sands drawn off at the bottom of the first go to a Wilfley table which is run at from 230 to 240 strokes per minute. The clean heads of galena are caught on one end of the table, but the seconds are not returned by the elevator wheel since Mr. Craze has found that better work could be done by dealing with this product on a separate machine. The heads from the Wilfley assay 51.50 per cent. of lead and 57.6oz. of silver per ton. The pulp from the second and third spitzkastens is drawn off and fed on to two Bartsch tables. These are circular tables made of cast iron and are 13 feet in diameter. The tables themselves are stationary except for a percussive blow which is meant to settle the heavy material from the pulp on to the table. This by the way seems to have been taken off all the tables at Zeehan, for they are run like the ordinary dressing tables. The novelty about this appliance is that the feed and water pipes, which are curved, deliver the pulp on to the surface of the table while the tail end of the curved water pipes sweeps the concentrates into a launder, which revolves at the same rate. The seconds are caught in another portion and the tails go to waste. Although the iron surface when planed up gives a perfect means of retaining fine heavy slimes, it is difficult to prevent it from rusting to a certain extent, or from being pitted by the acid waters and sharp sand falling on and continually flowing over it, so that in course of time it offers no advantage over some cheaper material, such as a tough cement or even linoleum. The products from this machine assay 52.5 per cent. of lead and 54.12oz. silver per ton. Mr. Craze finds that these tables are too expensive and slow to be worked on a material which has only a low value when saved. Each machine only treats from 4 to 5 tons of material in 8 hours. The cost also of each machine is high, amounting to £350. The summarised results as given in an excellent paper by Mr. Craze of the work done by the various machines is as follows:—

COARSE SIDE OF PLANT.

No. of Jig.	Size of Ore. Millimetres.	Lead Assay. Per cent.	Silver Assay. Per cent.	Silver per unit of Lead. Oz.
1	21 to 13	66.50	100.20	1.52
2	13 to 9	65.00	80.80	1.24
3	9 to 5	69.50	84.00	1.21
4	5 to 3	66.50	80.00	1.20
5	3 to 1½	67.50	84.80	1.27
6	1½ to 1	62.50	20.40	1.13
FINE SIDE OF PLANT.				
1	4 to 2	47.50	54.40	1.14
2	2 to 1½	52.50	54.40	1.03
3	1½ to 1	59.50	58.40	0.98
4	1	42.00	46.40	1.10
Wilfley	1	51.50	57.60	1.11
Bartsch	Fine slimes	52.50	54.12	1.03
Crude ore	—	7.20	12.71	1.76
Coarse tailings	—	1.01	2.80	2.77

It would appear from the results of the work done here that nearly 2-5th of the material bagged does not go near the concentrating plant at all, and that the whole of the trouble is to extract the remaining 3-5th of the total value. The material treated, amounting to about 32 tons per shift, is on the whole favorable for concentration, for the gangue is comparatively soft and the galena is not intimately associated with blende, making it unnecessary to go in for the finer crushing so essential at Broken Hill. Any carbonate of iron left in the galena concentrate is not viewed by the smelter as blende and barite would be looked upon. Touching the plant generally, it might be said that with regard to classification by size if the numbers of the mesh of the trommels were closer and corresponding jigs used that better work would be done, but of course there is a limit to the profitableness of such work, depending on the amount of material to be put through. The same applies to the crushing of products and returning them to the same trommels, but in this common business principles must also be applied. The interest and upkeep of the extra plant would amount to more than the saving effected. While the plant is not so elaborate as many of those advocated by German authorities, yet there is less of the small capacity machines, such as Luhrig vanners and less slime making than occurs in most ore dressing establishments. Taken altogether it may be said that the scheme of working for this particular ore has been well thought out, and that the results obtained amply justify the whole plant. Several visits were made to the ore dressing sheds, and always with the same result that every machine was seen to be performing its work in a most satisfactory manner. In this case also the man behind the machine is a more important factor than the machine itself.

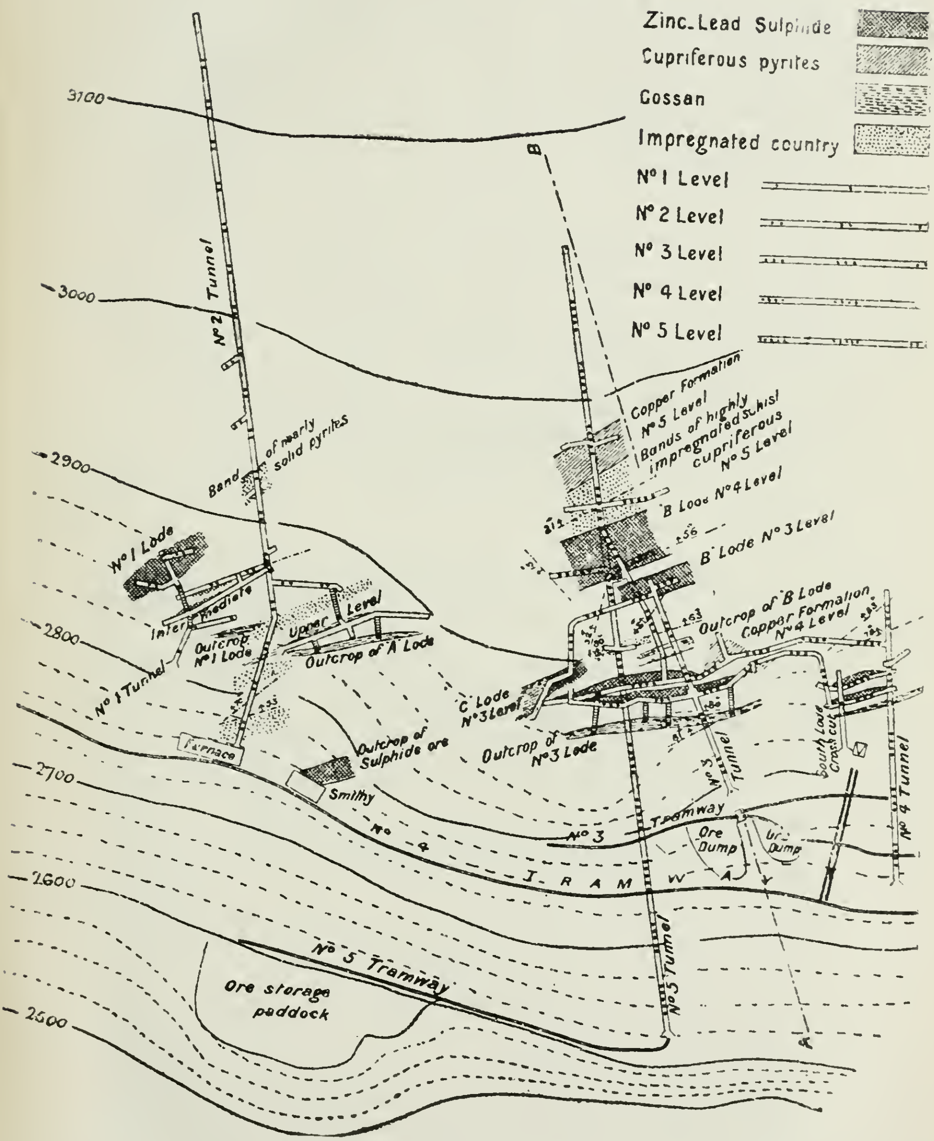
Zeehan Mines.

The greatest puzzle on the West Coast is to find out why a mine with the record of the Western Silver Mining Company shut down. Through the courtesy of Mr. Aug. Simson I have been enabled to see the accounts and reports from the starting of the company up to the time it closed down. The inspection of the plans also makes the problem all the more difficult to solve. The galena lodes were discovered in 1888 by a prospecting party; shareholders in the syndicate who sent the party out formed a company with 60,000 shares of five shillings each, and provided that one-third interest, or 20,000 shares, should be sold at 5s. per share on the formation of the company. Most of these shares were subscribed for by the existing shareholders, and £4750 was placed to the credit of the company. Work proceeded slowly, the directors preferring to wait the completion of the Zeehan-Strahan railway. In 1893 a magnificent concentrating plant was erected by the agents of the "Luhrig Ore Dressing Appliances, Ltd." This plant could deal with 50 tons per shift. No call was ever made, and dividends were paid almost immediately when progressive work was started. Up to 1901 the mine paid £102,000 in dividends, or 34s. per share on the whole of the 60,000 shares, and up to that year had obtained £466,659 net for the ore sold. Four or five sixpenny calls exhausted the belief of many shareholders in this wonderful mine, and shares were being forfeited wholesale when calls had to be paid. The directors viewed this with alarm, and fled to the Government for assistance unavailingly. The result was that the mine was shut down, while the shareholders are endeavoring to persuade outsiders that it still would give good returns if capital were put into it. In a previous article reference was made to levels put in, five of which only average 60 feet apart, and there is no doubt that this class of work in prodigal times left the mine undeveloped when the pinch came. It is admitted that the lodes at Zeehan are somewhat patchy in their mineral contents, and difficult to follow in faulted country, but these very drawbacks should impress the management all the more forcibly with the necessity of keeping development works well ahead. The stopes of rich ore to-day may cut out to-morrow and vice versa. The ground is for the most part most favorable for mining operations, but this is a lame excuse for having 13 miles of levels and drives when half that number would have sufficed. It is quite evident that money paid in calls would only have been frittered away. It is rarely that a progressive policy is adopted when monthly calls are on, so it would appear that the only chance of a new life to this mine would be by providing sufficient capital to put the mine in the position it should have been kept in when active work was going on. The deepest level in the mine is only 600 feet, while the most of the ore was obtained very near the surface: and most of the driving and crosscutting was done at such short intervals in the upper levels that the lower ones were neglected when trouble loomed ahead.

The plant at the Western is well designed from a Luhrig Company point of view, but other things being equal, I do not consider it would do much closer work than the Montana. The same system of sizing the coarse material by means of trommels is adopted, but the Luhrig jigs have five compartments. The product from the first as a rule is bagged; from the second repeated, while that from the third was crushed at a battery, the fourth and fifth going to waste. On this class of ore the two last compartments do not seem to have been necessary. The most radical departure from the system adopted at the Montana mine is the introduction of a 10-head battery with 260lb. weight stamps to crush the middle products containing specimen galena. This certainly is a great mistake from a concentrating point of view; but the introduction of a great number of Luhrig vanners seems to have been the main reason for the production of a large quantity of slime.

Some excellent returns are being obtained by tributers on various parts of the field. One party on the Spray Hill, working a tribute from the Argent, was bagging up an ironstone carrying over 1000 oz. of silver per ton. The metal was present as spongy native silver, chloride, and also chloro-bromide. There was a small percentage of antimony oxide in the ironstone, but the seconds, which were rejected, contained up to 15 per cent. of antimony as oxide, and also carried silver. It seems surprising that material of this grade is rejected at such a place.

Other parties lower down the hill were getting out galena and dressing it up in hand jigs. They claimed that better results could be obtained than by putting it through the elaborate ore dressing plants on the field. The methods of operation were simple, and yet in all the lead producing parts of Australia I have never seen similar work done. On many fields producing rich patchy galena on the surface much profitable work could be done. The ore as broken down into small lumps was thrown up against a screen made of ½ inch bars spaced one inch apart and set at an angle of 50deg. The finer material passed through to be separately dealt with, the coarser material was picked over and the clean metal selected. Large enough specimens were broken and the waste rock rejected. About 1 ton could be screened and picked in a day. The fine material went to a hand jig. An outer box of about 6 feet by 3 feet by 2 feet deep was supplied with clean water at one corner, and provided with an overflow on the opposite end. A box about 18 inches by 30 inches by 9 inches deep has a sieve on the bottom which varies in size according to the material (either ¼ or 3-15th inch) dealt with. A bottom of coarse galena 3 inches deep forms the bed. A row of holes is bored horizontally some inches above this to facilitate the escape of light sand. The box is hung by suspenders from each end on a horizontal bar. This in its turn is rigidly attached to a long spring pole which acts as lever for lifting the box. The fulcrum is supplied by a round axis which turns, gallows fashion, in a pair of upright forked props. The length of the pole to the free end from the fulcrum is about 12 feet. Beneath this end is a horizontal spring board on which the operator stands. A few shovelfuls of the screenings are put in the box; the operator, just able to reach the end of the spring



Sketch Plan of the Hercules Mine.

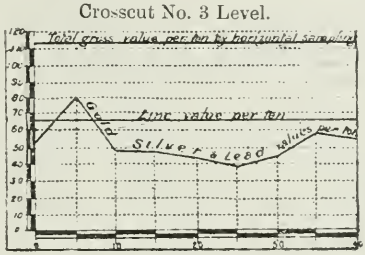
pole, throws his weight upon it, and jerks it down—the weight of the box, the resilience of the spring board on which he stands, and the pole all tend to reverse the stroke. The next downward stroke is made so as to synchronise with the movement of the pole and board, and the work thus rendered much easier than if rigid materials were employed. With the downward stroke water rushes through the screen at the bottom of the box, between the bedding, and raises the ore fed in. On the return stroke the heavier particles rapidly settle, leaving the lighter ones above. By repeating the strokes a heavy concentrate lies on the bed, and a worthless material above. The top layer is scraped off, and the bottom bagged up. The overflow and fine material which passes into the box below is dressed on a square buddle or inclined plane, with a box at the upper end. The fine material is evenly distributed over the head of the buddle, clear water allowed to flow over, the material swept lightly upwards to allow any gangue that may have settled to be brought under the action of the running water. The heads from the buddle are bagged up from time to time. From 10 to 12 bags can be dressed per day with the hand jig and buddle. It is somewhat surprising that such manual operations are relied upon more than the ore dressing appliances to be found in Zeehan, which by the way, for their particular class of ore are unequalled in Australia. The presence of so many of these hand-jigs on such a field as this must be taken as showing the advisability of pushing hand picking or metheds of simple selection as far as possible.

The complex ores around Zeehan afford a fine opportunity for the future metallurgist to distinguish himself. The zinc ores at the Comstock are fairly pure and blende is now hand picked and exported at a profit; but at Mount Reid, where the Hercules and Mount Reid mines exist side by side there are enormous quantities of complex sulphides containing copper, lead, zinc, gold and silver, whose market value in pure metals would run into millions of pounds, but whose present value as ores is a negative quantity—in other words, it would cost more to extract the metals than they are worth. The Mount Reid mine, on which work has been stopped, has an open quarry on a body of ore lenticular in shape, some 800 to 900 feet in length, and whose width at the widest part is about 80 feet. For upwards of 30 feet across there is a solid mass of compact sulphide minerals without one particle of gangue. On the footwall side are layers of quartzite and gossan. The work done on this mine has been to open up a quarry 80 feet wide and 40 feet deep for a length of a few hundred feet. This is said to have cost the company £60,000. One peculiarity of these mixed ores is that the gold seems to be associated with blende, and the silver mainly, as is usual, with galena. The Mount Reid and Hercules ores, rich in zinc, will go to upwards of 10dwt. gold per ton when taken in large quantities. The Hercules mine contains enormous bodies of mixed sulphides, and has been opened up by means of tunnels, which may be put in for some hundreds of feet below the lowest. The ore bodies in this are also lenticular in shape, and are interbedded in an indurated slate. This, as well as Mount Lyell, appears to be due to the action of hydrothermal solutions widening existing fissures and removing minerals from the original slates, replacing them with metallic sulphides.

The sketch plan and sections, taken from Mr. Waller's report, serve to show the size of the ore bodies and their variable nature. No. 1 tunnel was driven to intersect an outcrop of gossan, called the A lode. Further on the No. 1 lode was intersected. This was exceedingly rich, for 9 tons of gossan gave an average of over 10oz. gold and 212oz. silver per ton; and 62 tons of sulphide gave 2oz. gold and 68oz. silver per ton. Most of this rich material has been removed. The A lode supplied rich gossan for 120 feet in length by 10 feet in width. No. 2 tunnel was driven nearly 100 feet below No. 1 for some hundreds of feet, but no ore of value was struck. No. 3 tunnel was then started some 250 feet further south at about the same level as No. 1. This intersected a sul-

Crosscut No. 3 Level.

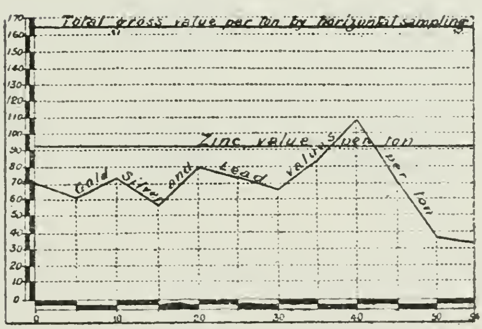
Scale of Shillings.



Scale of Feet across Lode.

Crosscut No. 4 Level.

Scale of Shillings.



Scale of Feet across Lode.
Basis of Calculation.—Gold, 4s. per dwt.; silver, 2s. 6d. per oz.; lead, 2s. 6d. per unit; zinc, 3s. 6d. per unit.
Lode Sections, Hercules Mine.

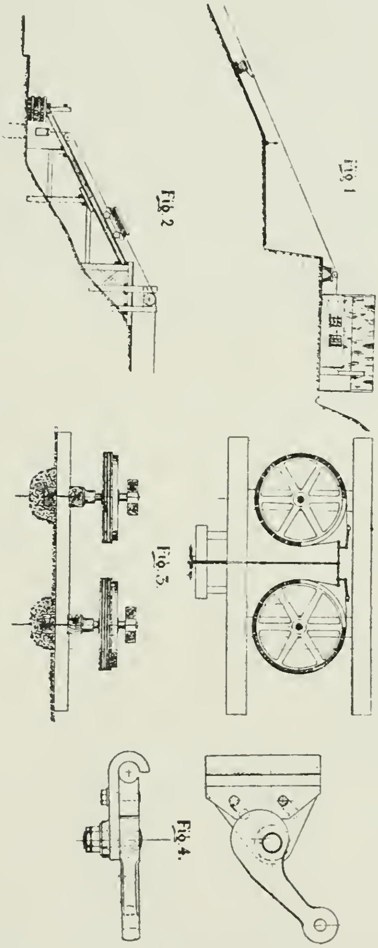
phide lode known as No. 3 flanked and capped with gossan. Further east a solid lode of pyrites was struck—called lode B. This has a capping also of gossan. This was driven along for some distance in a northerly direction, then brought back westerly to intersect No. 3 lode again. No. 4 tunnel is about 200 feet further south than No. 3, and about 50 feet below it. This was driven to intersect the south lode. This shoot is about 70 feet long, averaging 5 feet thick. The ore has a value of gold 10dwt., silver 33oz., zinc 19.8 per cent., copper 0.59 per cent. No. 5 tunnel is driven under No. 3, and 125 feet below No. 4. This has been extended east beyond the others, except No. 2. Although driven right under the shoots of ore in No. 3, none of these were struck, but at a distance of 550 feet bands of copper pyrites commenced to appear, and at 660 feet a copper lode about 40 feet in thickness was struck and driven through. This will average 4.4 per cent. of copper. A great deal of work has been done on this mine, partly prospecting, partly developing and partly removing the gossan ore for the local and Dapto Smelting Works. The results of sampling and assaying at the mine are plotted graphically as the work proceeds, so that the values for every foot driven along or across the lode may be seen at a glance. I am indebted to Mr. Sawyer, the assistant manager, for showing me over the mine and supplying information.

The gold values in this mine and also Mount Reid are high, and were the ores not of a refractory nature they might be treated with profit for gold alone. Considerable denudation must have gone on, for gold has been shed from these mines right down to the stream nearly 2000 feet below, and the whole has been so rich as to be stripped down to bed rock up to the outcrops of the mines themselves. In both mines also the amount of gossan below the surface was comparatively small, the hard dense sulphide rock appearing in places at the very outcrop. The question of treatment of these complex sulphides is a highly important one. Trials by concentration have been made at the Zeehan works, but only with indifferent results. From the dense nature of the intermixed galena and blende it is clear that if concentration is to be adopted the whole of the ore must be finely ground, otherwise only a mixture of galena and blende will pass out with the heads. This fine grinding would mean sliming a large portion of the ore and great losses could only be expected. In obtaining a 60 per cent. lead concentrate I should not anticipate more than a recovery of 40 per cent. of the lead, and probably not more of the silver and gold contents. Whether this would yield a profit depends on local conditions. At Broken Hill, where very large losses occur, a profit is made.

The discovery of a simpler method for the production of zinc from such ores would be very important. Such a process must be continuous, not intermittent, as most of the present ones are, and capable of dealing with large quantities; but assuming such a process to be discovered the price of the metal would fall and there are many places in the world with large deposits of blende, which could be utilised without drawing on Zeehan supplies. The outlook from this point of view is not encouraging. What is wanted at present is a method for recovering a marketable product of lead, silver and gold, leaving the zinc residues to be dealt with at some future time.

Both mines are nearly 2000 feet above the terminus of the North-East Dundas Railway, and the horizontal distance is less than a mile. The steepness of the pack track up to the mine may be imagined. In order to send ore down from the mine and haul timber and stores up a self-acting tramway was constructed. This is said to be the longest and steepest in the world. The vertical height from bottom to top of the line is 1642 feet, and the slope length is 80.5 chains. The track follows the surface of ground, being graded to a slight extent. The gradients vary from 1 in

$1\frac{1}{2}$ to 1 in $7\frac{1}{2}$. Two feet gauge is used with rails 20lb. per yard. Special precautions have to be taken to prevent the slip of the track down hill. At steep pinches the sleepers are bolted down to the solid rock, and the rails made fast to the sleepers. An endless steel rope passing round horizontal pulleys above and below has attached to it at 350 feet intervals the trucks for ore These trucks are either iron-skip trucks which carry about half a



Tram Track, Hercules Mine, Mount Reid, Tasmania.

ton of ore, or wooden trucks for timber and supplies. Owing to the rises and falls in the grade the trucks are attached to the rope by means of a chain. The grip attaching the chain to the rope is shown in Fig. 4. To take up the slack of the rope and to compensate for variation of length for variation of temperature the total length of the rope being two and a quarter miles, the rope at the lower end of the track is made to pass round a 7 foot grooved wheel lying on a carriage, which moves up or down an inclined plane. Weights are attached to the carriage to keep the rope taut. At the upper end the working is controlled by a powerful brake, which consists of wooden brake blocks on a steel strap, capable of being tightened around the pulleys by means of a compound lever worked by a screw. It is hardly necessary to say that this end of the tramway is cemented down to the solid rock. The brake block is shown in Fig. 3, Plate 3. To lessen friction rollers are used throughout, 8 inch wooden where the wear is not great, and 9 inch iron ones at the bends. The rope itself is 3½ inches in circumference, is made of steel, and its tested breaking strain is 42½ tons. Its total weight is 11 tons. The trouble of getting this to the top of the hill was enormous, even though in three pieces, which afterwards had to be spliced. It was hauled up, little by little, by block and tackle until the top was reached. The number of trucks which pass up and down each way per shift of eight hours is from 200 to 250; or about 100 tons of ore can be delivered per shift. The cost in labor per ton of ore on this tram is 6d., the capital cost of the tramway and plant being £8750. I am indebted to Mr. Sydney Thow, the general manager, for particulars and drawings of this interesting tram line.

The Blythe River Iron Deposits.

The gold produced per annum throughout the world is about 13 million ounces; the amount of iron produced per annum is about 40 million tons. In other words, the value of the crude iron produced per annum is more than double the value of the gold raised. Australasia produces nearly one-third of the total gold raised, but, although possessed of the finest ironstone deposits in the world, does not produce iron from the raw ore. Several attempts have been made in New South Wales to start iron smelting, and, curious to relate, as far back as 1867, 90 tons of pig iron produced in New South Wales were sold in San Francisco, and realised £6 per ton, and afterwards some hundreds of tons were shipped and sold for satisfactory prices. In New Zealand, in Tasmania and in Victoria attempts were made to produce iron on a large scale years ago, but the enthusiasm of the promoters of such enterprises was in advance of the technical skill at their command. All these efforts failed. Since that time Australia has become separated from the mother country by a new generation, and the production of iron is not viewed in the same way as it was years ago. Nowadays, it is comparatively easy to say, from the analyses of ore and fuel what class of iron can be produced from a given ore, at what cost it can be mined and smelted, and what the profit or loss on the operation would be. The low freight of pig iron from the old country, it being practically carried as ballast, the cheap rate of production, and the fact that the various States admitted iron or steel required for the construction of railway lines free of duty, have all precluded the idea of establishing ironworks to produce iron in Australia. So long as we are content to go on in this way our enormous iron deposits will stand out as monuments to our lethargy and indifference. On the other hand, if ironworks are to be started, it means that they must be such as to supply all the States with pig iron and ordinary steel; it means that a whole army of workmen must be trained to work entirely new to them, and it means the expenditure of at least a million sovereigns before a ton of pig iron could be turned out. More than this, it means that the States must be the largest customers to such works, so that before such a sum of money is found for Australia, investors want some guarantee that State custom will be given to the works, and that some bonus should be given for establishing such an industry and taking the risk. It is natural that some of the Representatives of the Commonwealth, seeing that the States would be large customers, desire that such works should be established under Government control. State management would mean failure for an enterprise competing with the keenest rivals in the world. Operations in this case could not be reduced to the red tape routine; while the officials would not dare to depart from old-time methods, and would thus be left hopelessly in the rear, leaving the States to make up an ever-increasing deficit. It is to be hoped for the sake of taxpayers that such a scheme will never be advocated seriously.

It is certain that in course of time Australia will be producing iron, and there is no reason why a start should not be made at once. The money can be found for starting the industry, permanent employment for thousands of people would be guaranteed, and our deposits of coal and iron opened up and developed. The high price of labor in our States as against that of European and Asiatic countries will be more than compensated, as in America, by the extensive use of labor-saving machinery and natural resources. For instance, in America the cost of production per ton of iron is £1 12s. 6d., while in England it is £2 12s.

It was long held to be a sine qua non that iron ores and coal should be in proximity for successful iron smelting. Modern methods of transport have altered all this, and the result is that almost all the iron ores are carried for great distances. The ores from Lake Superior are carried 800 miles to Pittsburg, while rich Spanish and Swedish ores are conveyed over 1000 miles to Great Britain. A distributing centre for the finished product is of importance, as well as the sites of the ores and fuel. The amount of fuel and fluxes required will vary with the weight of the ore. Ores running 60 per cent. and over will take about half their weight in coke and about three-eighths their weight in limestone, so that the weight of coke and fluxes is about equal to the weight of the iron ore. With low grade ores much more would be taken. If coal is used in place of coke from 30 to 40 per cent. extra would have to be added on the weight of fuel given. These considerations show that it is desirable to establish central works near a suitable harbor on some field which produces a first-class coal.

Nearly ten years ago samples of an exceedingly pure hematite were sent over from Burnie, Tasmania, with a notification that they came from the Blythe River. These consisted of ferric oxide with traces of phosphoric acid, silver, copper and gold. Since that time I have had many samples from the same place, but all analysis showed that the ores were exceedingly pure. A syndicate was formed to develop this property, and about £35,000 has been spent. A gentleman of very high repute in England, Mr. Darby, was engaged at a fee of £3000 and travelling expenses amounting to nearly another £1000, to report on the proposed scheme. Mr. Darby's report was very exhaustive, and included: (1) The deposit of hematite iron on the Blythe river, (2) the site of the proposed works, (3) the sources of fuel, (4) the manufacture of steel, (5) estimates of the outlay. Although much of the information given is the property of the syndicate, it is only necessary to indicate that the report was wholly favorable. The site for the works was fixed at Ryde, on the Paramatta River (N.S.W.) It was proposed to have a fleet of ships from Tasmania to Sydney carrying ore to the extent of nearly 1000 tons per day, while the works would turn out 150,000 tons of steel per annum. The field which was to supply this for 20 years at least was in Tasmania, and on the Blythe River. The mine is situated on the north-west coast of Tasmania, about six miles from the mouth of the Blythe River and about 150 feet above the level of the sea. The picturesque town of Burnie is about ten miles away. Through the courtesy of Messrs. Wm. Jones and Son I was enabled to visit the mine and obtain particulars of the work done. The country in the neighborhood is hilly, the hills being covered with a decomposed basalt

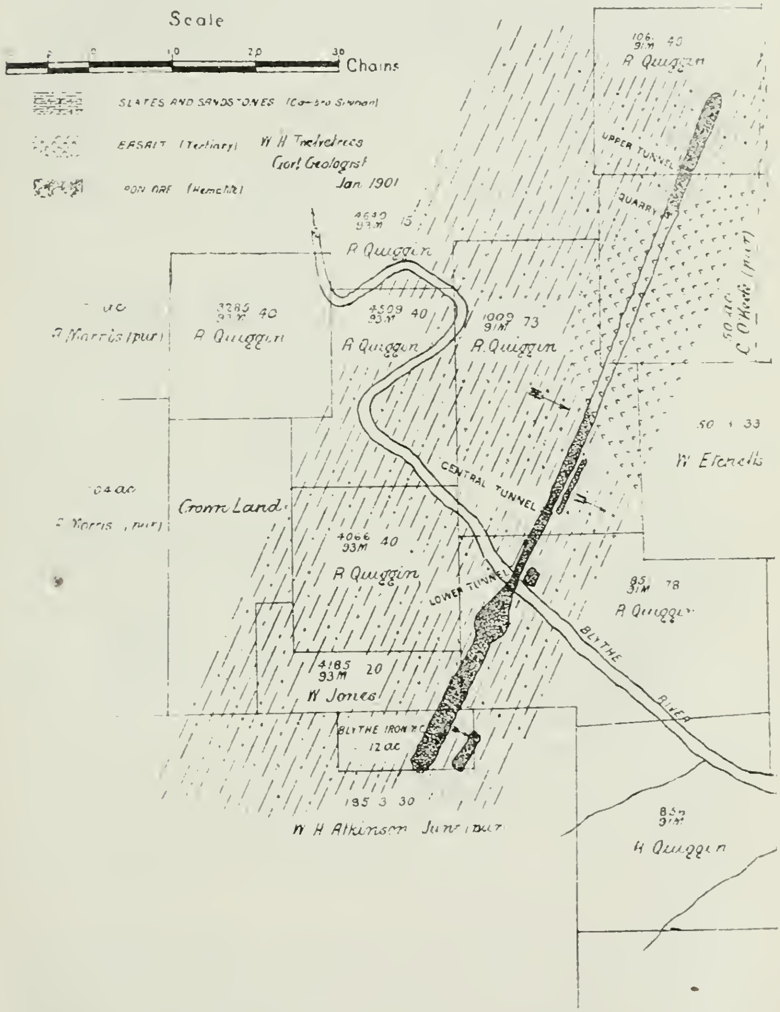
Blythe River.

giving a rich chocolate soil. Near the river the basalt disappears, and the country becomes more rugged. A gorge, some 500 feet deep, has been cut by the Blythe River, and on both sides massive monoliths of hematite rise from the river bed to the summit of the hill, showing where the deposit has been cut through by the eroding stream. In the bed below great blocks of pure hematite may be found a long way below the source of supply, and it would appear as if thousands of tons could be picked up in the bed of the stream. The outcrop runs N. 27deg. E., and may be traced for a mile, while it continues both north and south for a much greater distance. For about 20 chains on the north side of the river a patch of basalt overlies the ironstone, but independent of this there is a vast amount showing. The deposit is nearly vertical and its outcrop rises some 600 feet above the bed of the river. It appears to be interbedded between the slates and sandstones. The width at the surface has been approximately determined by means of trenches and tunnels. On the south side, about 100 feet above the river it measures, according to Mr. Twelvetrees, 147 feet, and consists of hard hematite; higher up the hill it measures, as exposed by a trench, 260 feet, but contains more silica. Near the top of the hill is a huge isolated block of solid ore known as the purple crag. The width is over 100 feet. At the river level the ore is exposed for over 30 feet, and as the northern slope is ascended the width increases to over 107 feet. Mr. Darby had several tunnels driven in order to define the outcrop. The lower tunnel is at the base of the northern bank of the stream. This was driven along the deposit for 225 feet. Crosscuts into the ore were driven at frequent intervals. The first, at 30 feet, was put in for 12 feet, and last 6 feet being in pure hematite; the second, at 45 feet, was driven 6 feet into ore; the third, at 66 feet, was driven for 10 feet, part of the material being jasper; the fourth, at 77 feet, for 17 feet, all in solid hematite; the fifth, at 142 feet, cut into good ore; the sixth, at 199 feet, into ore; the seventh at 225 feet, into good ore for 13 feet. Analysis of specimens selected by Mr. Twelvetrees and determined by Mr. Ward, the Government Analyst:—

Crosscut at	Iron. %	Silica. %	Phosphorous. %	Copper. %	Sulphur. %
66 feet	46.0	34.2	Nil.	Nil.	Nil.
77 ,,	65.0	7.0	,,	,,	,,
142 ,,	67.2	3.8	,,	,,	,,
167 ,,	68.1	2.4	,,	,,	,,
199 ,,	68.5	2.0	,,	,,	,,
225 ,,	68.7	1.6	.04	,,	traces

The second tunnel, or upper tunnel, put in by Mr. Darby is driven through the deposit at the northern end. It entered ore at 79 feet from the surface, and was driven through ore for 84 feet. This gave 60 per cent. iron and 14 per cent. silica. A large amount of work has been done on the property, which has made assurance doubly sure with regard to quality and quantity. The members of the syndicate deserve great credit for the painstaking way they have proved the property. They have shown beyond doubt that the ironstone is not a mere superficial capping over some pyritic body, and that at least down to the river level it has maintained its dimensions and quality.

The specific gravity of the average Blythe furnace ore is 4.807, so that a cubic foot weighs 299lb., and a cubic yard 3.6 tons. Taking the length of the deposit as 6000 feet and the depth from the surface only to the level of the river—assuming an average height of 450 feet, with 100 feet, and length 6000 feet—this would give 450 x 100 x 6000 cubic feet, or 270,000,000 cubic feet, or 10,000,000 cubic yards or 36,000,000 tons. Mr. Darby estimates the available supply, after halving his estimates, at 24,500,000



Plan of Blythe River Leases, Tasmania.

tons, or if 300,000 tons were smelted per annum this one mine would only be worked down to the river level in 82 years. The iron contained in this may be estimated at 14,000,000 tons, worth about 50 million pounds. The weight of iron used by Australia per annum as steel rails, sheets, girders, and wire is about 250,000 tons, so that this one mine could supply the Commonwealth for half a century. The analysis of the ores as obtained by Mr. Darby I have been enabled to obtain through the courtesy of Mr. Wm. Jamieson, chairman of the Blythe River Iron Mines, Ltd. Mr. Darby divides his samples into four parts. No. 1, taken from outcrop north of basalt covering: Ferric oxide, 93.64 per cent.; iron, 65.54 per cent.; silica, 5.19 per cent. No. 2, taken over the slope to river on N.E. side; Ferric oxide, 85.93 per cent.; iron, 60.15 per cent.; silica, 12.41 per cent. No. 3, taken over the S.W. slope: Ferric oxide, 85.38 per cent.; iron, 59.76 per cent.; silica, 11.09 per cent. No. 4, waterworn boulders in river and north of deposit: Ferric oxide, 97.08 per cent.; iron, 67.95 per cent.; silica, 1.13 per cent.

An average sample of the whole deposit gives the following complete analysis:—

	Per cent.	
Ferric oxide (Fe_2O_3)	86.934	63.259 of iron
Ferrous oxide (FeO)	3.074	
Silica (SiO_2)	7.312	
Alumina (Al_2O_3)	1.756	
Lime (CaO)	0.068	
Magnesia (MgO)	0.071	
Sulphur trioxide (SO_3)	0.060	0.024 sulphur
Phos. pentoxide (P_2O_5)	0.083	0.036 phosphorus
Titanic acid (TiO_2)	0.03	
Copper	trace	
Arsenic	trace	
Manganese	trace	
Chromium	none	
Combined water	0.324	
Moisture	0.160	
	99.892	

Mr. Darby also took a sample of the ores and reduced it with coke, using lime for a flux, and obtained a cast-iron button. This on analysis gave:—

	Per cent.
Carbon	1.800
Silicon	0.032
Phosphorous	0.062
Sulphur	0.092
Manganese, chromium and titanium	Nil.

Since the coke ash used would give 0.5 per cent. of phosphorous, and part of the sulphur was introduced from the burning of the fuel, it is evident that by smelting with a good limestone that an excellent hematite pig suitable for manufacturing a high class steel could be produced. The yield obtained by fire assay was 66.5 per cent. of the ore used, and 12.5 per cent. of limestone was added on the weight of the ore.

A railway could easily be constructed from Burnie to the mine. Ocean-going steamers could be loaded direct from the trucks, so that the freight from mine to vessel would not exceed 6d. per ton, while the breaking of ore and loading of trucks would not be more than 3s. per ton, or a total cost 3s. 6d. per ton. There are not many places in Australia as favorably situated as this; but there are many fine deposits in Victoria, Queensland and South Australia, whose ores no doubt would be sought for the manufacture of special classes of pig iron or steel. The deposits at Nowa-Nowa, at the head of Lake Tyers, Gippsland, contain manganese; those at Beaconsfield (T.) chromium; while fine deposits of magnetite occur on the Fitzroy River, and near the coast in Central Queensland.

QUEENSLAND.

The area of this great State is 427,838,080 acres, while its population is only a little more than half a million. A large portion of it consists of extremely fertile soil, capable of producing any form of tropical vegetation; ordinary temperate climate crops are grown over wide areas in the southern portion; while cattle and sheep thrive from one end to the other. The climate of the southern tableland is all that could be desired. The winter in the northern areas is delightful, but the summer, as in all tropical countries, is exceedingly trying to those not inured to it. It is, however, not worse off in this respect than Western Australia. Many generations will elapse before Northern Queensland will be developed by white labor alone. The mountain ranges running from the north to the south, as well as a large tract of country inland from the Gulf of Carpentaria, is rich in minerals of economic value. Silver, copper, lead, tin, zinc, bismuth, antimony, wolfram, manganese, chromite, molybdenite, as well as opals, sapphires, and other gemstones, are present in commercial quantities. Coal in immense quantities is widely distributed; while the production of gold up to date has only been surpassed by Victoria and New Zealand. The auriferous deposits occur in rocks of various geological ages. The veinstone, as in most other places, is usually quartz, more or less impregnated with pyrites, galena, blende, and similar minerals. In the Charters Towers, Etheridge, and Croydon districts, the surrounding rock is granitic. The Mt. Morgan rock is said to be of Permo Carboniferous age, as is that on the Gympie field. In the former case, the rock worked is said to be a siliceous sinter, in the latter case the quartz veins are auriferous when they cross the black slates, which they do nearly at right angles.

Charters Towers.

In this series of articles, I desire in the first place to acknowledge the courtesy and help received from the permanent heads of the Mines Department, Queensland, also from the Wardens of the various goldfields, and finally, almost without exception, from the mining and metallurgical men throughout that great State. It may be fairly assumed that there is little need to describe in detail a centre which but lately prided itself on being the premier goldfield of Australasia. The place is so well-known, and the reefs so well mapped out, that an extended description would be out of place. It is not, however, generally known that the reefs are so flat over the field that one could walk up and down in them in almost all places, the angle of dip from the horizontal being from 10deg. to 40deg. The underlie as measured from the vertical would be from 50deg. to 80deg. This, of course, has necessitated an altogether different method of working from that practised in Victoria, the shafts being invariably sunk on the underlie, or a vertical shaft sunk to intersect the reef, which is then continued on the underlie until the reef passes out of the property. The ordinary method of stoping and running the stone down passes cannot be so successfully carried out as in Victoria; partly owing to more handling, hard country, and on the average smaller lodes, the cost of mining is much higher.

Ventilation, as might be expected, is a difficult problem. It certainly must have been shamefully neglected, when small shafts, blocked up most of their time with mining appliances, were deemed sufficient. Fortunately, most of the workings are connected with those of the adjoining mines, thus obviating the necessity for special methods.

Water does not give trouble in most of the mines. Many of those now working do not contain a drop, and one sinks ankle deep along the levels in a fine, impalpable dust. The country is of a granitic material, composed mainly of quartz, orthoclase, felspar, and hornblende: thus it would be ordinarily termed syenite. Some of it contains triclinic felspars instead of orthoclase, and resembles closely the quartz diorites of Swift Creek, Omeo (V.). In many places it has undergone a great strain, and is loose and jointed, and swells when exposed, requiring very careful timbering; in other places it is so hard and dense that it has to be cut away with explosives.

The veinstone consists of quartz, carrying about 7 per cent. of metallic minerals, consisting of iron pyrites, copper pyrites, galena, blende, and very little arsenical pyrites. The reefs vary in thickness from a thread to bulges of quartz 20 or 30 feet from wall to wall, yet, on the average, a reef 3 feet wide would be considered a good one. Good timber being scarce, and long timber expensive, the pig-sty method has been adopted, and used with great success. A sty is built up as a pillar, and filled with mullock until it reaches the hanging-wall above. Many of the props and stulls have been crushed and splintered by the swelling of the ground when opened up. The stone, in many instances, is still carted in drays holding about two tons each; but in advance of Victorian methods actual weights are given, and not the indefinable Bendigo load, nor the number of cubic feet, which too often varies with the yield.

Rock-breakers are almost universally used, and much ingenuity is shown in their arrangement. The original sites chosen for batteries being rather flat, a deep rectangular trench was made parallel to the battery. This was bricked and cemented, and the grizzly and rock-breakers placed in it; the drays or trucks tip their material on to the grizzly, the coarse going to the breaker, which, of course, is firmly fixed, and transmits no shocks and vibrations to the rest of the machinery. The fines and crushed material are then automatically picked up by buckets on an elevator chain belt, and delivered either into trucks, or emptied into a hopper. Thence it passes into an automatic feeder, which delivers it into the battery. Mercury is usually fed into the box. The pulp passes through punched screens, passes over plates sometimes absurdly short, then into an amalgamator. Leaving this, it goes to a Brown and Stansfield concentrator, the sands and slimes passing away to be cyanided, while the concentrates are ground in Wheeler's or other pans running continuously. The ground material is further treated by fine grinding in Berdan basins, into which lime is fed. Two drags are used, and no ball. The pyritic slimes or sludges are then run into pits and settled as far as possible. These are then sold by tender to the cyanide firms.

At one of the show plants of the place, the Brilliant Blocks, they have two stone-breakers down in the concrete trench, elevators for the broken stone, the hoppers, and the automatic feeders. The battery of 40 heads was running at the rate of 75 blows per minute. The stamps weighed about 8cwt., while punched screens having 225 holes per square inch were used. The dies were level with the bottom of the screen. The pulp passed over a short copper plate, then into a patent amalgamator, thence on to the usual Brown and Stansfield, the sand and slimes going to the cyanide works, and the concentrates to Wheeler's pans, thence to a settling pan. The concentrates were ground as usual in the Berdans, revolving 16 times per minute, then sold by tender.

The various appliances consist of 2 grizzlys, 2 stone-breakers, 40 stamps in 8 boxes, 8 automatic feeders, 8 Brown and Stansfield concentrators, 12 Wheeler pans, 8 settlers for these, 30 Berdan basins, and 3 settlers for these.

Steam was generated in five large boilers at a pressure of 100lb. per square inch. A compound engine with 14 and 24-inch cylinders, and 2 feet 6 inches stroke, supplied the motive power, which was transmitted by hemp ropes for the whole plant. A surface condenser was effectively used, and the heated water was used for the battery and for the amalgamating pans. The temperature of the water was over 100deg. F.

In addition to these the company has a large cyanide plant, which was engaged in treating the accumulation of tailings from former years. It consists of three cement lined vats, fitted with Butters' automatic distributor, 14 vats, each holding 60 tons of sand, 4 sumps for receiving the solutions from the sand vats, and two as reservoirs and settlers for the zinc boxes, of which there are two of the usual type. Only a working strength solution of cyanide is used. The zinc containing the precipitated bullion is screened through a sieve containing eight holes per running inch, is placed on trays in a reverberatory furnace, and roasted to ash, after which it is smelted to base bullion by the ordinary fluxes.

At another typical mill—Craven's—there is a 10-head battery with 1000lb. stamps, and five Huntingdon mills, the latter being preferred to the stamps. The uneven wear of the Huntingdon mill has been successfully overcome by crossing the belt and reversing their running from time to time. The ore is broken by a stone-breaker, and fed into the mills through an automatic Challenge feeder. The pulp passes over three shallow wells—the first having a baffle board—and then over four copper plates of about 8 feet in total length. After that comes the usual string of appliances mentioned before. Eight Watson and Denny pans and twenty-six Berdan's do all the grinding. The tailings and slimes from the grinding appliances are elevated by a large double tailings wheel and removed for cyaniding.

During my visit to Charters Towers in 1901, the metallurgical methods taken as a whole could not be viewed with anything but disappointment. In some respects it compared favorably with other Australian goldfields. The batteries were as a rule well constructed, and as heavy as usually made. The output per stamp was not so high as in other places, but even with the heavy stamps running at 100 blows per minute and over, the same number of tons will never be crushed as with ores, say, from Johannesburg, the Boulder or Ballarat. The quartz at Charters Towers contains up to 7 per cent. of heavy metallic minerals, and it is the discharge of these through the screens that will always limit the output. The old system was to roughly concentrate, grind the concentrates in amalgamating pans, the residues from the pans being sold to customs works. The sands and slimes were got rid of where possible. This system was persisted in until experts in cyaniding proved that the sands could be treated at a profit, and the slime residues from the pans at a greater one.

Through the courtesy of Mr. S. Horsley, Inspector of Mines, I have been informed that progressive work has taken place in the installation of new boilers, working up to 120lb. per square inch, and of the transformation of the high pressure, non-condensing engines into compound condensing engines. The want of capital rather than conservativeness accounts for the retention of inferior plant. Mr. W. A. MacLeod also states that the mill work is gradually receiving more attention, and that concentration and smelting are replacing, wholly or partly, the old grinding processes.

Unfortunately, the fields of Northern Queensland adopted the Charters Towers methods of grinding and amalgamating in Berdans, and there is no doubt this has led to the abandonment of mines which would have proved profitable with more rational methods of treatment. The ore at Charters Towers is not an exceptionally difficult one to deal with, and there is no reason why the whole of the gold, less a very small percentage, should not be got out on the spot. It is further absolutely necessary for outlying fields to adopt a system of treatment which will not be costly, but which will give them over 90 per cent. of the gold.

The Defiance Plant—Refining of Cyanide Bullion by Parting from a Gold-Silver-Zinc Alloy.—The Defiance plant at Charters

Towers presents no new feature so far as crushing and amalgamation go. The 30-head battery has its plates, its Brown and Stansfield concentrators, and its long double row of 32 Berdans spinning round. A couple of buddles do portion of the work that is done by the Brown and Stansfield at other places. As much of the ground slimes as can be settled in pits are saved, and these are sold by tender.

This company has an excellent manager in Mr. D. McIntyre, who came out many years ago to work the MacArthur-Forrest process for the Australian Gold Recovery Company. In those days it was generally stated that the process would be useless in Australia; now the pendulum has swung round to almost the opposite extreme. The cyanide process has been made a success by trained scientific men, and though the process is successfully worked in many instances by men who know very little of the reactions involved, yet if the conditions are varied the latter will fail when the former succeed. This plant contains 12 sand vats 25 feet x 8 feet, six solution vats, three reservoirs for the solutions from the sand vats. The reservoirs were at a higher level than any of the vats, the gold solutions being pumped into them after leaving the sand vats. From these the solutions passed through the zinc boxes, of which there are three, of the ordinary type, 17 x 2 feet, thence to the solution sumps. Three vats are emptied each day, and two on Saturdays, or 17 vats per week are treated. The vats have the side-discharge doors, and suction is used to assist filtration.

Only a working solution, 0.5 per cent., is used, and 80 gallons of liquor to every ton of ore; 1lb. of lime per ton is mixed with the sand, and the consumption of cyanide is about 1lb. per ton, while 1lb. of zinc is used for each ounce of pure gold recovered. From 6dwt. sand, or over, 82 per cent. is the usual extraction. The sand, which for months of the year is almost dry, is run up an inclined tram, mixed with the requisite quantity of lime, is dropped into a hopper having a rapid shake, then on to an automatic screen, the coarser particles going through a disintegrator, which knocks them to dust. The whole is then elevated by a travelling belt with buckets and conveyed to the sand vats. The sand is mixed with the slimes from time to time. There, as elsewhere on this field, a great inroad has been made on the tailings heaps, and the amount of gold recovered from them is telling testimony of the inefficient work done by the batteries and gold-saving appliances on the Towers.

Mr. McIntyre adopts a plan for refining his bullion which might well be copied on other goldfields. By sending away unrefined cyanide bullion the aggregate bank and mint charges at so much per ounce leaves only a profit of 6d. per ounce on the silver, whereas by refining on the spot, and sending only refined gold away less than half the expense is incurred, while the refined silver bullion shipped to the old country fetches its full value, with only a deduction of ⅝d.

The zinc precipitate is placed in a retort somewhat like the D retorts of the gas works, and the mercury distilled out. It is then roasted and smelted in the usual way, care being taken to have the resulting base bullion carrying about £2 worth of gold per ounce; in other words, a little less than half. The alloy is then granulated by pouring it into cold water, and the grains and flakes attacked in cast iron enamelled dishes with nitric acid. The zinc, lead, and silver rapidly dissolve, and after the brown fumes of nitric peroxide cease to come off the solution is decanted into a tub and well washed. The residual gold is dried and smelted into bars, which assay 99 per cent., and fetch as high as £4 4s. 4d. per ounce. The silver is precipitated in the tub with salt as silver chloride. This is stirred round, and allowed to subside for some hours. The process is repeated a couple of times with clean water, after which zinc is introduced and the silver chloride is rapidly decomposed—metallic silver and zinc chloride forming—when the change is complete, which may be readily tested by taking some of the precipitate, washing it well with water, and then treating with pure dilute nitric acid; if the silver is all in the metallic state it will all dissovle; if undecomposed chloride is present it will not be attacked. The precipitated silver is dried and smelted into bars, which are nearly pure.

This is in reality a revival of the old method of parting by what used to be termed inquartation, in which 3 parts by weight of silver were taken to 1 of gold; later, in assaying process or parting, $2\frac{1}{2}$ of silver are deemed necessary for 1 part of gold, but lesser quantities of silver will suffice, the minimum being about 2 parts to 1 of gold. In an ordinary case, the metals present would be gold, silver, lead, and, perhaps, a little copper, with the metallic zinc; in other cases, antimony, arsenic, and larger amounts of copper would be present. The silver, lead, copper, and zinc and arsenic, if in the right proportions to the gold, would dissolve in the acid, while antimony would be transformed into an insoluble compound. Mr. McIntyre found that about $1\frac{1}{4}$ parts of the base metals were sufficient for 1 part of gold. Probably the lower atomic weight of zinc and copper accounts for the lesser quantity necessary. If much lead is present, the action with dilute nitric acid should be continued until brown fumes cease to come off; stronger acid can then be put on. If chlorine is present in the acid, a little silver nitrate can be added to it, and the chloride of silver allowed to settle. If chlorine is present in the water, it may be removed in the same way. In either case, a little chlorine will do no harm. The gold may be washed with water, subsequently with ammonia water to remove any silver chloride which may have been present through using water or nitric acid with chlorine present. The gold may then be smelted in a clay pot, with a small proportion of nitre; borax may be added on top. The antimony will be wholly removed. The silver chloride, which may contain lead chloride also, from the solutions, should be dried; it may be then melted in an old mercury bottle, with the head cut off, at a very low temperature, poured out on an iron slab or into a mould. The fused chloride can then be reduced by placing it in a vessel with a little dilute sulphuric or hydrochloric acid, and adding some wrought iron. When all is reduced the silver is washed with hot water to remove the iron salts, and then smelted in a clay pot, with a little nitre and borax.

The solutions, after the precipitation of the silver chloride, should all be run into a tank or sump, containing scrap iron. Any gold, any silver, and any copper which may have escaped will be precipitated. This latter percaution may not be deemed necessary by some, but I find that in all cases when cleaning this sump out at the end of a year, there are appreciable quantities of gold, silver, and copper. I have used this modification of Mr. McIntyre's method now for some time, and have always got excellent results and almost pure gold and silver.

The Towers Chlorination Works.—One of the most striking landmarks at Charters Towers is a tall chimney stack standing on the edge of a semi-circular ridge about a mile from the town. On enquiry amongst the cyanide men one learns that this is a monument erected to the failure of chlorination processes in North Queensland.

In the days of prodigal returns, when rich pyrites were ground to slimes, and when there was no gold for companies but that obtained by primitive processes, the Towers Chlorination Works had the happiest hunting grounds in Australia. With the advance of cyanide treatment, the low cost of handling and treating sands and slimes, and the utter simplicity of working methods adopted, the chlorination works, once so prosperous, had to shut down, and the works are now converted into a cyanide plant.

Since the chlorination plant erected possesses many novel, original, and instructive features, and since the failure of the chlorination works as such may not be altogether due to failure in working the process, a description of the works will be given. More has been learnt very often by failures than by successes, but unfortunately the failures and their causes are always hushed up. Mr. D. A. Brown installed the original works, and in the hey-day of the company's prosperity, was sent to South Africa and England to gather what information he could from both those countries about modern methods of treatment. As might be expected, Mr. Brown saw the processes applied in a far more elementary way than adopted in Australia, but at the same time he was much impressed with chemical—and especially electrical—work done in England. On his return, he built his patent "Hillside" furnace, erected works, and in all spent about £60,000 on the new plant.

The furnace erected is wholly due to Mr. Brown, and, since its principle has always been the dream of metallurgists, it may be of interest to give its details. As is well known in roasting pyrites, if heat is gradually applied the first atom of sulphur volatilises, and if the temperature is high enough it will burn at a slight distance above a layer of finely divided sulphides. The action at this stage is a reducing one, since any oxides which may be present will be acted upon by the sulphur vapor. If the material be stirred portion will be brought in contact with the oxygen of the air, and an oxidising action will be set up; if too much hot air is admitted then the temperature may rise so high as to fuse into large lumps the finely divided sulphides, especially if oxidisation is not far advanced; or, again, since a single grain of pyrites contains thousands of atoms, each of which must be individually acted upon by the oxygen atoms of the air, if the temperature rises and melts portion of this grain then gold is locked up in it until it is oxidised again, or until it has been acted upon by solvents. A trained man knows that the first changes which take place within his furnace practically govern the condition of the ore when it is discharged. Almost the whole of the sulphur may be removed in four hours from an ore properly handled at first, but to get rid of the last 1 per cent. gives more trouble than the ninety and nine, and often takes longer. In this case ore is fed in at the top of a hill, and allowed to flow through a reverberatory furnace to the bottom.

The ore is fed in from a hopper by an ingenious contrivance, which regulates it to a nicety; it is then swept along on a hearth 10 feet wide for 33 feet by automatic rabbles. When it starts in its sliding downward journey—the angle of slope being 38 degrees—it is pulled up at another short horizontal floor, at the lower end of which there is a subsidiary fireplace; it is worked along this floor by the automatic rabbles as before, when it starts on its final descent, only to be pulled up at the last horizontal hearth, along which it is worked by hand rabbles to the discharging door. The main fire-box is at the end of this hearth. The total time occupied in the descent is about three hours.

The furnace is surmounted by the monumental stack 167 feet high and 17 feet 6 inches in diameter at the base on the outside. The stack itself has absorbed 120,000 bricks.

Between the stack and the feed hopper are elaborate dust chambers—four double and five single ones. These chambers have a V shaped cross-section, and are 28 feet wide on top, have their sides sloping at an angle of 45deg., and terminating in a trough at the bottom. These chambers are built and buttressed in brick, and must contain as many as the stack. The trough in the bottom is fitted with automatic scrapers, which continually work the dust accumulating in the chambers into the furnace again. The great volume of these V shaped chambers serves to diminish the speed of the flowing gas, and thereby allow the dust to settle; breaks in order to further deposition of dust have been placed across the chambers.

In order to economise the heat a coil of pipes has been inserted in the upper portion of the furnace. These are then led below the hearth right down to the bottom. By means of a Root's blower, air is forced into the coil, thence through the hot hearth into the furnace above the fuel in the fireplace.

The fireplace is the full width of the hearth—i.e., 10 feet—and may be opened from both sides. It is 3 feet in width, and the bars are 3 feet below the bridge.

Near the lower end of the furnace is provision for what is known as a chemical spray; in other words, instead of feeding in nitre or salt, or both together, which I learnt was the practice here, a solution is made, and this is forced in in the form of spray. It is claimed that by this means there is a better chance of chemical contact between the various ingredients.

The furnace, instead of having a continuous arch from side to side, has a number of arches which are parallel to the length of the furnace. Girders are run across at intervals from wall to wall; skewbacks are carried on these, and the arch spans across from girder to girder. The flame and heated air must be deflected under every girder, which is about 9 inches above the floor, the centre of the arch being about three feet. The total fall of the furnace was 275 feet, the length of the hearth therefore being considerably more than 300 feet. The total height from the furnace door to the top of the stack was 442 feet! There is little wonder that the automatic rabbles dragged 60 tons per week out of the dust chambers.

The wood consumed by this furnace is said to average 4cwt. per ton of ore roasted, while it was claimed that from 250 to 300 tons of mixed concentrates and slimes were put through per week.

In addition to the furnace described, another Hillside furnace, 145 feet in length by 10 feet in width, had been in use at the Towers Chlorination Works. Mr. Browne was so satisfied with the work done by it that he used the design as the basis of the

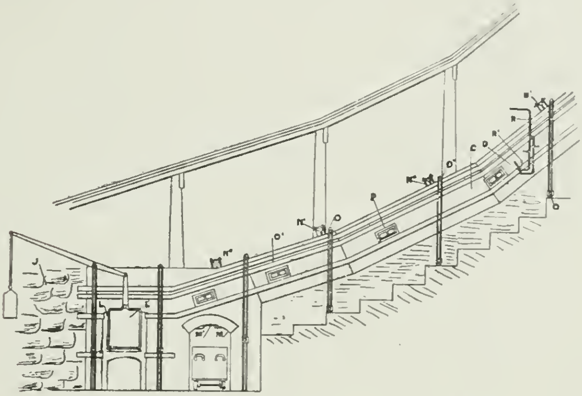


Fig. I.—Elevation of Lower End of Furnace.

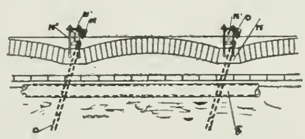


Fig. II.—Longitudinal Section Showing Arrangement of Arches

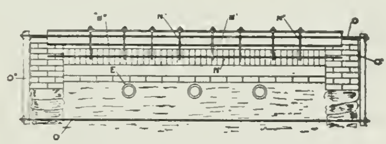


Fig. III.—Cross Section.

Charters Towers Chlorination Furnace.

larger one. The total cost of roasting a pyritic ore containing from 20 to 30 per cent. of S. is stated to be 8s. per ton.

The roasted ore was disintegrated, and all fine lumps of slimy, caked material knocked into powder by a Carr's disintegrator, after which it was conveyed to a cooling floor. The sand was then damped, and automatically conveyed and discharged into wooden vats, of which there were four, each vat being 22 feet in diameter and 4 feet deep. These vats were made of tallow wood, and well protected by asphaltum and pitch. The filter bottom consisted of a false bottom of hardwood, bored with small augur holes. On top of this was coarsely broken glass, then finer, the top finishing off with fine material. The glass was broken in a Carr's machine, and afterwards screened.

After filling the vat with damped sand chlorine gas was passed up through the sand, and when it reached the surface a solution of chlorine water, containing 450 to 500 grains of chlorine to the cubic foot, or about a 0.1 per cent. solution, was run on above. This was allowed to remain on until the gold was dissolved. The liquor, instead of being run into the precipitating vats straight away, was passed through the other sand vats, so as to enrich it as far as possible in gold and chlorine. This solution was then run through a vertical column or tower, into which high pressure steam was blown; the bulk of the chlorine was driven off, and after passing through an exit pipe, was reabsorbed in a scrubber; this solution was then stored in a reservoir, to be used on a fresh lot of ore. The gold was precipitated from the solutions by ferrous sulphate, a small amount of hydro-chloric acid also being added in order to keep basic salts in solution. There were eight precipitating vats, each 9 feet in diameter and 3 feet deep. Instead of collecting the ferrous sulphate precipitate, the solutions containing the finely divided precipitated gold were run through filter beds made up of sawdust and fine charcoal; this was taken out from time to time and burnt in a reverberatory furnace, leaving the ash and gold to be smelted, or sometimes the gold was redissolved and re-precipitated, after which it was collected and smelted. The charcoal and sawdust filters were cement boxes 4 feet square, with partitions arranged on the same principle as the cyanide precipitating boxes. Chlorine was generated in a special still made of iron, lined with porcelain tiles, the spaces between the tiles being well jointed with rubber. Manganese dioxide, found near Rockhampton, sulphuric acid and salt were used for the production of the gas. Since the stills were only heated by steam only the bisulphate of sodium could be formed—$MnO_2 + 3H_2SO_4 + 2NaCl = MnSO_4 + 2HNaSO_4 + 2H_2O + Cl_2$. The cost of sulphuric acid was £10 per ton. The company has installed a very complete electrical transmission plant, and by means of motors the ore is hoisted up an inclined tram to the top of the hill; the dust is dragged back from the chambers, the Roots' blower is worked, the furnace is rabbled, and the ore conveyed from place to place.

The whole plant has now been converted into a cyanide one; four more sand vats of similar dimensions to those in use for the chlorination process have been added, and the red sand which had been chlorinated was undergoing re-treatment by cyanide, as well as a heap of finely-ground concentrates which had been purchased. The only special feature in connection with this was the large amount of lead said to be precipitated on the zinc filaments. Mr. Browne informed me that this amounted not to pounds, but hundredweights, and that he had to refine this specially with chlorine solutions.

Mount Morgan.

We all remember the wonderful stories in the Arabian Nights and kindred works, in which genii and other supernatural figures poured out blessings, wealth, and prosperity on fortunate mortals; but the plain unvarnished tale of Mt. Morgan rivals even those stories, as truth surpasses fiction. In 1864 Donald Gordon took up the two blocks of land containing what proved to be the richest gold mine in the world, but, after twelve years' struggle, abandoned his grazing selection and trekked further north. His brother, Sandy Gordon, one of those sanguine, restless men who are always on the verge of discovering something big, was prospecting for Messrs. Burns and Twigg, the well-known founders and engineers at Rockhampton. His stories as to the great lode on his brother's land were heard as idle tales, and he was always sent prospecting in other directions. His patrons, having special work to do on the Fitzroy River, sent Sandy temporarily to the Morgan Brothers, who employed him. His oft-repeated tale did not fall on deaf ears in this case, for the Morgans, taking Sandy with them, started in July, 1882, on a prospecting tour. They first prospected what Sandy termed a silver lode, and on returning further prospected Gorden's property. Having noticed some ironstone boulders, they broke portions off and afterwards crushed them upon a shovel and panned off, when they found so much gold that it made them even doubt the metal itself. Not a word was said to Sandy about the find. Interests were sold to T. S. Hall, manager of the Q.N. Bank, D'Arcy, and Pattison, who went to see the mine, the Morgans still retaining half. Messrs. Burns and Twigg again had their chance, for they were offered a large share in the mine for a 10-head battery. They again missed it. The gold won by the first battery erected paid for the battery and gave back all the money to those who bought in with the Morgans, and left a handsome profit besides. Sandy Gordon died no better off than he lived. Donald Gordon sold his land to the lucky shareholders for £1 per acre, not knowing what they had discovered, and the block which he rejected has become the chief corner-stone of a great gold-mining State.

The phenomenal returns obtained, and the enormous amount of material available, caused the original holders to water their scrip to the extent of a million. Again the local people were offered those shares at £1 apiece; again they refused. Shares went up steadily until they were £8 in 1877, while in the following year they mounted up to £17 15s., when the inevitable collapse came. One shareholder with 200,000 shares found for some time his scrip increasing in value by £1 per day, so that his capital was increasing at the fabulous rate of a million per week!

Mount Morgan means to most people a mountain charged with gold. Such is not the case. The original top of the mount was between 500 and 600 feet above the level of the Dee River, a stream which runs round the base of the mount, and which supplies water for the works. The crest of the hill was an ironstone cap, most of which has been removed. Below this crest the present open-cut operations are in progress, and almost the whole of the material now being removed is as white as chalk. The cut is dazzlingly bright, with the sunlight reflected from every surface. The kaolinised material and the silica, which form a large portion of the auriferous ore treated, are as unlike the ordinary specimens shown in museums from the mount as the kaolin of Broken Hill is unlike the original ironstone outcrop. Below the kaolin and porous silica the material becomes hard, dense, and compact, and consists largely of quartzite, with occasional bands of quartz, in places fairly heavily charged with pyrites. The adjoining rock is felsite, while the whole hill has been penetrated by dolerite dykes, more or less decomposed.

The auriferous mass, so far as is proved, it lenticular in shape, 800 feet long, and from 500 to 600 feet in width. The gold present is all in an extremely finely divided state, so that specimens are unknown. In the upper levels the gold is all free, but in the lower and pyritic ores there is good reason to believe it exists largely as a telluride. Litte or no free gold may be obtained by panning off, while samples may be very rich as tested by assay. Tellurium has also been proved to be present.

The original cap was 90 feet above the present surface, and the mount is being worked to a depth of 600 feet below this. The various levels are worked as far as possible by tunnels, while the ore is wholly removed and the square-set system of timbering adopted, the sets being higher than those used at Broken Hill, and colonial hardwood used instead of oregon. The open-cut system is also adopted, and by its means much valuable timber has been recovered, to be re-used at the lower levels.

The ventilation of the mine is practically perfect. The mine is cool, the air fresh, and it speaks volumes for the management when, despite the millions of tons of ore removed, there has not been a single fatal accident.

Concerning the origin of this great mass much has been written. Dr. Jack's famous theory is still accepted by most authorities. It has been attacked by many other scientists, including Mr. J. Macdonald Cameron. Looking at the mine one sees the ironstone crust on top, the sintery white mass below, and the unaltered material below this. Judging from other deposits, it seems that while acknowledging the whole mass to be originally due to hydrothermal action, yet it also appears that the upper portions are but alteration products of a more heavily mineralised material. Pyrites have oxidised and are still oxidising in the mine; free sulphuric acid and basic ferric sulphate have formed and are now forming; layers of gypsum indicate a change in the lime compounds. The iron salts have crept upwards, as solutions will through blotting paper, through the porous silica, and have in turn changed to hydrous oxides. Mount Morgan is not alone in this respect, for in many places in Eastern Gippsland the quartzitic masses in felsite have their iron cap, a porous under-crust, though not so light or sintery looking as Mount Morgan, while the quartzites underneath are bluish in color, and just as heavily charged with pyrites as Mount Morgan—unfortunately the gold in payable quantity is missing. One premise nearly all these theories start from, and that is that the gold was in the form of a chloride. Now, since nearly all the deep-seated solutions are alkaline, this will not stand. Ferrous oxides or salts are mentioned invariably as the precipitating agencies in the face of the fact that these are more common at deeper levels than at the surface. Many facts point to alkaline solution of gold, but the evidence against it being in solution as the ordinary chloride is irresistible.

It has always been stated that Mount Morgan gold is the purest in the world. Since all the gold is now recovered chemically in an almost pure state, it was not clear whether it was the gold won or the gold as it existed in the mine which was so pure. Dr. Lebius in 1884 stated that he examined 10,000oz. as received from the mine retorted. This assayed 99.7 to 99.8 per cent. of gold, only a trace of silver being present, copper and iron making up the balance. At the present time appreciable quantities of silver are to be found on assaying the pyritic portions. A sample of the roasted sand when panned off showed a tail of gold, which did not resemble gold of a high degree of purity.

This question is worth following up, since, if the gold in the upper levels is not alloyed with silver, while that in the lower levels is, it follows that there must have been a secondary solution of, and re-precipitation of, the metal. In the upper levels of most mines there is abundance of sulphuric acid, and this, with an oxidising agent such as manganese dioxide and saline waters, gives all the conditions for the solution of gold. No work, so far as I know, has been done to show whether free chlorine exists in any surface water—it certainly could not exist at any depth. Some years ago I was informed by Mr. A. W. Howitt that he knocked off the points of many stalactites of limonite which had formed in an old auriferous mine. These secondary products were assayed by the late Cosmo Newbery, and gave a return—so far as I remember—of about 15dwt. per ton. It is therefore certain that, if this gold were not previously in a finely divided state, so fine as to be carried along with the trickling water charged with iron salts, there must have been a re-solution and a re-deposition. I cannot see that such surface solution would tend to form nuggets or masses of gold, but it would tend to enrich masses of limonite which it would accompany in this action, being different from a mere oxidisation and solution of pyrites, the gold contained being unacted on, and consequently left behind.

Everybody knows, or should know, the difficulty of amalgamating and saving by ordinary processes such fine gold. The Mount Morgan metallurgists learnt their lesson, and in return have given the world an object lesson on the economical and effective treatment of low-grade ore.

It would fill a volume to describe in detail the plant in connection with Mount Morgan. There are numerous departments, each with its own skilled officer in charge, while the whole of these are under the general metallurgical engineer, Captain G. A. Richard.

Looking over the side of the mount, a bird's-eye view is obtained of buildings and sheds, scattered round the slope, covering many acres in the aggregate. The foundry and engineering shops at the base of the hill; the electrical plant works; the sulphuric acid works; the metallurgical furnaces and sheds, all tend to impress one with the magnitude of the operations conducted. The fine buildings for offices; the manager's and directors' houses; the laboratory and officers' quarters, all indicate the wealth of this great mine. The electric trams, with overhead wire, conveying the material all over the works; the ceaseless train of trucks carrying material from the mine; the giant stone-breakers cracking; the mills pulverising; the furnaces roasting and reddening the sand; the final solutions and the exit of the sand from its receptacles to the worthless tailings dump; the clouds of smoke by day from almost innumerable chimney stacks, and the brilliant glare of the electric lights by night, all of these show never ending activity.

In spite of the name of Mount Morgan, in spite of the millions of pounds worth of gold won and the royal dividends paid, the mine is now a low grade one, and it is only by a masterly knowledge of both principles and practice of scientific work, a clear-sighted policy, and a most admirable system of organisation, that the mine has been looked upon as an investment, and that a large proportion of the ore can be handled at all. To give a single instance of economy. The motive-power some time ago was supplied by compound engines, non-condensing, fed with steam from Cornish boilers—a method common enough in Victoria. The modern installation consists of triple expansion engines, Babcock and Wilcox boilers, condensers of the ammonia type, and fuel economisers. The combustion of fuel in these as compared with the old plant is as 6½lb. is to 13¾lb. for the same amount of work. Considering that 1500 horse-power at least is required for the various appliances, the saving amounts to a very large figure, while of course these savings mean dividends to shareholders. The same spirit of calculated economy prevails throughout, and the result is that ore containing only 7dwt. per ton is mined, crushed, roasted, and treated by chlorination at a profit, a record that has never been equalled for the same process for the world.

Mount Morgan affords a splendid example of the evolution of a process, and of the survival of the fittest. Even as it may not be with the happiest feelings that we look back—the branches lopped off in the process, of abnormal growths in some directions, and of the slow and tedious process of reaching the present stage of development—yet modern mines in many places may have the same stages to go through as Mount Morgan, and the footprints in the sand may enable them to reach their destination in shorter time. While not stating that the methods adopted at the Mount would suit every ore, yet their own problem has been successfully solved, and there is not the slightest doubt that had the ore been different in composition the necessary modification of the process would have been determined. How often is it said: "The ore at Mount Morgan is specially suited for chlorination"?—a most absurd statement, since thousands of difficulties had to be got over before the process was reduced to its present simplicity. A study of some of those difficulties, the failures of ordinary processes, may serve as landmarks to present companies in evolving methods suitable for their own ore, and perhaps save thousands of pounds of unnecessary expenditure.

As may be readily imagined, in the earliest history of the mine all that the original shareholders troubled about was a battery, and what need had they to worry when returns of 25oz. per ton were readily obtained with such primitive appliances. When the returns dwindled down to 2oz. per ton they were troubled, and when they found that even out of surface stone, without one solitary speck of pyrites or refractory material of any description whatever, they were only able to get one-third of the gold present, they were dumbfounded. Parcels of ore were sent to Germany, to England, and to America. The only advice they got was valueless; while, though smelting gave good returns, it was quite out of the question at Mount Morgan; while the expense of shipment and smelting would have swallowed up the profit on high-grade ore and prevented any low-grade ore from being touched. Concentration was advocated, but the most sanguine inventor was discomfited when the value of the tailings from his machine was placed before him. The same old stamper method was continued, a 10-head battery being erected in 1882, and this was followed by a 15-head four years later. And for four years these stampers sent over their plates twice as much gold as was caught. When the great loss was discovered dams were made, and upwards of 20,000 tons of tailings caught. Afterwards the same old category of grinding and amalgamating appliances followed—Wheeler's pans, Watson and Denny pans, Berdan's Chilian mills, Arrastras, and Huntingdon's amalgamator, and a host of smaller fry. How much oftener will it have to be stated that these methods will not give an adequate return when fine gold is to be saved? These methods failed at Mount Morgan, and have failed everywhere else when a close extraction was aimed for.

Chlorination was then tried. Mr. Lymburner and Dr. Benson, two experts from Gympie, entered into a contract to treat the tailings by the Plattner method of chlorination. They were to receive a percentage of the gold over 75 per cent. They failed, very often not getting 50 per cent. In 1885 Newbery and Vautin sought to introduce their process, the special feature of which was that chlorine was to be used in a barrel, and that excess of pressure was to be supplied by forcing air in.

Mear's process had some basis, since if chlorine is forced into a closed vessel the pressure of the gas itself will cause it either to liquefy or to be dissolved in the water present to a very large extent, consequently any gold present will be subject to a fierce attack, and the time of action should be much shortened. With the Newbery-Vautin process the amount of chlorine to be used, as a rule, did not generate pressure, so this was made up for by forcing air in; as if the pressure of another gas or gases was sufficient to liquefy the chlorine present. Surely the time-honored law of partial pressures must have slipped out of the memory of the talented Cosmo Newbery when he lent his name to such a patent. The patentees designed ten barrels, holding one ton each. They were made of cast-iron lined with lead, the latter being protected internally by wooden staves. Newbery was amongst the first to suggest the method which finally led to the success of the process. He no doubt saw the difficulty in leaching; he saw the fineness of the gold and the comparative coarseness of the grains of sand and limonite encasing it. Now, by simply raising the mass to a red heat the hydrated or chemically combined water is driven off, and all the fine clayey material loses its plasticity and becomes converted into a sandy, porous material, which does not clog like clay. In fact, there is the same relation between the clay from a pug mill and brick dust. Now, since the co-efficient of expansion of gold and quartz is different, there is a tendency for the kernel of gold to crack the case of quartz around it by its expansion when heated, and further, when the water chemically combined with various oxides is driven off, the mass becomes more porous, and the gold is brought into actual contact with the solvent solutions. The faulty fittings and awkward methods of handling led to the rejection of the system in 1887.

The next step was in the replacement of iron barrels by wooden ones, made of eucalyptus staves 3 inches thick, the end boards being 4 inches thick. Iron hoops were shrunk on, and these were made thoroughly watertight. The barrels lasted about 10 months, but they often blew up, scattering and spattering everything with sand, gold solution, and acid, and poisoning the air with chlorine. O'Driscoll, in a series of articles written some years ago in "Engineering," described this method, and pointed out that anything going 10dwt. per ton and under was sent over the mullock dump. Yet the process was looked upon as perfect at the time. The barrel held a ton, and to this was added 70 gallons of water, 25lb to 30lb. of chloride of lime, and about the same quantity of chamber sulphuric acid.

Captain Richard saw how he could economise and abolish the barrel system entirely, substituting large rectangular open wooden leadlined boxes, and running sulphuric acid and chloride of lime in solution on at the same time—a modification, in fact, of the Muncktell process. The absurd method of running the solution in through the same pipe, as recommended by Muncktell, was not adopted; yet it is carried on in Victoria where the method is practised, with the extraordinary belief that nascent chlorine is being generated! As if chlorine was not always nascent when freshly liberated; yet, after having been liberated for some time before it ever comes in contact with gold, to assume that it is then nascent is palpably absurd. This was a great stride forward, and it has now been further improved upon by running a solution of chlorinated water, free from acids and salts, on the ore, while the wooden boxes have been replaced by 100-ton capacity cement tanks.

In the crushing of the ore a concurrent evolution went on. Wet crushing was objectionable, since a large amount of wet slime formed, and it was of no use amalgamating portion of the gold from material which had to be all chemically treated subsequently. Dry crushing in the battery was next tried, but the process was so slow, and the amount of dust formed so large, that it did not take long to condemn it. The avoidable errors in mining methods are appalling, and for a remedy we must look to a better system of scientific and technical instruction, better literature on modern methods, and better opportunities for those engaged in the work to become acquainted with what is done elsewhere.

The next dry crusher tried was the Bunche disintegrator. This was a disc, carrying on its face short steel rods. The disc revolved at the rate of 200 revolutions per minute. The casing was also supplied with projecting steel rods. The stone was dropped in between the casing and the disc, and was knocked to dust by the steel rods. This machine, or some form of it, perennially crops up and was tried some years ago at Bendigo. The same result was arrived at, the wear and tear being too great.

Then followed the Blake-Marsden stone-breaker and rolls. This was successful. A further trial was made with rolls having their faces spirally grooved, but these were almost useless. Dodge's disintegrator, as a forerunner of the ball mills, deserves mention. This machine consisted of a cast-iron drum, containing loose iron discs; the drum revolved; the ore was broken by the pressure of the discs against the sides and each other. This machine also failed to stand the test of practice. Next, Krom rolls were used, and, though excellent wcrk was done with them, these in their turn had to make way for the Krupp mills, which are doing excellent work at the present time. Jaques stone-breakers, made by Jaques Brothers, of Richmond (V.), have stood their tests unaltered, and serve now to crack the ore before it goes to the ball mills.

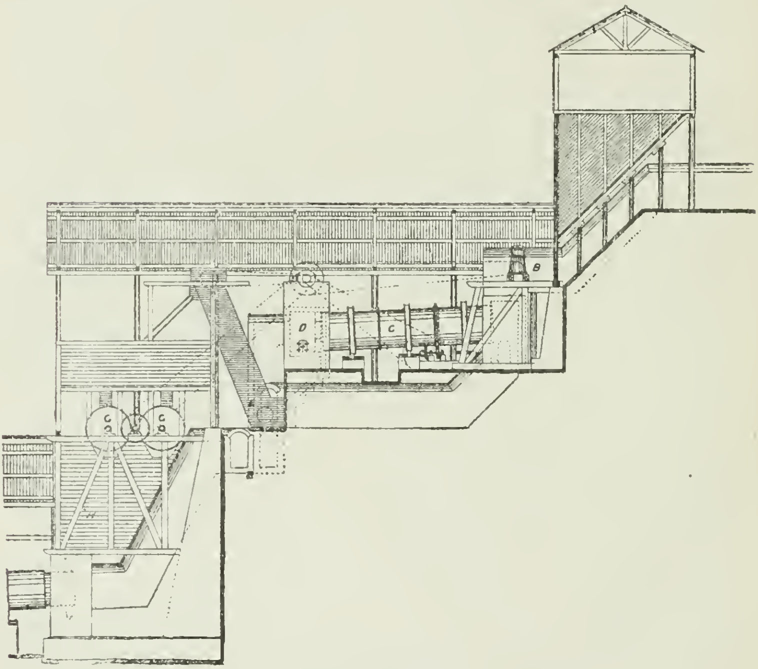
It was further found that in order to crush the ore so that it would pass through the screens it must be absolutely dry. Even chemically combined water has the effect of making finely crushed ore choke the screens. The drying is effected by making it pass through a heated revolving cylinder after it leaves the stone-breakers.

Even in the furnaces themselves, the "rising unto higher things" is as marked as with other appliances. The original type of reverberatories measured 11 feet by 10 feet by 6 feet; they then grew to 22 feet by 10 feet by 6 feet, but were all workd by hand rabbling. After these came the Ropp straight-line furnace. The experience at Mount Morgan was in favor of hand rabbling as against this mechanical furnace. The present plant has two furnaces, one a revolving cylinder, with many improvements on those commonly in use: this is used for dehydrating and oxidising material containing a small percentage of sulphides. The second furnace, as designed by Captain Richards, is wholly original and is used for the pyritic ore. It is stated that this will roast for less than 5s. per ton. Their experience in precipitation should also be of value on account of the very large quantity of bullion handled.

Sulphate of iron was condemned because of the large amount of room required for precipitating vats, and the contamination of the liquors which may have to be used over again. Sulphur dioxide and sulphuretted hydrogen were open to the same objections. Sulphide of iron was tried, but since considerable quantities of sulphate were formed this also was condemned. Sulphide of copper was tried. This necessitated the recovery of the copper with scrap iron, and the consequent contamination of the liquor. Charcoal was found to be an effective precipitant, and yet at the same time it did not contaminate the solutions, but rather tended to clarify them, so that the water from which the gold had been removed could be used again for charging with chlorine.

Even in smelting of the gold a radical change has been effected. The charcoal having been burnt off, and the ashes, impurities, and gold left behind, these were collected, smelted with fluxes, and the gold recovered. Most people who have had to melt such a mixture know the trouble caused even with small parcels, but when the amount runs into thousands of ounces what a task it must have been for the smelters. The first advance was to amalgamate the burnt residues and gold, retort, and then smelt the retorted gold—a comparatively easy task. A further advance has been made. The whole of the ash and gold from the charcoal furnaces is placed in a reverberatory furnace, after the style of a copper furnace, and there it is melted with the necessary fluxes, the gold and slag being tapped off from time to time.

The most modern portion of the works at Mt. Morgan for treating low-grade ore is known as the new West works. These were erected at a cost of £113,000, and form a most instructive study. The ore as delivered from the mine is discharged into the hopper marked A in sectional plan; thence it passes automatically to one of the stonebreakers (B.) Each stone-breaker is supplied through an opening over which a slide is moved up or down by means of a long lever, thus regulating the feed. Three of the stone-breakers are known as Jaques rotary No. 2, made by Jaques Bros., Richmond, Melbourne. The speed these machines are run at is 420 revolutions per minute. The fourth machine is a Blake reciprocating stone-breaker. The vibrations of the former are not so severe as in the



Sectional Elevation.

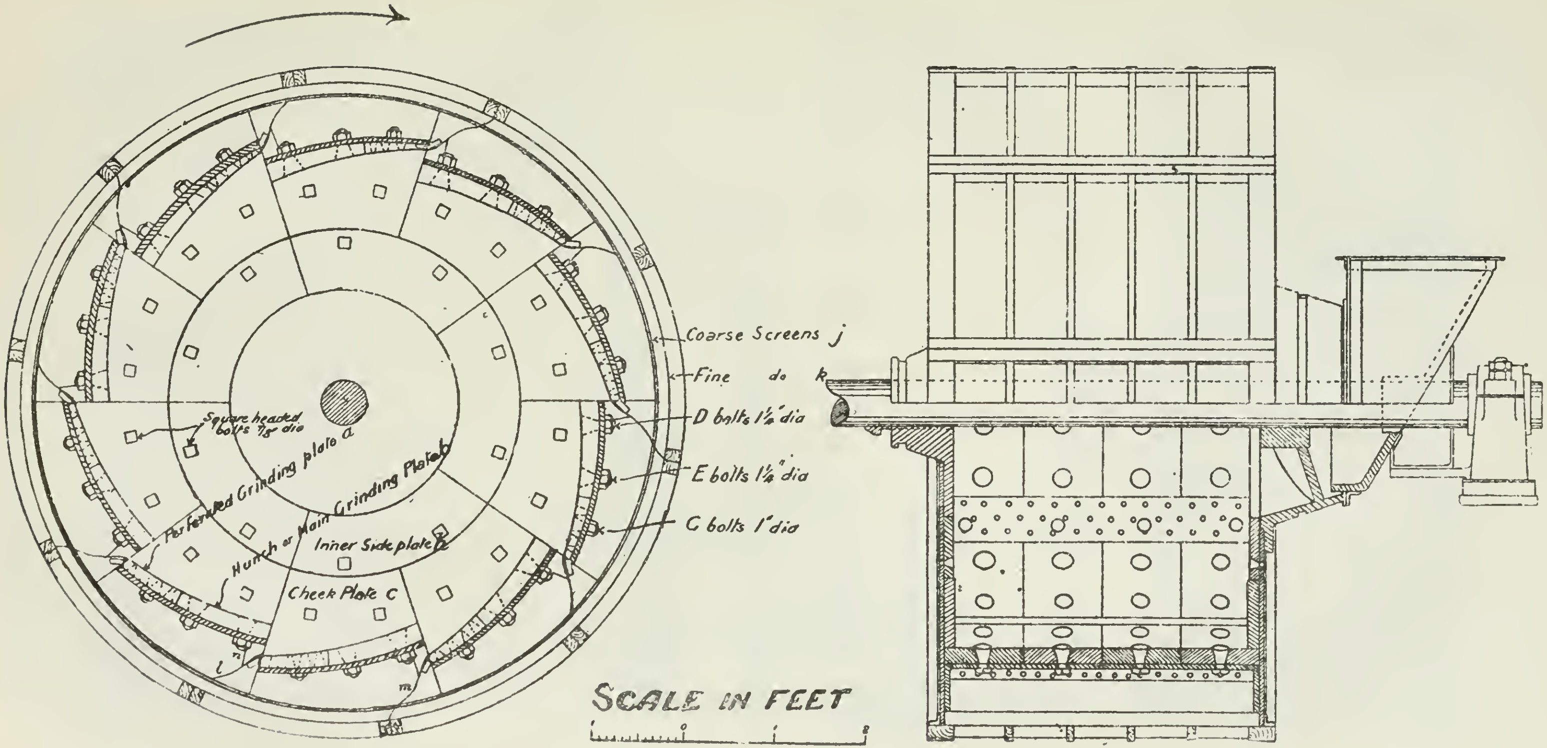
latter type, but the Blake shows a saving in wear and tear per ton crushed. One the other hand, the Jaques is the cheaper machine. Each machine will easily crush 100 tons in 24 hours. The horse-power required by the Jaques is six and by the Blake eight. The crushed ore after leaving the stone-breaker slides down a shoot into a revolving dryer (C.). The dryer is 30 feet in length, and 6 feet 8 inches internal diameter. The shell is made from ½-inch mild steel plates, and is built in five sections, one being 10 feet in length and the others five feet. The sections are united by cast iron tyres rivetted to the shell. These tyres are flanged, and each rests on a pair of friction rollers. The cylinder has a fall of ¾-inch per foot; the tyres are tapered to a like extent. The rollers are also tapered, so that tyre and roller represent frustra of cones, with the surfaces in contact horizontal. The cylinder rests on three pairs of rollers, the base of the rollers being set at an angle of 30deg. with the horizontal. The plates carrying the rollers rest on massive concrete pillars. These plates may be raised or lowered. The cylinder is driven at the rate of one revolution per minute by a small cog working into a circumferential spur wheel. The interior of the shell is lined with firebrick tiles 10 inches thick, and moulded radially. The ore travels slowly down the cylinder until it drops into an iron-lined shoot below the fire-box. The heat is supplied by a furnace (D) at the lower end of the cylinder, while at the other end is a dust chamber, which also serves for the roasting furnace, both being connected to the same stack. The amount of fuel used per ton of ore dried was 75lb. of wood and 30lb. of coal. The firebrick tiles were formerly obtained from Scotland at a cost of 2s. 3d. each, but they are now made locally for about half that figure.

Since the fire-clay and the bricks made from it appeared to me to be of a very high quality, and since it stood the ordinary tests as well as the best English firebricks, I obtained an analysis of the material, through the general manager, from the analyst to the company, Mr. Henry H. Bruce Leipner. He also kindly forwarded results of other analyses made of firebricks previously used.

	Firebrick, Clay-pit, near Brick Kiln.	Firebrick, Mountain Clay.	Firebrick, Mount Morgan.	Firebrick, local, made from sandstone overlying clay deposit.	Fireclay Shales, near Mt. Morgan.	Firebrick Shales, near Mt. Morgan.	Stourbridge Fire-clay.
	1	2	3	4	5	6	7
Silica, SiO_2	73.480	63.220	69.87	83.04	69.82	68.55	63.3
Alumina, Al_2O_3	29.922	29.334	24.39	16.46	22.29	31.49	21.18
Oxide of Iron, Fe_2O_3	3.048	6.566	4.93	—	min. tr.	min. tr.	1.8
Lime, CaO	.190	tr.	—	min. tr.	0.15	0.10	0.73
Magnesia, MgO	tr.	tr.	tr.	min. tr.	min. tr.	min. tr.	—
Alkalies, etc., K_2O	1.901	.777	.81*	.5*	0.25*	—	—
Water and organic matter, etc., Loss on ignition	0.118	.600	—	—	7.49	—	10.30
Hygroscopic water	0.174	.234	—	—	—	—	—
	99.833	100.731	100.00	100.00	100.00	100.00	—

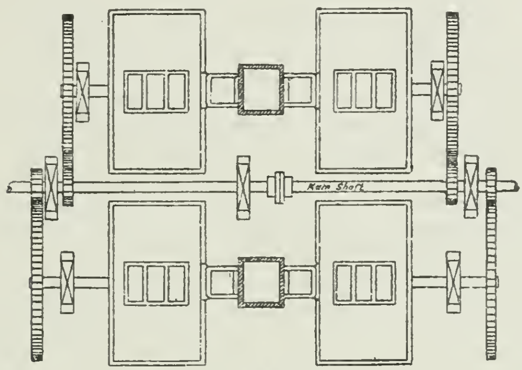
* By Difference.

Some of these bricks were made from the decomposed dykes on the company's property, but No. 5 represents the clay and No. 6 the firebrick which is now used. These shales form a valuable deposit, while they compare very favorably by analysis (No. 7) and by physical properties with the best Stourbridge firebricks. A curious result was arrived at accidentally. The manganese ore and salt are pulverised together in the ball mills; the fire-clay was also ground in the same mill, and the first lot run in came out evenly mixed with a very small quantity of manganese dioxide and salt as additional ingredients. The bricks prepared from this material were found to be much denser, and would ring like phonolites when clinked together. Some of the bricks came out with that peculiar green stain which has been found to be due to the presence of vanadium.

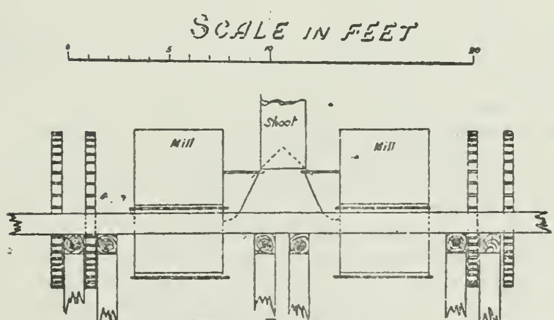


Square headed bolts 7/8" dia
Perforated Grinding plate a
Hunch or Main Grinding Plate b
Inner Side plate k
Cheek Plate c
Coarse Screens j
Fine do k
D bolts 1¼" dia
E bolts 1¼" dia
C bolts 1" dia
SCALE IN FEET

The ore after leaving the dryer falls into a hopper, from which it is conveyed by an elevator E into a hopper F, thence into the ball mills G. The elevators are Ewart's standard link chain, No. 85. A bucket is placed at every four links, and adjustable bearings are provided to counteract the lengthening of the chain by wear. The chain moves about five feet per second; the ore arriving at the top in buckets is emptied into the hopper almost continuously. The hoppers F are square boxes, lined with T. and G. boards. The ore is let into the ball mills through the shoots attached to the bottom of the hopper. Three men look after the sixteen ball



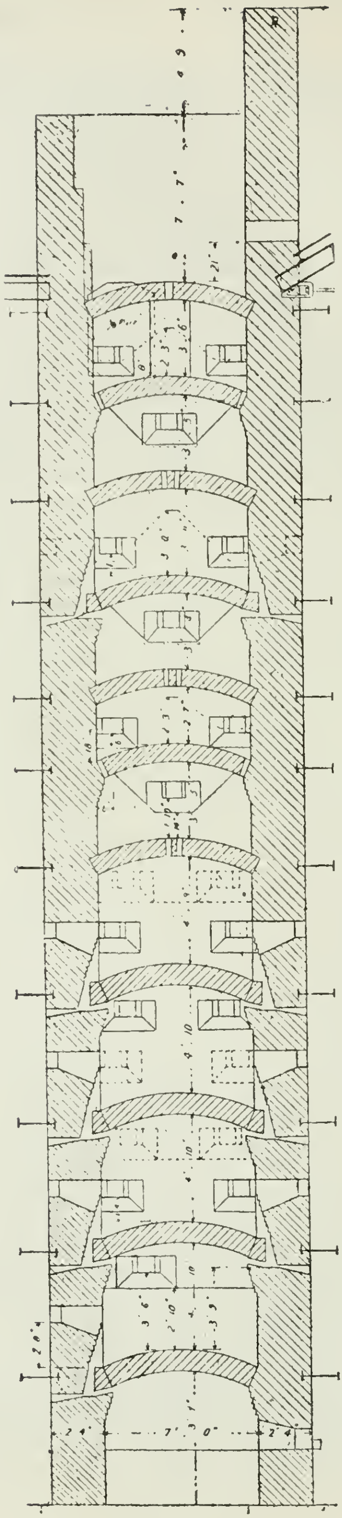
— PLAN —



— FRONT ELEVATION —

mills, feeding them from time to time by moving a sliding door across the shoot by a long lever. These men also change the screens, oil the machinery. With wages at 8s. per shift the expense amounts to 2.34d. per ton for this item. If an ore can be treated successfully in bulk, and if it is required to comminute it, then dry crushing is the most rational method to adopt. Yet it would be rash to recommend this method for all classes of ores, since many conditions may cause the final result to be far from economical and profitable. The advantages are that the whole of the ore may be treated without separation of slimes, which always accompanies wet crushing. The pulverised particles are more even in size than in wet crushing, while the proportion of fines produced is not sufficient to prevent percolation and leaching. If water is scarce the cost of providing it is saved, and a continuous system may be introduced whereby the ore is not wetted to be dug out of pits and re-dried, but instead passes at once from the mine through the processes already indicated, and is in a fit state for roasting. Krom rolls were successfully used at Mount Morgan, but the material crushed was for the most part soft. When hard material was struck in the lower levels the cost of crushing went up. Krupp's ball mills were introduced, and in a short time it was abundantly proved that they would actually crush the hard rock cheaper than the Krom rolls would crush the soft. The actual figures are as follow:—Krupp mill, 1s. 7d. per ton. Krom rolls, 3s. 10.36d. per ton. The return for a Krupp mill for a 12 months' average was 19 tons per day, the indicated horse-power being about 10. The screens through which the pulverised sand passed contained 1225 holes per square inch. It is needless to say that in the new Western works 16 of these valuable mills have been introduced, and that the Krom rolls have been overshadowed by them.

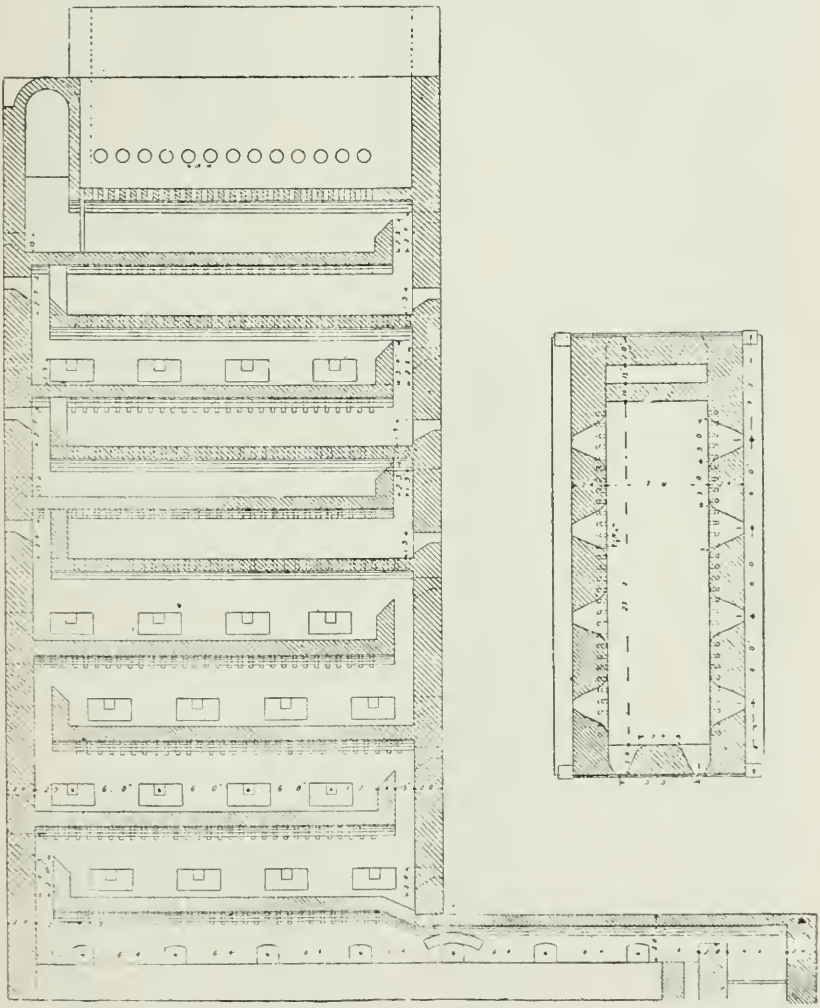
The Krupp mill is so well known that a full description of it would be redundant; yet a short account of it is necessary. It consists of a horizontal mild steel cylinder, having an axis through the centre, with a continuous feed through an aperture in the side near one of the bearings, and a continuous discharge through screens at its circumference. Between the axis and the outer rim are 10 curved cast steel plates, so arranged that their free edges and all sections of the plate parallel to these are parallel to the axis. The end edges abut against the vertical casing or sides of the cylinder. There is a drop, always in the same order, between the edge of one plate and thé edge of the next. Cast steel liners are also attached to the inner sides of the circles forming the sides of the cylinder. A number of steel balls are placed in the cylinder, the ore is fed in, and the whole mill caused to rotate so that the ore and the balls are continually dropping from plate to plate. When crushed, a portion drops through the perforated grinding plate, and thence to a coarse screen made of punched steel plate; the finer grains pass through this and meet a finer steel wire gauze, through which only the finest material passes. The coarser material is led back into the machine, and can only finally escape when fine enough to go through the outer mesh. The whole mill is encased in a steel casing, which prevents any escape of dust. Many improvements in these mills were suggested by the experience of the officers at Mount Morgan, and these were carried out by the Krupp firm. The steel balls for the mills are now obtained in England, since it has been found that they can be made there just as well as in Germany, and are much cheaper. The total cost of crushing, allowing for running expenses per ton, and also maintenance, is as follows:—Stonebreakers, 4.671d.; dryers and elevators, 11.153d.; ball mills, 15.533d.; engines, 2.923d.; boilers, 12.827d.; total, 3s. 11.107d. per ton. The power necessary to drive the four stonebreakers is 26 horse-power; for the 40 dryers and elevators, 24 horse-power; for the 16 ball mills, 208 horse-power. This power is obtained from a triple expansion horizontal steam engine, with cylinders 15, 24, and 39 inches in diameter, and 24-inch stroke. Five Babcock and Wilcox boilers of 120 horse-power each, working at a pressure of 150lb. per square inch, supply the steam. The air pump and water-circulating pump are connected with the tail rod of the lower pressure engine. The condenser is the corrugated iron pipe atmospheric one of the ammonia type. The speed of the engine is 96 revolutions per minute, and the horse-power developed, as obtained from indicator diagrams. is 321. The cost of crushing is exceedingly low, but this is achieved partly by working on such an extensive scale. Independent of this, it shows that the greatest care has been exercised in the design of such a magnificent plant, and that every man employed over each department has a thorough scientific training. In a paper by Mr. N. F. White to the A.I.M.E., to which I am indebted for supplementary information to that obtained from the general manager at the mine, the problem of dry crushing is exhaustively dealt with. The ore, having been dry crushed, drops into a close hopper, from which no dust can escape, thence it passess automatically into a cylindrical continuous roasting furnace. The continuous discharge revolving furnace is a cylinder 46 feet long, 7 feet 6 inches in diameter for 16 feet at the lower end, and 6 feet 9 inches in diameter for the upper end. The cylinder is made of ⅜-inch steel plate, stiffened by having the edges of the steel plates turned up, and also by cast iron tyres. The inner lining consists of fireclay tiles, set radially, the inner circle presenting a smooth surface of fire tiles. The cylinders is supported on four pairs of friction rollers, each of which bears on four massive concrete pillars. The centre of the axis of the rollers, and the centre of the cylinder, are equidistant, the angle formed joining these lines being 60 degrees. The same arrangement of surface of the roller and the track on the cylinder is adopted as with the dryers. They are both horizontal surfaces of contact of sections of cones rolling on each other, the slope in this instance being ½-inch per foot. The cylinder is driven by friction rollers going one revolution in 100 seconds. The ore is picked up by a pair of scoops on the end of the revolving cylinder; it passes into the furnace proper, travelling slowly downward all the time until, when thoroughly roasted, it drops into a hopper at the lower end of the furnace. Care is taken to admit no more cold air than is necessary, also to fit the end of the cylinder near the fireplace with a section which can be readily replaced. The general arrangements and construction of the furnace may be readily seen from the plans and sections shown. This furnace does a marvellous amount of work, roasting 1cwt. 1qr. per minute, while 3000 tons per month are put through. It should be stated that the ore treated in this furnace is comparatively free from sulphur. The ore conveyor is worked off the same shaft as the furnace. It consists of a revolving cylinder, with a pair of scoops attached. Vertical sliding doors admit more or less sand packed up on the other side of them, according to their height above the ground. The sand flows in to maintain its angle of slope; the scoops pick it up, pass it on the conveying cylinder; there it is partly cooled by a small pipe with a longitudinal slit, attached to the length of cylinder emptying water on the outer surface. When a revolution brings it above, the pipe descends into a reservoir below at the downward revolution, is carried up again full of water, and the process is repeated. The ore partly cooled is conveyed to the cooling and storage floors, from which it finally



Baffle Arches.

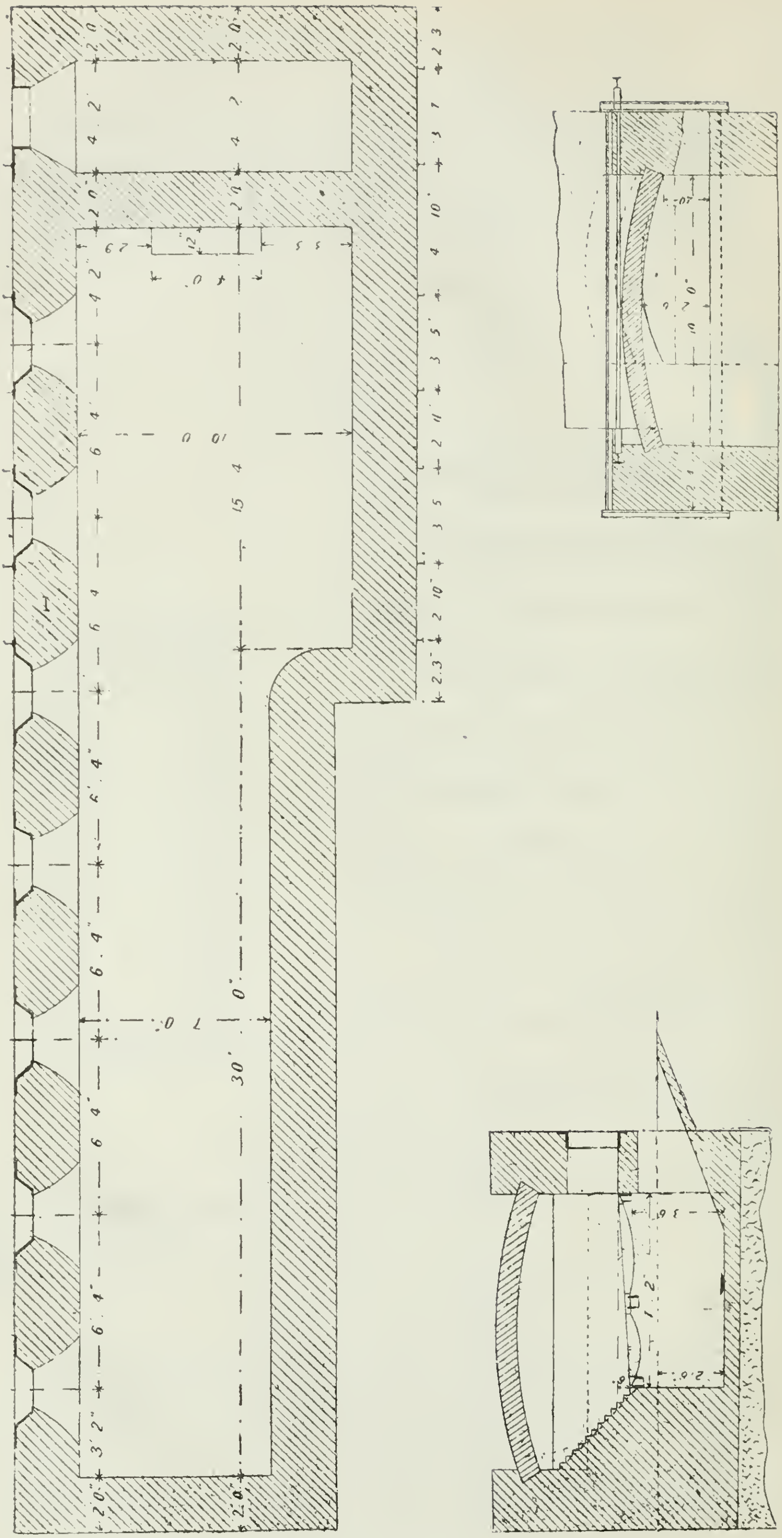
passes into the vats. The upper floor, on which the stonebreakers stand, is 136 feet above the top of the vats.

It was found that these furnaces were not fitted for roasting ores containing a high percentage of sulphur, Captain Richard attacked the problem in a systematic way, and has evolved a furnace which has certainly suited the Mount Morgan sulphides, and in which



Shaft Furnace.—Sheet No. I.

an enormous amount of stuff may be economically and thoroughly roasted. Attacking the problem from a physical point of view, he found the ordinary reverberatory furnace gave an efficiency of about 5 per cent.; that is, on calculating the heat reactions 100 calories were produced in order to get five back again. In order to overcome that waste it might be proved that a furnace should be indefinitely long, yet on the other hand other conditions come



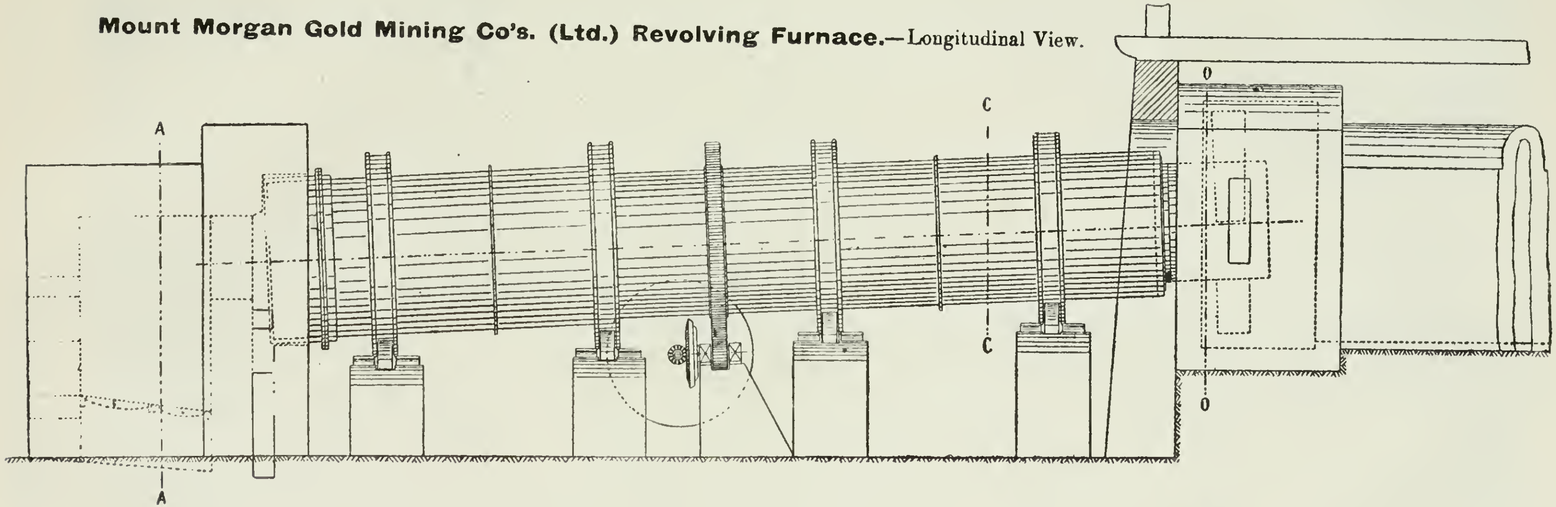
Shaft Furnace.—Sheet No. II

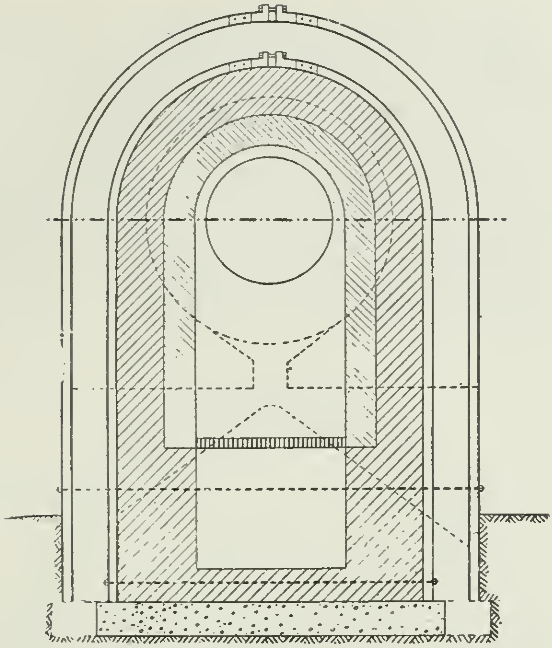
in, such as leakage of air, more handling of material, and the increased temperature required at the lower end of the furnace. From a chemical point of view it is found that not only is heat wasted by passing through the furnace, but that a very small proportion of oxygen is used up within the furnace itself, so that the main amount of fuel is burnt in heating inert nitrogen, water vapor, and oxygen not used, and allowing them to escape up the flues. Further, in roasting the heat reactions of the first atom of sulphur expelled from pyrites, the heat reaction of the sulphur burnt, and the iron oxidised at the same time show that if a furnace is properly arranged very little external heat would have to be supplied say with fuel. Of course, the heat lost by warming inert material, such as sand and oxide of iron, should be taken into account, but as the specific heat of such material is low the loss in that respect is not great. These and other considerations guided him in his choice, and led to the construction of the shaft furnaces. This furnace, as shown in the illustration, is 64 feet high, and stands on a base of 11 feet 8 inches by 29 feet 6 inches, and is terminated by a reverberatory furnace 25 feet in length. The principle of this furnace is oxidation of the pyrites, by causing them to fall in a finely-divided stream through a current of heated air. Stetefeldt attempted to do the same in his furnace, but the time allowed and the space passed through were altogether insufficient for perfect oxidation. In this furnace baffles, in the form of arches perforated on top, or having discharge holes at their base, prevent the pyrites from flowing too freely. The heated air is led over the hand-rabbled reverberatory portion up a short flue at the end of the first arch, over the length of the second arch, up another flue, and back over the next one, and so on. The sand in descending has to pass over and through 11 of those arches, each 7 feet wide. In some, as indicated in the drawings, it pass through the central holes in the arch; in others it piles up against the walls, and in flowing down through small openings near these, it is forced over the next arch by a jet of compressed air. On coming to the lowest floor it is worked along by hand and discharged.

The furnace is well braced with H irons. There are numerous doors provided in case a blocking of passages with sand may take place, and as the best proof of the care bestowed on their design, eight of them have been erected with barely an alteration from the original. The amount of ore—carrying 10 per cent. of sulphur—which one of these will put through is 200 tons per week, while the cost of roasting has been reduced to 5s. per ton. Crushing and roasting of the sulphide ore may therefore be done for 2dwt. per ton—a result which may lead to the profitable working of many large bodies of low-grade ore when mining managers take advantage of the Mount Morgan lesson.

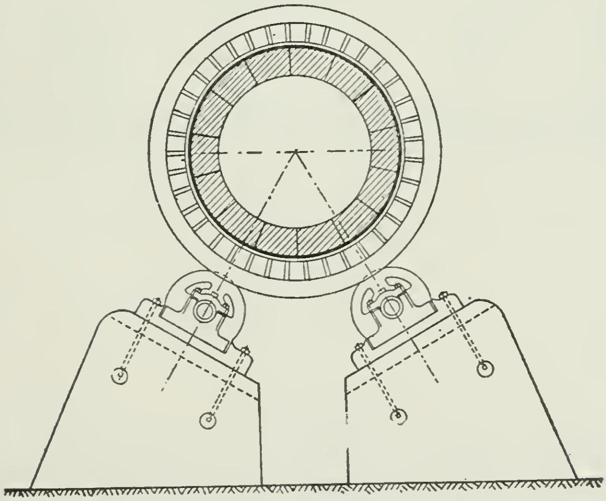
The illustration herewith presented is that of the continuous discharge revolving furnace. It is a cylinder 46 feet long, 7 feet 6 inches in diameter for 16 feet at the lower end, and 6 feet 9 inches in diameter for the upper end. The cylinder is made of ⅜-inch steel plate, stiffened by having the edges of the steel plates turned up, and also by cast-iron tyres. The inner lining consists of fireclay tiles, set radially, the inner circle presenting a smooth surface of fire tiles. The cylinder is supported on four

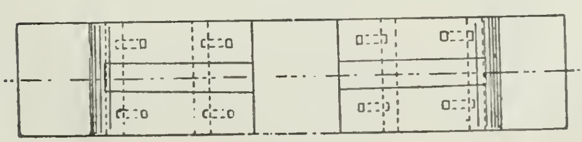
Mount Morgan Gold Mining Co's. (Ltd.) Revolving Furnace.—Longitudinal View.





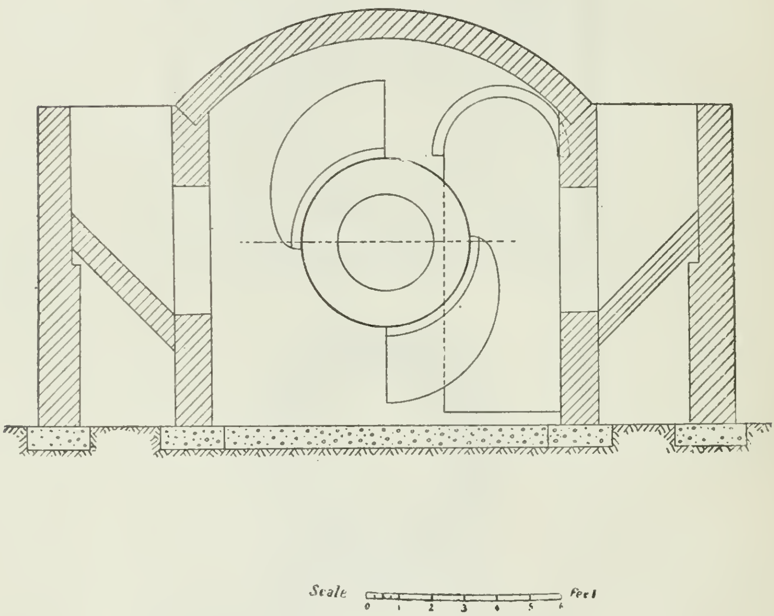
Section A.





Section C C.

pairs of friction rollers, each of which bears on four massive concrete pillars. The centre of the axis of the rollers, and the centre of the cylinder, are equidistant, the angle formed joining these lines being 60deg. The same arrangement of surface of the roller and the track on the cylinder is adopted as with the dryers. They are both horizontal surfaces of contact of sections of cones rolling on each other, the slope in this instance being ½-inch per foot. The cylinder is driven by friction rollers, going one revolution in 100 seconds. The ore is picked up by a pair of scoops on the end of the revolving cylinder; it passes into the furnace proper, travelling slowly downward all the time until, when thoroughly roasted, it drops into a hopper at the lower end of the furnace. Care is taken to admit no more cold air than is necessary, also to fit the end of



Section O O.

the cylinder near the fireplace with a section which can be readily replaced. The general arrangements and construction of the furnace may be readily seen from the plans and sections shown. This furnace does a marvellous amount of work, roasting 1cwt. 1qr. per minute, while 3000 tons per month are put through. It should be stated that the ore treated in this furnace is comparatively free from sulphur.

Continuing the description of the works dealt with, in which the ore was traced in its progress from the hoppers to the stone-breakers, thence through the drying furnace, to be elevated and fed into ball mills, to be delivered from these into the roasting furnace, through the coolers to the cooling and storage floor. The next receptacle for the sand ready for chemical treatment is the vat. The construction of each of these at Mount Morgan is absolutely original. At the mine the wooden lead lined vats, the stoneware stills, and the expensive leaden vessels of all descriptions, have had their day, and have had to give way to the cheaper and more durable material, cement.

The vats at the west works have an internal capacity for ore 100 feet long, 12 feet 6 inches wide, and 3 feet 6 inches deep. A fall of three inches is given to the bottom of the vat from end to end. The vats are arranged in groups of four. Two side by side, with a wall dividing them for their whole length, and two end on to these. The gold solution and chlorine solution pipes run between the ends of the vats. The walls are two feet, and the floors 18 inches thick. The walls are first built of cement, six to one, then the floors are filled in; the junction with the walls being roughened so as to make a perfect joint. The final linings are put on in the way a room is plastered, the old surface being roughened while wet, then allowed to dry, and a new coat put on. This layer is two to one, and is laid on for half-an-inch in thickness; the final layer is laid in neat cement. Many of the vats, however, and all the lead work in contact with chlorine or chlorine solutions are coated with a mixture of two parts of pitch and one part tar, which has been boiled for two days and put on hot; this gives a beautiful, even, hard enamel-like glaze to the surface. The filter bottoms are laid on bearers of hardwood, 3 inches by 2 inches, which lie along the length of the vat; these are spaced 2 feet apart. A loose flooring of hardwood boards, 9 inches by 2 inches, covers the whole surface. A number of 1½ inch holes are bored through this flooring. The filter bed consists of gravel of diminishing dimensions from 1 inch to ¼ inch, while over this is laid a layer 9 inches thick of ⅛ inch sand. The filter bed is about 18 inches above the bottom of the vat, so that the total depth is 5 feet at one end and 5 feet 3 inches at the other. A timber frame spanning the width of the vat carries four trucks; these serve for filling and discharging the vats. This carrier moves along on wheels running on rails on the edges of the vats, and by its means 100 tons of wet sand can be discharged by six men in eight hours.

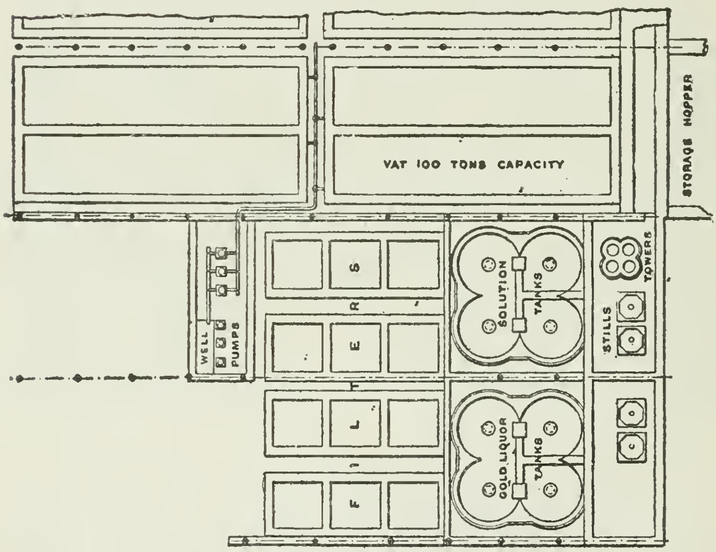
The solutions are led to each vat by a branch from a main which, as stated before, passes between the ends of the vat. The branch pipe has a length of rubber and a screw clip attached. The gold liquor main, which runs under the solution main, has a branch pipe from every vat, also a valve attached. This main may be connected on to suction pumps, which are used to assist filtration. Provision is also made for taking samples of liquor from the outlet pipe of each vat. The suction pumps are made of type metal—i.e., an alloy of antimony and lead. The mains carrying chlorinated water and solutions are made of ordinary drain pipes dipped in the boiling tar composition mentioned. The joints are packed with tarred oakum. This is a great improvement on the old system of lead pipe and lead linings. Lead is rapidly eaten away by chlorine solutions, especially if the liquors are warm.

Chlorine is generated in stills similar in type to those used in the chlorine manufacture from hydrochloric acid. The stills are octagonal in shape, 4 feet 8 inches in width, and 4 feet 6 inches in height, and are built of imported Yorkshire flagstones. The stones are grooved along the edges, and in this groove is laid a strip of round rubber, the whole being clamped together to form a perfect joint. The discharge hole has a cover clamped on in the usual way. Steam is admitted through ebonite taps at a pressure of 150lb. per square inch. The outlet pipe for the chlorine is of earthenware, the joints being packed with the oakum and tar mixture. Each of the stills made of imported material cost £98. It was found necessary to patch these from time to time; this was done with cement. The material stood so well that a still was built of it: this has answered almost as well as the flag, and the cost was only £12. Manganese dioxide, salt, and sulphuric acid are used to generate the chlorine. The mixture is made up according to the following equation:—

$$MnO_2 + 2NaCl + 3H_2SO_4 = MnSO_4 + 2HNaSO_4 + 2H_2O + Cl_2,$$

but 3 per cent. of MnO_2 in excess of the theoretical

Fig 1.

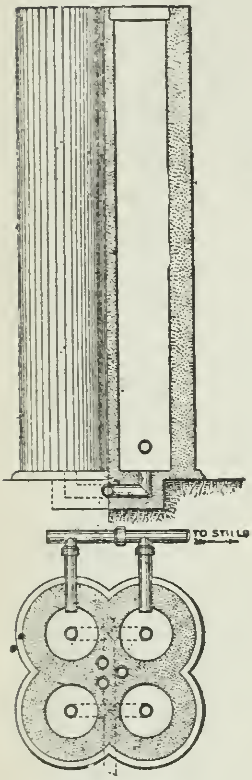


Scale, 1 inch = 40 feet.

General Arrangement of Chlorination Plant.

quantity is used. The manganese ore is fairly abundant on the coast near Rockhampton, a lode of it being worked near Gladstone. It contains about 75 per cent. of the dioxide, and costs at the mine about £6 10s. per ton. The material is massive and hard, and has to be crushed in the ball mills through a 3000-mesh sieve. The powder is then mixed with the requisite quantity of salt, and the mixture is ground to an impalpable powder in Chilian mills. The cost of the salt at the mine is £3 10s. per ton. Four hundred and eight pounds of this mixture are fed into a still through a funnel pipe from the floor above the still. The charge is exhausted in about four hours. Chamber sulphuric acid is then added, about 3 per cent. in excess of the theoretical quantity. Sulphuric acid is made at the mine, the plant being of the usual

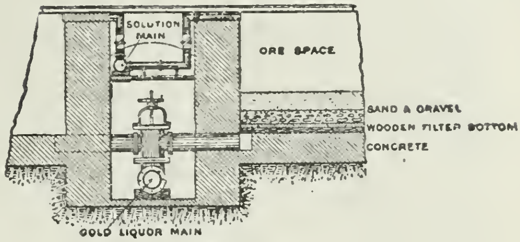
Figs. 3.



Scale, ¼ inch = 1 foot.

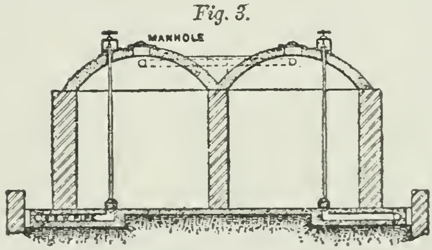
Sectional Plan and
Elevation of Towers

Fig. 4.



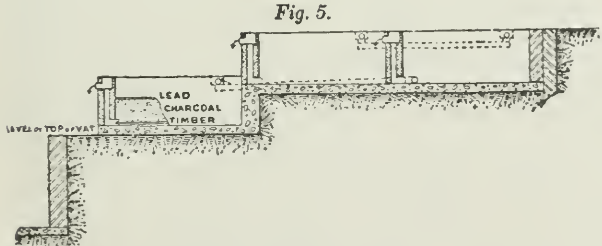
Scale, ⅛ inch = 1 foot.

Fig. 3.



Scale, $\frac{1}{16}$ inch = 1 foot.

Fig. 5.



Sectional Elevation of Vat.

description, only that sulphur is burnt instead of pyrites. Chamber acid having a specific gravity of 1.615 is produced at a cost of about £3 per ton, the amount used for the year being about 3500 tons. The escaping chlorine is led away to towers, through which water trickles over stones to absorb the gas. Since the chlorinated water has to be kept at a constant strength, a ready quantitative test has been supplied to the men in charge. The water supply to the towers is thus made to vary with the amount of the chlorine present, so that the solution is kept constant about 350gr. to the cubic foot, or less than .1 per cent. The towers or scrubbers are built of concrete, four of these being built in one nest. The internal dimensions of each are: Height, 20 feet; diameter, 2 feet 3 inches; the external measurements being 25 feet and diameter 5 feet. Each tower is packed with stones, old broken crucibles, and glass bottles to within about 1 foot from the top. All these are carefully arranged so as to secure an even downward flow of water. The top of the tower is fitted with a leaden dish, through which a number of open leaden pipes project upwards for one inch. These pipes are covered by inverted leaden cups. This arrangement forms a water joint, and yet allows a stream of water fed in above to pass down through the leaden pipes into the scrubber. The chlorine gas is let in at the bottom, and in rising meets the descending stream of water and is partly absorbed; the balance is led away through a pipe from the top of the first tower into the bottom of a second one, where the same process goes on. The chlorinated water is led away to solution tanks. These are built of brickwork lined with cement, and face coated with the mixture of tar and pitch. Four of these are built in one nest, each one being 11 feet 4 inches in diameter and 40 feet in height, with a dome-shaped top, fitted with a man-hole. An indicator is attached to show the quantity of liquor used. The solution is allowed to pass out through a valve at the bottom to the mains, and then enter the sand vat. The vat having been filled with sand, the surface is levelled off and that at the sides raked a few inches higher to prevent solutions running down there. The chlorine water is then run over a board on to the sand. When it is covered the outlet valve is opened and the solution allowed to flow through the filters until no chlorine water appears on the surface. The surface is then raked over to cover up any cracks or air blow-holes formed by putting on the first liquor. Chlorine water is again run on continuously until the escaping liquor is strongly charged with the gas. When this has happened the liquor is allowed to rise 4 inches above the surface of the sand, and is kept in contact until the chlorine acts on the gold. In about 20 minutes the strength has been found to become much less, so this solution is run off and more chlorine liquor added. This process is continued until the gold is dissolved. It is sought to give a number of separate washings rather than allow a single strong solution to act on the ore.

It takes about 2½ hours for the oxidised material from the upper workings to become charged with chlorine, and four hours for the roasted pyritic material. The chlorine appears to be used in oxidising ferrous salts, and in rendering basic salts soluble before it is free to attack the gold. Ferric sulphate $Fe_2 (SO_4)_3$ and sulphate of aluminium $Al_2 (SO_4)_3$ are abundant in the first liquor coming from the vat, while little or no gold is dissolved. It is noticed that if the escaping solutions from the first liquor are yellow or brown, then some difficulty may be anticipated in subsequent treatment; if the solutions come away clear, then gold will be readily and rapidly dissolved.

With regard to the pyritic ore, it is found that a slow roast gives a better result than one during which a high temperature is attained. The generally accepted re-actions for the changes which take place in roasting pyrites are not those which occur in practice. Plattner has stated, and his statement has been accepted by a number of other authorities, that in roasting pyrites the following changes occur. First on heating sulphur is distilled off

$$3FeS_2 = Fe_3S_4 + S_2,\ 7FeS_2 = Fe_7S_8 + 3S_2,$$

the material remaining behind being magnetic sulphides. The sulphur vapor coming in contact with the oxygen of the air becomes sulphur dioxide, while the intervening mass of porous brickwork causes some of the dioxide to become trioxide, or

$$S_2 + 2O_2 = 2SO_2, \text{ and } 2SO_2 + O_2 = 2SO_3.$$

The next step given is not clear, for it is stated that the sulphides become slowly oxidised, and through the intervention of the sulphur trioxide (SO_3) become converted into sulphates, which are decomposed in turn to become oxides. Thus

$$2FeSO_4 = Fe_2O_3 + SO_2 + SO_3.$$

Roberts-Austin states that probably by far the greater part of the iron sulphide only becomes sulphate for a very brief period, being decomposed into the oxides of iron, mainly ferric oxide, sulphur passing off.

It seems marvellous that these re-actions should be accepted in face of the facts that in almost all roasting furnaces that while the first change indicated is correct, the magnetic sulphides of iron are converted into magnetic oxide by the heated air passing over the surface. This in its turn is slowly converted into ferric oxide (Fe_2O_3) on being brought slowly into a part of the furnace at a higher temperature. Ferrous sulphate is not formed and decomposed as indicated, since it is only at the lower end of a furnace that the dissociation temperature is reached.

It is found at Mount Morgan that a sulphating roast, or a roast at a low temperature with a limited supply of air, which probably is the only one the text-book authorities have ever considered, places the roasted ore in better condition for chlorination than that generally given in a reverberatory furnace. By roasting very slowly a large amount of the sulphide of iron may be made to pass through the sulphate stage, while the copper present is mainly converted into sulphate. It must not be taken as a general statement that this method of roasting would be suitable for all ores. Mount Morgan has solved its own problem, but other details to suit varying conditions would have to be adopted. Copper exists in the ore to a payable extent in the lower levels, and steps have been taken to recover this metal from the solutions.

As soon as the gold is dissolved the chlorine liquors are run off until the ore surface appears. Water is then run on and drawn off, this being repeated until there is no re-action for gold when tested with ferrous sulphate. It is found that a much more sensitive re-action may be obtained from ferrous sulphate made by dissolving cast iron than from purer material. Filtration is assisted by vacuum pumps, which are made of type metal, are vertical, and have two external valves. The plungers are 10 inches in diameter, and the stroke is 16 inches. The solution is drawn from the vats and discharged into a small well. Force pumps, fitted with hardwood plungers 9 inches in diameter with a stroke of 16 inches, force the liquors from the well into the gold storage tanks. The solutions are drawn off at an even rate and run through charcoal filters. These are built of brickwork on a concrete foundation, the internal surfaces being cemented and coated in the same manner as the vats. Their internal dimensions are 10 feet by 12 feet by 4 feet 6 inches deep. They have a wooden filter bottom. Cheese cloth is laid over this to prevent the fine charcoal from falling through. The charcoal is broken into fragments in a disintegrator, then sieved through screens having 20, 30, and 40 holes per linear inch. Grains which remain on the largest sized sieve are re-crushed, those which stop on the 30 sieve are termed coarse, and those that stop on the 40 are called fine, the dust passing through the 40 not being used. The filter is packed by stamping the charcoal round the edges to prevent free flow of the solutions down the sides. A layer of 5 inches of coarse is laid on the cheese cloth, then a layer of 2 feet of fine; on this comes another layer of 5 inches coarse. The whole surface is then covered with a sheet of perforated lead. This prevents the charcoal rising or becoming disturbed with the inflowing solutions. The space above the charcoal serves as a reservoir. The general arrangement of the filters may be seen from Fig. 5. The gold solution flows downwards through the charcoal, passes up a pipe from the bottom, and flows through the next filter. The bulk of the gold is obtained from the upper layer of fine charcoal. The charcoal when ground is stored in bags, each holding about ½cwt., and about two of fine are crushed for every one of coarse. Fifty bags of the material will precipitate 2000oz. of gold. The charcoal becomes exhausted or loses its power of precipitating gold in about six months.

The following list of analyses of the roasted ore from Mount Morgan has been supplied by Mr. Henry H. Bruce Leipner, analyst to the company:—

		1 Mundic (pyrites) roasted.	2 Oxidised ores calcined.	3 Low-grade ore calcined.
Moisture		.376	1.006	.700
$Fe\,SO_4$	Soluble	.021	.836	.209
$Al_2\,(SO_4)_3$	in	—	.409	.208
SO_3	water	.289	.739	.272
$Si\,O_2$		80.954	89.770	84.310
$Fe_2\,O_3$		16.469	3.598	12.031
$Fe_2\,O_3\,2SO_3$		1.040	2.384	1.518
$Al_2\,O_3$		.430	.788	.838
Cu O		.680	—	—
Ca O		trace	trace	trace
Mg O		trace	trace	trace
		Per ton. Oz dwt gr	Per ton. Oz dwt gr	Per ton. Oz dwt gr
Gold		1 13 2	1 16 10	0 9 11

These analyses show how highly siliceous Mount Morgan ore is, and though a great deal of it looks like kaolin, and is soft, white, fine, and powdery, the amount of alumina present is so small that there can be very little felspathic material present. How such a mass of silica became concentrated is, in the opinion of most people, due to hydrothermal action, and the general manager, who has had more opportunities of judging than casual observers, holds that the upper portion was due to a thermal spring, thus upholding Jack's original theory.

The solutions from the sand are highly charged with salts, consisting of basic and other sulphates held in solution by chlorine, some of which separates out as soon as the chlorine is destroyed. It will be seen that if 100 tons of ore be treated and 150oz. of gold precipitated, then if only 5 per cent. of the salts in solution separate out, about 1cwt. of these will come down with the gold. It is in dealing with problems of this sort that the young student trained only in laboratory methods will be completely upset when put in charge of works. His assays will turn out all right; he will inform his directors that, based on the difference between the tailings and sand assay, he will get a certain amount of gold. The bulk of his precipitate will astonish him, and the results of his smelting will astonish him still more. A tiny button of impure gold, masses of slag, and the collection of the whole of his ashes, leave him in such a fix that his last state is worse than his first. His lesson has been learnt at the expense of the mine, and unfortunately sometimes at that of his own reputation. It is astonishing to believe that there are still experts who believe in training students to work with a plant of diminutive dimensions, a plant in which many of the actual operations which take place in practice are never seen or considered. Students are taught assaying, chemistry, drawing, surveying, and other subjects to a full scale, such as they will be required to work afterwards, but metallurgy is taught to a scale of an inch to the thousand feet, and a student is supposed to understand a lot of facts, as well as principles, absorbed through his ears alone, without actual contact with the apparatus he will have to subsequently deal with. One might as well claim that mining could be taught efficiently by a superficial study of the model mines in museums, or that geology could be better taught by building microscopic sections of the various formations and teaching the students from these models.

Practical men will know the meaning of keeping solutions clear, and at Mount Morgan, before they had their reverberatory furnace, they had sand filters or separators. These were round, lead-lined vats, 3 feet 6 inches by 5 inches, having a false bottom of perforated wood, then 5 inches of coarse gravel graded, and finished off by a layer of 8 to 9 inches of sand. The sand is tightly packed. These filters catch the basic salts, and in course of time become cemented together. They are then broken out, re-calcined, and the gold removed. The solutions must flow very slowly through the sand, or the basic salts will be carried on to the charcoal.

The solutions are then run down through the charcoal. After going through the first filter the solution rises through a pipe and passes through the second; the water can then be run on the ore as a water wash, or can be used for recharging with chlorine, or run to waste. It has been found that it is not advisable to pass chlorine into water that is highly charged with salts. In the new West works chlorine is not blown out of the solutions by means of steam. If the solutions are 80 to 90deg. Fahr. in temperature the gold will readily precipitate on the charcoal. The chlorine is rapidly destroyed in presence of sunlight, so that the means usually taken to get rid of it might be simplified. It is remarkable that the cause of the precipitation of gold in charcoal has never been satisfactorily explained. The charcoal is burnt in reverberatory furnaces with dust chambers attached. The analysis of the ash remaining was as follows, when the sand filters were used:—

$Ca\,SO_4$	2.343	Soluble in water.
$Mg\,SO_4$	trace	
K_2SO_4, Na_2SO_4	1.777	
KCl, NaCl	trace	
$F_2\,O_3$	25.825	Soluble in hydrochloric acid.
$Al_2\,O_3$	3.475	
$Ca\,SO_4$	1.751	
Ca O	1.018	
Mg O	.273	
$Si\,O_2$	13.610	Insoluble in acids.
$Al_2\,O_3$	1.903	
Ca O	.925	
Mg O	.570	
Au	45.915	

At present the gold from the charcoal ash runs from 20 to 40 per cent. This is mixed with the requisite quantity of fluxes and placed in a short reverberatory furnace, built after the style of a copper-smelting furnace. The slag is tapped off periodically, and the gold is run off and cast in ingots weighing about 300oz. each. The assay of this is about 990 fine. The slags, which accumulate by the ton, were sent to Aldershot Smelting Works. It has been found that these can be treated more economically on the spot. The chlorine solutions are tested with an indigo solution. Experience has shown it to be of greater service and more easy application than the ordinary re-agents used, such as arsenite of sodium, ferrous ammonium sulphate, etc., and I obtained from the analyst of the company the following details:—"Three ounces of finely ground Indigo is added slowly, in small portions at a time with constant stirring, to 4lb. of strong, pure sulphuric acid (H_2SO_4). (Not Nordhausen as recommended by most authorities for the solution of Indigo, as it was found that the final color re-action with the chlorine water was not distinctly marked, and more especially so at night when working with the aid of the electric light)."

Any great elevation of temperature is to be avoided, and it is advisable to place the vessel—preferably a porcelain evaporating dish 10 inches or 12 inches in diameter—in cold water. When the whole of the indigo has been added to the acid, the vessel is covered and allowed to remain for at least 48 hours, when the contents are poured into about twelve times the quantity of water (any great rise in the temperature being avoided), the whole mixed well, and filtered first through fine mill silk and then through a coarse folded filter. (See Fresenius, p. 76). The standard indigo solution I have found to answer all requirements is one of such a strength that 1 cc. is equivalent to 10 grains of chlorine per cubic foot of chlorine water, using a miniature gallon, namely—70 cc. of chlorine water used for each test. This standard indigo can be made by taking a solution of chlorine water, estimating the amount of chlorine in the same by a decinormal arsenious acid solution—

$$1\text{ cc} - \frac{N}{10}\,As_2\,O = 0.0355\text{gr. Cl.}$$

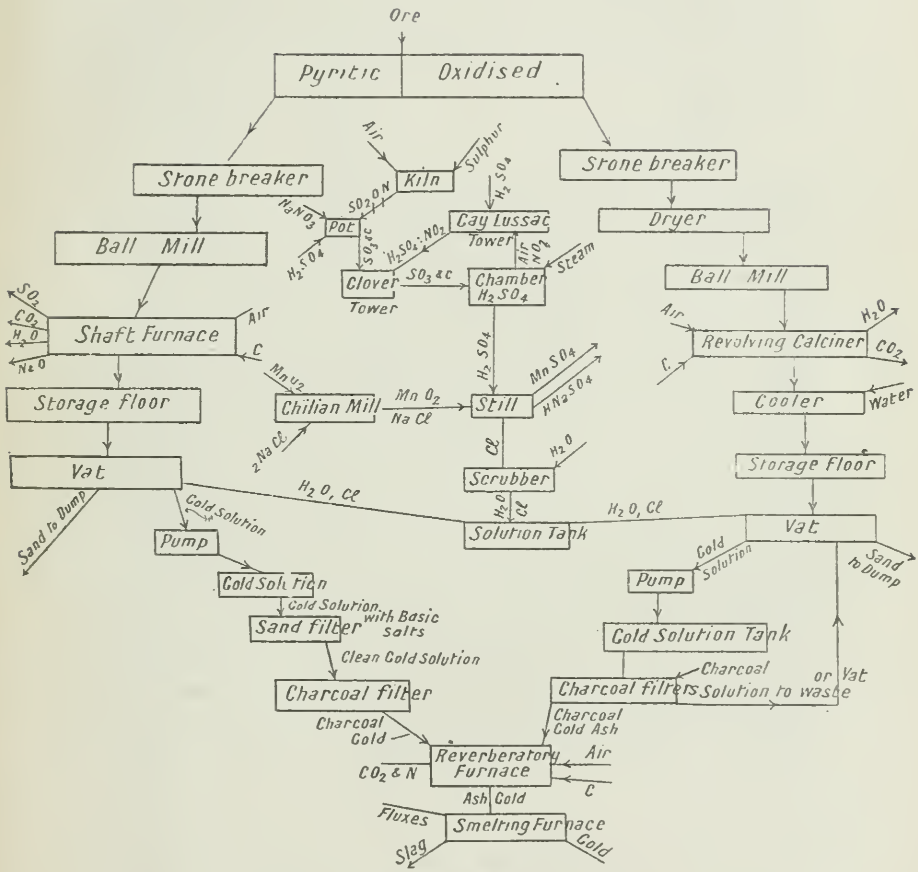


Diagram of the Mount Morgan Plant.

And after having found the strength of your chlorine, titrating with indigo solution and calculating the required amount of water to bo added, so that 1 cc will equal 10 grains of chlorine per cubic foot, using 70 cc of chlorine water for a test,

70 cc chlorine water taken

$$X \text{ cc } \frac{N}{10} As_2O_3 \times 3.55\text{gr. of Cl per gal.}$$

$$X \text{ cc } As_2O_3 \times 22.1\text{gr. of Cl per gal. } \frac{//}{}$$

The assaying department is under the charge of Mr. H. St. J. Somerset, and from him I obtained much information as to methods of assay, strength of solutions, and other details. The assay fluxes are made up in bulk, and evenly mixed in the following quantities:—

	Ordinary ores.	Siliceous ores.
Litharge	18lb.	48lb.
Sodium bicarbonate	54lb.	57lb.
Borax pulv...	9lb.	—
Flour	26oz.	25oz.

For pyritic ore 400 grains is taken, and 1500 of flux; an iron spike is introduced, and a lead button weighing 350gr. is obtained.

Mr. Somerset found that he could get exactly the same returns from pyritic ore, roasted or unroasted, and consequently has dispensed with this preliminary operation. Ores carrying from 1½oz. to 3½oz. per ton yield all their gold but 20gr. to 26gr. by chlorination. Assays are made on from 2000gr. to 3000gr. Low-grade ores going about 12dwt. per ton yield from 93 to 94 per cent. of their gold to the chlorine. Roughly speaking, crushing and roasting cost about 8s. per ton, chlorinating 6s. per ton, and mining about 8s. per ton, so that for about £1 per ton the various operations necessary for the recovery of gold are conducted. About 2s. 6d. more per ton is required for general management and other expenses. The low-grade ore as treated in the works described can be treated at a profit if it contains over 4dwt. per ton, the actual returns obtained being slightly over 7dwt. per ton. Much of the material which had previously gone over the dump is now yielding dividends. The average of all the ore treated is between 11dwt. and 12dwt. per ton, and since the whole of this, amounting to half a million tons per annum, has to be treated chemically the results form an object lesson for the whole world as to the treatment of low-grade ore The dividends paid during the last seventeen years amounted to 6¼ million pounds.

The Queensland Smelting Company.

The works of the company are situated at Aldershot, about six miles from the port of Maryborough, and on the main trunk line from Gladstone to Brisbane. Coal from the Burrum coal field may be obtained within a radius of a few miles from the works. The company carries on the business of buyers, smelters, and refiners of gold, silver, lead and copper, whether in ores, concentrates, mattes, cyanide slags, or other furnace products. It is the only Customs work of its kind in Queensland. Since lead is the life-blood of such works, the company has always had a direct interest in mines in various States, but at present their own mine at Pinnacles, N.S.W., supplies all the necessary metal.

The ore, as delivered in railway trucks, is weighed, and the tare of the trucks deducted. If the material is in lumps it is raised to an upper floor. For elevation to the feed and upper floors, two hydraulic lifts, each capable of raising one ton, or an inclined way, along which a three-ton load may be hauled by a winding engine, is made use of. The lumps of coarse ore are broken in a Blake Marsden crusher, having jaws 9 by 15 inches. The fluxes are also cracked up in this. The broken ore then goes through a Dodge breaker, and then through rolls, which crush it down to three-sixteenths of an inch. Since it is not desirable to have too much fine ore for blast furnaces, the fine crushing is mainly done for sampling purposes, so that unless the whole of the ore requires roasting, only every second, third or fifth bag is chosen from a parcel for the breakers and rolls. In order to make doubly sure of accuracy of sampling, the work is done in five-ton lots; that is, if 20 tons came in four lots would be made, and each lot sampled down independently. This method appears to be better than working down a large parcel to a few ounces, and only checking the assays against each other. A system of checking the sampling as well as the assaying is highly desirable.

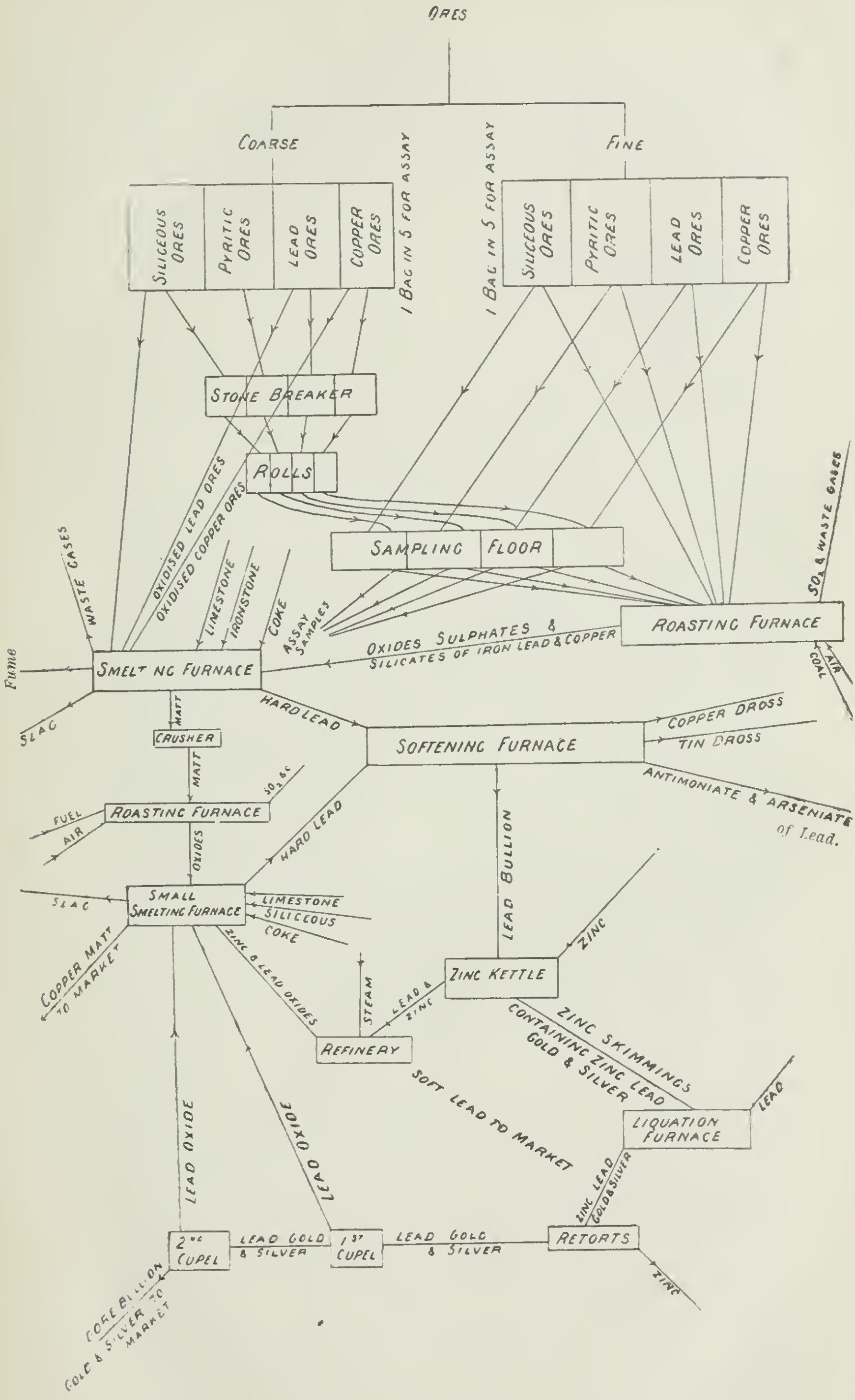
The portion selected as a sample is run over a point in a steady stream; a small cone of sand forms. This continually receives concentric coats until the final conical heap is formed. The cone is then spread out evenly by working its sides down with a shovel, and becomes a low circular heap, whose centre is the point over which the ore was run. The workman then divides this in two diameters at right angles, and shovels away the opposite quarters. This is again run down, another cone spread out, and again quartered, and the process repeated until a few pounds are left. This constitutes the assay sample, which is ground down, sieved so as to eliminate the coarse gold. The coarse gold is estimated on this sample. The fine material is further sampled down, and the gold or other metals determined by assay, and from this the value of the whole parcel is estimated. The sulphide ore is all crushed, and this, with the concentrates, is roasted in reverberatory furnaces. The object of these is twofold: first, for the expulsion of the arsenic and sulphur; and, second, for the sintering of the finely divided material making it more suitable for blast furnace work.

The reverberatory furnaces are hand rabbled, and are of the following dimensions:—External width, 16 feet; length, 64 feet; height of arch at spring, 18 inches; at crown, 30 inches above hearth, the height diminishing away from the fireplace to the extreme end. The fireplace is divided by a partition parallel to the length of the furnace, each compartment being 5 feet by 2 feet. There are six of these furnaces, each having a daily capacity of seven tons. I am not at all in accord with authorities who recommend such designs. A simpler, more effective, and more suitable roasting furnace would do more work at less cost.

There are two smelting furnaces, one 100 by 36 inches at the tuyeres, of which there are six on each side, with wrought iron jacket the full length of the furnace. This was made by Walkers, Limited, Maryborough (Q.). This has a capacity of 80 tons per day. The other furnace is circular, 36 inches internal diameter, and has a capacity of 30 tons per day. About 30 per cent. of the total tonnage is made up of fuel and fluxes. The blast is supplied by two No. 5 Root's blowers, and the pressure used is 7-10 inches of mercury.

As is well known to smelters, but not so well to the ordinary gold milling man, the working of the furnaces used for smelting lead ores depends on the following principles:—First, that the whole of the quartz or other siliceous material must be got rid of as a fusible slag by causing it to combine with oxides of iron or manganese, generally also with lime as well. Second, the lead compounds must be reduced to the metallic state in the furnace, and dissolve the whole of the gold and silver present. Third, that the sulphur present in the ore should be just sufficient to collect all the copper and a portion of the iron as a matte. The slag, the matte, and the lead will not dissolve in each other or unite with each other to any appreciable extent, and therefore may be separated in virtue of their relative weights or density. While the ideal state of things is simple enough, in practice a great many changes go on, which lead to difficulties. For instance, part of the copper is reduced to the metallic state, and this passes into the lead. Antimony, arsenic, and tin also are reduced, and these pass into the lead, giving a very impure metal. Part of them also pass into the matte, making it a very complex substance. The zinc present in the ore gives trouble, since it is partly reduced to metal, is volatilised in the furnace, and is again oxidised. The oxide by itself being infusible clings to the walls, where it forms accretions. In other ways also it is highly objectionable unless passed into the slag as a silicate. If too high a temperature is attained, or if an excess or deficiency of lime is used, metallic iron tends to separate out. This is objectionable in causing the formation of iron bears, or infusible masses, low down in the furnace itself. It will be seen that the successful working of these furnaces depends on the exact knowledge of the composition of the material fed into the furnace, and a proper mixing of the same, so as to produce the necessary quantity of each ingrdient, the temperature attained, and the volume of air blown in. At Aldershot, probably the most complex ores in Australia are dealt with successfully.

Part of the lead sulphide at Aldershot is converted into silicate by feeding galena in with the ores requiring roasting, and then fritting these before withdrawing from the reverberatories. This



Diagrammatic Plan of the Course of Materials at Aldershot.

is reduced to metallic lead by the joint action of ferrous oxide and carbon, or by ferrous silicate, lime and carbon. Of course, unless the fluxes are properly adjusted silicate of lead may form part of the slag itself. The aim of the metallurgist is to keep the amount to less than 1 per cent. The slags produced at the time of my visit were as follow:—

			Type.		Good Zinc Slag.
Silica (SiO_2)	30 to 35	...	33	...	27.7
Ferrous oxide (FeO)	33 to 36	...	36	...	39
Lime (CaO)	15 to 25	...	21	...	—
Zinc oxide (ZnO)	5 to 11	...	—	...	—

The lead bullion runs from 20 to 50oz. of gold, and from 20 to 150 oz. of silver per ton. The charge is 9.25cwt., coke 1cwt. It is arranged to carry from 10 to 14 per cent. of lead, and sulphur in sufficient quantity to form a low-grade matte, the matte produced running about:—Copper, 15 per cent.; lead, 25 per cent.; iron, 31 per cent.; the balance being mainly sulphur.

The base lead bullion is tapped off periodically from the usual syphon-connected well; the matte collects in a special pot, the slag overflowing to another which is wheeled over the dump. The matte is re-crushed, re-roasted, after which it is fluxed and run down in the smaller blast furnace, the product this time being hard lead and a matte rich in copper. This matte used to be shipped away, but designs are now being got out for installing converters so as to produce blister copper on the spot. The hard lead obtained from the last operation, as well as that from the larger furnace, passes into the softening furnaces. It contains copper arsenic, antimony, iron, bismuth, sulphur, and, in this case, tin as well as silver and gold. All these elements, with the exception of gold, silver, bismuth and copper, are more readily oxidised than lead when subjected to a high temperature. Copper forms an alloy with lead when melted at a low temperature, and floats on the bath of molten lead. By mixing ashes with this alloy the whole of it may be removed as a scum from the surface. An original plan was adopted at the works. By mixing pure galena with the fluid hard lead the copper formed a sulphide, and in that way was removed.

The softening furnace is a water-jacketed reverberatory, the pan being 18 inches deep, and capable of holding 30 tons base bullion. After the bulk of the copper has been removed air is allowed to play over the heated lead, and tin is oxidised and removed as a scum. Tin is a troublesome metal to get, for it robs the lead bullion of part of its gold. Then arsenic and antimony become oxidised, and, combining with the oxide of lead which also forms, are removed as arseniate and antimoniate of lead respectively. The furnace is allowed to cool slightly, so that these salts may be removed as a semi-solid scum. Finally, only litharge forms on the surface of the lead, and when this takes place only gold, silver, bismuth, and a small amount of copper and other metals are present in the lead. The molten metal is then run into zincing kettles. These are large hemispherical cast iron pots, built in brickwork, and heated from below. The size of those at Aldershot are 8 feet 10 inches on top, 3 feet 3 inches deep, and having a capacity of 30 tons. After the lead has become heated up sufficiently molten zinc is poured in, and stirred round with a paddle. The bath is then allowed to cool slowly, and the zinc containing silver, copper, and gold collect and solidify on the surface. These are removed with a perforated ladle. The amount of zinc added depends on the assay value of the lead. The metals are removed fractionally; i.e., enough zinc is added the first time to remove the whole of the gold, copper, and a portion of the silver. When the bath is cooled down the zinc, with these metals alloyed, rise through the lead, and float on the surface, where they solidify. They are removed with the ladles, the liquid lead being allowed to drain back. The balance of the silver is obtained after a second or even a third zincing. By withdrawing samples and assaying them, the progress of the work can be readily followed. Lead poor in silver takes more zinc proportionately than rich, since it takes a certain amount of zinc to saturate the lead, although the gold and copper may be removed even before the lead is saturated. The rich zinc-silver-gold bullion is kept apart from the rich zinc-silver bullion.

The lead, deprived of its gold and silver, is heated, and the zinc oxide removed as well as some litharge, the pure lead going to market.

The zinc bullion is introduced into Faber du Faur retorting furnaces, and the zinc distilled out to be used over again. Lead, with silver and gold, is left behind. These retorts have been described in connection with the melting of large quantities of cyanide bullion in Western Autsralia. It takes about from 7½ to 8 hours to distil off about 3½cwt. of zinc crust. After the zinc has ceased to come off, the lead alloy is poured out by tilting the retort. This is then placed on the hearth of an English cupelling furnace. This is simply a short reverberatory furnace with a movable hearth. The test or frame of the movable hearth is made of iron. This rests on a carriage, and may be run into place in a few minutes. The test is filled with a mixture of cement, limestone and clay. The bullion is placed upon it, and worked up to 50 per cent. The litharge which forms is run into a separate receptacle. The 50 per cent. bullion is finished off in another smaller cupel, and the resulting dore bullion shipped away without further refining.

The plant is also fitted with a 5-foot Huntingdon mill, provided with amalgamated copper plates, a Wilfley table, and two Frue vanners, as well as a Berdan pan. A cyanide plant further enables the company to test all classes of ore, and give advice as to treatment.

The power is supplied by a Lancashire boiler, 26 feet by 7 feet 6 inches, working at 150lb. pressure per square inch. A semi-portable 18 horse-power boiler and engine and two vertical 8 horse-power engines with 9-inch cylinders.

The installation of an electric lighting and power plant is being proceeded with. There are well-equipped engineering and fitting workshops, as well as assay offices, provided with all necessary appliances. Fluxes are obtained near the mine.

The head offices of the company are at Dashwood House, New Broad-street, London. Mr. Arthur Keft is managing director, and the local management is in the hands of Mr. Eric E. Watson, M. Inst. M.M., F.I.C.

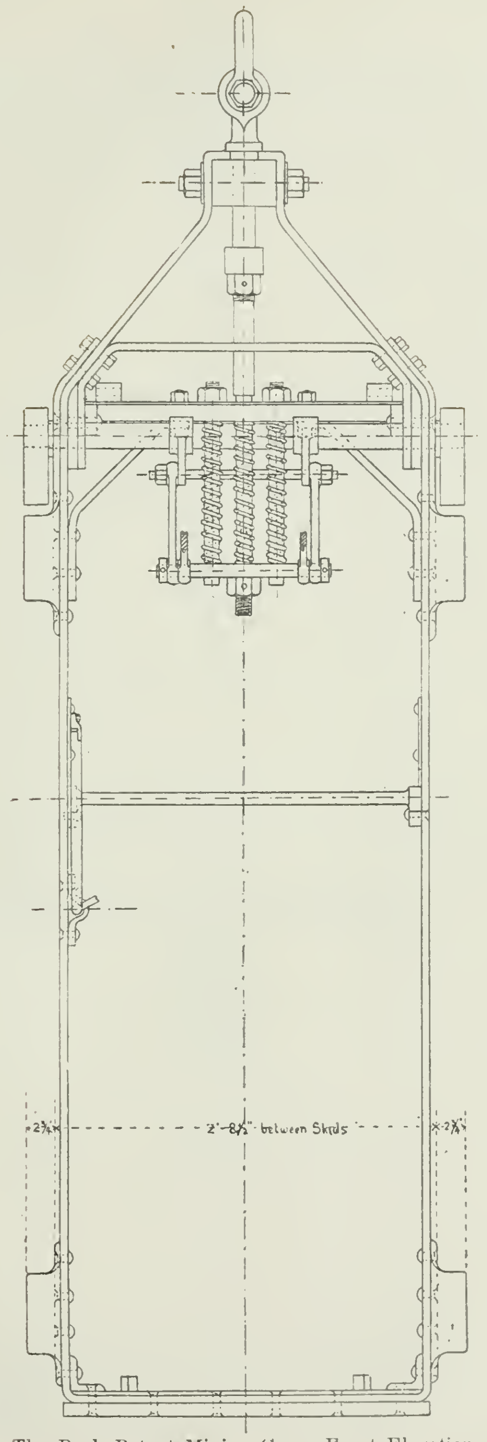
Since my visit to these works Vezin automatic samplers have been introduced, and the whole of the sampling is done automatically, with the exception of wet slimes and cyanide slags.

Gympie.

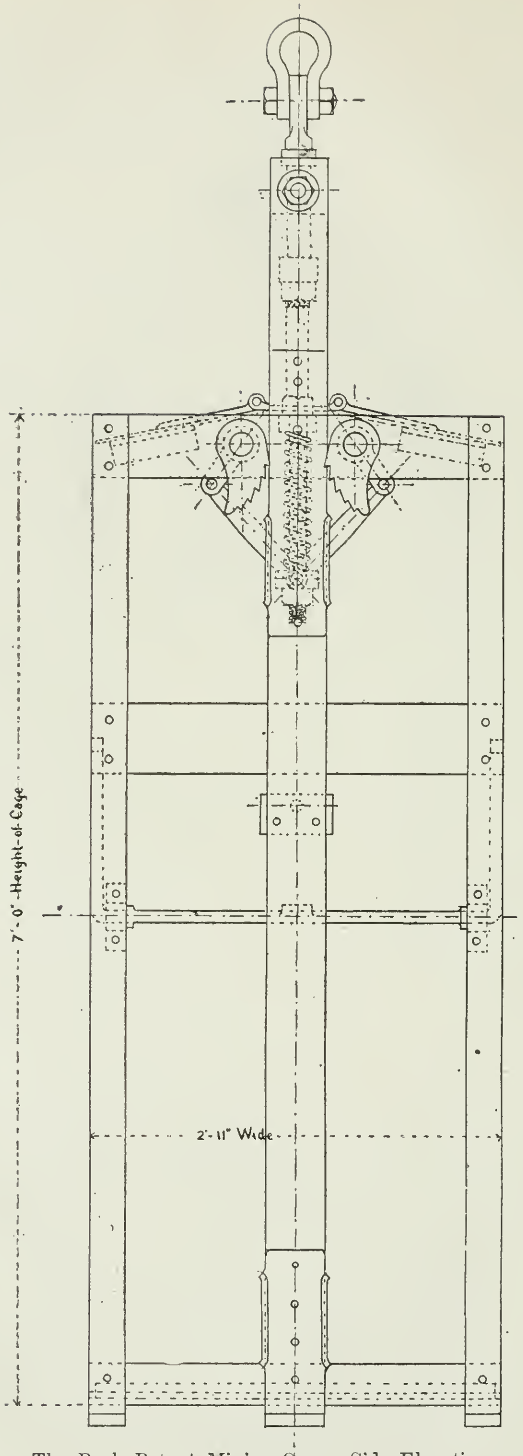
Gympie might be termed the Ballarat of Queensland, but as far as regards natural scenery, the northern town has the advantage. This goldfield differs from those visited in North Queensland, for instead of the sparse and scattered stunted gums, one views a fine forest on the hills along the line between Maryborough and Gympie.

It is almost needless to repeat the oft-told tale of the richness of the Gympie reefs when they intersect the black slates, and their poverty everywhere else: it is accepted as a general maxim on the field, as are the indicators at Ballarat. The material crushed consists largely of a black slate permeated by seams of white quartz, very similar to the spurry country of Bendigo, except that the slate is softer. The gold as a rule is in the quartz, and magnificent specimens are often obtained. Mining at Gympie, on the whole, is in a depressed condition, the Scottish Gympie and a few other mines being notable exceptions. A deep sinking craze has come over the mining community, and instead of letting one or two main shafts go down to prove the lower slate formations, a number are at work, and the result is that calls for this work amount to the same figures as dividends from the productive mines. Fortunately for the Gympie investor, the main trouble seems to be to find the gold in payable quantities. The trouble of extracting it is reduced to a simple mechanical problem only, so that a short description of the methods in vogue at one of the most important mines on the field will serve to show the simple nature of the ore. The Sco tish Gympie in 1901 was the leading mine in the place, and supplied enough ore to keep 100-head of stampers at work continuously. The formation is being worked for a width of 150 feet, and the acting manager, Mr. Affleck, informed me that it had been proved for more than three times that thickness. The ore is removed in part, and pillars are left to support the workings. A system will have to be devised so that ultimately the whole of the material may be removed.

Water does not appear to be abundant. although in many cases many of the mines on the field are inundated by floods, which rise to a height which never would be seriously taken were not the high water-mark left in a clear and unmistakable manner on the chimney stacks. The ore is raised in a double-deck cage, two trucks coming up each time, the load being 2 tons 12cwt., and the speed 1500 feet per minute, the diameter of the rope being 1⅛ inches. The cage is known as Bush's patent, and, being a local invention, a drawing of it is given. Walkers Limited, of Maryborough (Q.), hold the patent right, and, since it has been adopted almost universally on this field, this is sufficient proof of the esteem in which it is held by mining men. It has been claimed for it that it will stand the most severe test, and will not slip and tear the skids from top to bottom, as often happens with other safety cages when they get sufficient momentum up to overcome the slight resistance due to a superficial grip. The ore, after being raised to the poppet head landing, is broken by two stone-breakers placed up



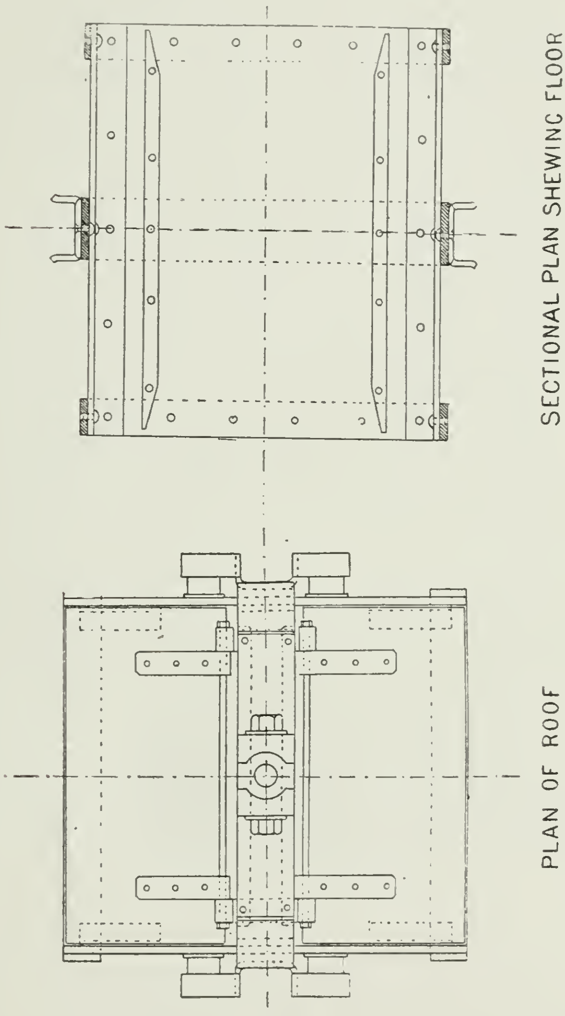
The Bush Patent Mining Cage—Front Elevation.



The Bush Patent Mining Cage—Side Elevation.

aloft; each of these breaks 30 tons per hour. The stone-breakers were originally placed behind the stampers and over the feeding hopper. This arrangement gave rise to a great deal of vibration, and led to the removal of the breakers.

The broken ore is then trucked into hoppers, and it gravitates



The Bush Patent Mining Cage.

thence into a feed box and falls into the stamper boxes. The feed box is of the simplest kind; a sliding shutter at the upper end admits the ore in the desired quantity; it slides down an inclined rectangular shoot, open on top and having a dividing board across not quite reaching to the bottom, near the lower end. Any run of stone is checked by this lower baffle board, and the stone trickles under it, dropping down into the box, aided by gravity and the vibration of the stampers alone. It is claimed that this arrangement works perfectly. There is not much to be learnt about the arrangement of the battery, which consists of 100 head of stamps in five-head boxes. These are arranged in line. The cam shaft is divided in the centre, and 50 heads work on the right and 50 on the left of the dividing platform. An engine is coupled direct on to crank attached to the end of the cam shaft, and the whole 50 head worked without intermediate gearing of any kind. The arrangement does not seem to have much to recommend it, for the thrust and vibration is transmitted direct to the engine without any appliance to cushion it, and no set of stampers can be thrown out of gear without lifting each stamp clear of the cam. Each stamper weighs 900lb., and delivers 65 blows per minute. It was stated that on running 70 blows per minute more amalgam got on to the plates than with the lower rate of speed. Woven screens were tried, but discarded in favor of punched sheet iron, having 196 holes per square inch. Mercury is fed into the boxes in the usual way. It is somewhat surprising to see the divided screw discs or tappets on a modern battery such as this. The pulp falls on to a copper apron, passes through a shallow well, and then over amalgamated copper plates. A Wilfley table was tried at the end of one of the boxes, but although it did its work well the clean pyrites saved only assayed about 13dwt. per ton; these were practically valueless on the field. The present method is to run the stream from fifty head down a launder, 3500 feet in length, to the Mary River. Cleats of wood, 2 inches by 1 inch, are placed across the launders at the upper end; these act as ripples. An amount about equal to 1 per cent. of the ore crushed is saved; this is sold for from 7s. to 10s. per ton. The fall in the launders is 1 in 60. The amount crushed per week is 1000 to 1100 tons. The stone raised averages 14dwt. per ton. There could not be a simpler ore to deal with in the whole world than at Gympie, since the tailings from the end of the amalgamated plates will only yield 1dwt. 10gr. per ton. Cyanide plants and chlorination plants are said to be defunct on the field, and even concentrators have had to go. Beyond mechanical devices, which may be studied with more advantage in Charters Towers, and methods of dealing with large bodies of ore, there is little to be learnt at Gympie. They are indeed a fortunate people to have such a field as this, which is even a simpler metallurgical proposition than is to be found in the goldfields of Ballarat and Bendigo.

Ravenswood and Donnybrook.

If there is one candle which attracted the mining moth more than any other in Queensland it is to be found in Raverswood. This place is easily reached by a branch line from the Charters Towers to Townsville railway. The main rock, as at the Towers, is granite, which is penetrated by dykes of later age. The whole of the surface soil is blackened by crystalline grains of magnetite, no doubt derived from the later igneous rocks.

In the early days of Ravenswood many miners were attracted thither from the southern States on account of the richness of the alluvial diggings. Later on the outcrops of reefs almost innumerable were discovered, and these when worked gave very rich returns. The stone was oxidised and porous, and highly charged with iron oxides, and locally obtained the name "brown-stone." Deeper down the unaltered mineralised veins were struck, and each man with a process rushed to Ravenswood, only to fail as others failed before. The remains of a matting furnace, and a heap of slag with sows strewn over its surface, silently testifies to the failure of that process.

Almost adjoining the old smelting works one sees the remains of the antiquated Newbery-Vautin chlorination barrels, and a furnace, constructed at variance with all the principles of economic and effective roasting. There is little wonder that the men who were expected to work that process also failed.

With the spread of the pan system from Charters Towers over Queensland, Ravenswood was taken under the wing of its overshadowing parent, and altogether ignoring the fact that there was an additional element or two to deal with. The same disappointing results were attained as with the smelting and chlorination processes.

In conversation with a very intelligent battery manager, who was working the battery, pan, and cyanide system, I was specially cautioned against touching a certain area almost within the town of Ravenswood itself, and this was simply because it contained arsenic! What hope could there be for such a field, where arsenical pyrites were looked upon as fatal to gold extraction?

In happy contrast to the failures on this field, one sees at the battery owned by Mr. A. Laurence Wilson, M.I.M.E., a method which shows that he has weighed the methods adopted in this State, and found them wanting. For present purposes he has adopted a system which commends itself as being rational and sound. The ores from the various mines contain more or less quartz and free gold, but, in addition, a small quantity of metallic bismuth and bismuthinite; also a large quantity of galena, arsenical pyrites, iron pyrites, copper pyrites, and zinc blende. The stone containing these minerals is put through a Dodge stonebreaker, 12-inch face, and is fed into a 10-head battery. No mercury is used in the box, nor are amalgamated plates made use of. The crushed material is fed on to two Wilfley tables, and these useful machines show their streaks of concentrates as a Luhrig vanner does. At the back of the table there is a long streak of gold which at once indicates the yield. Next to it is a glistening parallel band of bismuth, then follows a wider band of galena, then arseno-pyrite, another of pyrite, a yellow band of chalco-pyrite, and a black band of zinc blende, finishing up with a broad seam of waste sand almost deprived of its metallic ingredients. A strip of about 3 inches in width is caught on the golden side of the concentrates, and this is ground specially in Berdan basins, the water in the Berdans not being allowed to overflow. The amalgam obtained from the Sunset mine, the ore from which was being treated, retorted one-sixth, but the retorted mass would have puzzled many a mining manager. Analysis of a sample supplied showed it to contain from 91 per cent. to 93 per cent. of bismuth, and from 7 per cent. to 9 per cent. of gold and silver.

A cupel furnace for refining this bullion was erected at these works. It was of the usual English type with a movable hearth, consisting of a cast iron tray into which bone ash was packed. This was heated up, and the bullion fed in with lead. As a general rule 1000oz. of bullion would produce 300oz. cupelled metal. The concentrates obtained, independent of the golden streak, were sold to the smelting works, as were also the cupel bottoms. The tailings sand was saved for cyaniding. While this system has proved eminently successful in the hands of such a capabale man as Mr. Wilson, the difficulty with regard to the treatment of ore far removed from a railway cannot be said to have been overcome up to the present. It may cost far more to send concentrates to smelting works than they are worth.

VICTORIA.

TREATMENT OF AURIFEROUS ORES.

Bethanga.

Bethanga is a small town fourteen miles to the east of Wodonga, and but a short distance from the Bethanga-road railway station. The country for some miles along the Murray is composed of rounded hills of granite, giving a fair grazing soil and fine rolling country. Near Bethanga several lodes have been opened up, which, like most veins in granite, are small and heavily mineralised, but are said to be persistent in strike for great lengths. The upper portion, or oxidised stone, was treated in 1876 by several parties with ordinary batteries, there being eight of these on the field. About 50 feet down the mineralised ores were struck, and the stone yielded no gold to these; then concentration and amalgamation, somewhat on similar lines to that practised now on Charters Towers, was introduced, but the results were very bad. Then patent pans were introduced for grinding the raw material and amalgamating it; these gave the same poor extractions as they had given before or have given since. Then roasting, followed by grinding in pans; the results were such that often the whole pan would be coated with the copper precipitated from the solution, and the ultimate fate of that process was sealed by the low extractions obtained. Samples were sent all over the world for experimental treatment, but, although a hundred schemes were suggested, they were either palpably unsuitable or deficient in vital essentials. One method that seemed to promise better than others was introduced by the Hon. J. A. Wallace. This was the ordinary smelting for matte. The analyses of the raw ore was as follows:—Silica (SiO_2), 30.4; alumina (Al_2O_3), .8; iron (Fe), 27.4; copper (Cu), 1.85; antimony (Sb), .25; zinc (Zn), .20; arsenic (As), 12.30; sulphur (S), 19.5; magnesia (MgO), .2; oxygen and loss, 7.1. The average amount of sulphur is about 24 per cent., and of arsenic 7.5 per cent., while gold runs at the rate of 1oz. 12dwt. per ton, the minerals present being pyrite (FeS_2) and pyrrhotite (Fe_7S_8) arsenopyrite (FeAsS), chalcopyrite ($Fe_2S_3Cu_2S$), and small quantities of blende (ZnS), galena (PbS), and calcite ($CaCO_3$), the gangue being quartz or silica (SiO_2). Galena and blende sometimes amount to several per cent., but, as a rule, they are only present in small quantities or in blotches. The ore was roasted in the usual way, and the roasted material suitably fluxed with limestone; the copper matte produced, containing the bulk of the copper and almost the whole of the gold and silver, was shipped to England. After this Mr. Wallace introduced a blast furnace, which ran for three or four years. Holloway's process was tried, but did not prove successful owing to the wear and tear on the converters. Since the coke and limestone for smelting had to be brought from Melbourne, it could not be expected that this method of treatment would long survive. A curious instance of the freedom allowed to the sanguine inventor by Mr. Wallace, and one which indicated the dawn of a process which is now eminently successful, is afforded by the work of the late Dr. Wanderlich. This gentleman was so convinced that he had discovered a perfect electrolytic process that he had the matte, which had been sent to England, shipped back to Bethanga for treatment. He got it re-smelted into cakes about 2 inches thick and 20 inches square. These were placed in tanks in a solution of sulphate of iron and a current of electricity passed. The solid material remaining after the reaction was smelted in ordinary crucibles. After spending about £2600 and working for some time the final day of cleaning up arrived. The expectant directors waited for their cake of gold—for the matte used to run 56 per cent. of copper and 32oz. per ton of gold—from this parcel. The doctor came to his final smelt, poured his cake and turned it over. His exclammation, "Got dam," and his hurried departure left the onlookers in no doubt as to this process being before its time.

During this period heap roasting followed by kiln roasting was practised. From Mr. Martin, the works manager, who courteously supplied me with all particulars, I learnt that heap or stall roasting, followed by roasting in reverberatory furnaces, is no cheaper than dealing with the crushed ore in the latter form of furnaces straight away.

Owing to the expense in smelting, and the heavy charges against the matte and its contained metals, another change of treatment became necessary. The cyanide people and bromo cyanide experts tackled the raw ore with their processes, only to meet with failure. It would be waste of time to enumerate the processes tried in contradiction to all physical and chemical principles. The marvel is they were allowed on the works at all.

The first glimmer of hope of successful treatment was raised by a system of chlorination introduced. The vats were connected in series, with the idea of allowing the solutions to become enriched in passing from one vat to the other. Probably the possibility of having a badly roasted lot in one vat never was considered, for only about 50 to 60 per cent. of the gold was extracted. Mr. French, who came from Glasgow, then introduced the Plattner system in small vats, and was fairly successful. In 1888 Mr. French went home to float a mine, and during his absence Mr. Thos. Martin, jun., looked after the works. By careful work he found he could get very good extractions. During this time the roasting furnaces were badly constructed—like nearly all those used on smelting works, where fuel seems to be a secondary object; the firebars were set low, the distance between the floor and the crown of the arch high, and the arch was sharply curved; all these led to excessive consumption of firewood and bad roasting. This was remedied, and now the company has typical furnaces for treatment by hand rabbling of such complex ores. In 1895 Mr. W. S. Hazleton had charge of the works, and was working the Plattner process successfully. Mr. Martin tested the modified Munktell process—that is, a solution of chloride of lime and dilute sulphuric acid in molecular proportions—against the Plattner system. It was found that the tailings from the Plattner process gave from 3dwt. to 5dwt. per ton: those from the Munckteli process gave from 2dwt. to 4dwt. Mr. Hazleton left shortly afterwards, and since then Mr. Martin has had full control.

The works are well equipped, being provided with Galloway tube boilers working to 100lb. pressure, a compound engine with 12½ and 22 inch cylinders and 3 feet stroke, giving 100 h.p.: one of Parker's dynamos running 970 revs., and supplying 80 incandescent lamps and four arc lights. The ore as delivered at the works is weighed, every 10 tons to within 7lb.: a sample is taken out of each truck and the moisture deducted and its assay value determined. Since no free gold can be seen in the stone, nor can any be shown by panning off in a dish, it is probable that the gold is excessively fine or that it exists in a combined state in the ore, perhaps as a telluride. The stone is passed through a breaker with a reciprocating jaw 14 inches wide, 30cwt. being crushed by this down to small size in nine minutes. The small stone is then fed into a Krupp mill, which crushes the stone to a fine size at the rate of two tons per hour. This mill is No. 5, and cost £670 or £1000 erected. Ninety-five steel balls, weighing 20lb. each, are fed in with the ore, and as they wear down they are replaced. The wear is at the rate of ¾lb. per ton of ore crushed. Repairs are only required once in six months.

The crushed ore is fed direct into reverberatory furnaces, of which there are six, placed side by side, all worked by hand labor. These furnaces, or those which have been erected in other places on similar lines to these, are about the only hand rabbling furnaces worth describing, and they differ in a marked manner from the cumbrous and clumsy ones seen on other fields.

The dimensions over all are 56 feet long by 10 feet wide. For the first 16 feet at the firebox end the furnace is flat; for the next 27 feet there is a gentle rise to facilitate the working down of the ore: while for the last 13 feet, where the ore has a tendency to run when being rabbled, the floor is flat. The fireplace is 3 feet wide, and extends to the arched roof above, the height of the firebars being 3 feet above the ground and 16 inches below the bridge. The bridge is about 2 feet wide, but is not raised above the level of the hearth. A chamber, 3 feet deep and 1 foot 6 inches wide, extends across the lower end of the hearth; this serves as a storage place for hot ore before being withdrawn from the furnace. This chamber is open on the opposite side to that used for introducing fuel, and thus any ashes or charcoal from the ash pit does not get into the ore. The arch has a slight downward sweep from over the fireplace to the upper end of the ore chamber mentiond above, thus causing the flame to be deflected when passing over that part. The arch at is highest point at this place is only 10 inches above the floor; this is continued until the upper flat end is reached, when a rise of 15 inches is given. The internal width of the hearth is 7 feet 6 inches. The arch has a rise of 3 inches from the skewback to the centre. The skewback consists of a railway rail with head flush with the outside brickwork, and the bottom being set to spring the bricks from. The arch is built on sand and is nine inches thick, being made with bricks on end, breaking joint, the upper edges being filled with chips of stone and grouted well with a mixture of sand and clay. There are 12 strapping irons on each side, and 10 rabelling doors placed alternately on each side. Four air-holes 2 inches by 3 inches are placed at the end of the furnace, to supply an additional quantity of air through the fireplace, if required; also two large doors 12 inches by 9 inches, which also serve to admit a large volume of cold air, should this be necessary. The furnace is built with ordinary brick throughout, a well-tempered mixture of clay and sand being used for setting all parts exposed to heat. This form of furnace is cheap, durable, and is capable of doing better work at lower cost than many more pretentious ones. The small amount of iron used serves to keep it together without pulling it to pieces, as is so often done in furnaces overloaded with straps and stays. The rise at the end is an innovation which will be appreciated by those roasting heavy sulphides, the trouble being to keep the heat down at this end. In the ordinary furnace this is almost impossible without cooling the lower part down also, which is undesirable. The amount of dust formed is also reduced since the volume of heated air, in arriving at this point, has a greater sectional area to escape through, hence its velocity is diminished and oxidation goes on at a reasonable rate.

The ore is roasted in the usual way, but when it gets to the lower hearth it is tested in the manner recommended by Kustel. A bright iron rod is taken; a sample of the ore is withdrawn and placed in a stamped out sheet iron pan; a spoonful of this is emptied into an enamelled basin; this is stood on the hot iron, when it will boil in a few minutes; the rod is introduced and kept in for about a minute; if discoloration shows the ore is not sufficiently roasted—if the ore remains clean then the roasting is looked upon as being perfectly done. The test, of course, though it may serve for ores in which copper is a constituent, would not serve on others where it was not, sulphate of iron and other substances inimical to chlorine not responding to it. However, as soon as this stage is reached, salt to the extent of 12lb. to 4cwt. of ore is fed into the furnace, mixed well with the ore, the whole lot then worked into the receptacle for it at the lower end of the furnace, and subsequently withdrawn. Mr. Martin considered the salt necessary to chloridise the silver present and alloyed with the gold; but I do not think this is the case, since the silver present in gold could not be attacked in such a short time without the gold being volatilised; and on testing ores after similar treatment with hyposulphite of soda no silver was dissolved out. Probably the main effect is oxidising some material unacted upon by the furnace gases and chloridising portion of the copper. Chlorine and hydrochloric acid are given off freely after a certain temperature is reached. Mr. Martin does not consider that he loses any gold by volatilisation. The ore after having been damped is fed into the sand vats, of which there are four capable of holding 16 tons of roasted ore each, five holding 7 tons, and two with a capacity of 12 tons 16cwt. each. All these vats are shallow, only carrying 2 feet 10 inches of ore down to the filter beds. There are four solution vats, two holding 700 gallons and two 1100 gallons each.

The solvent solution used successfully for several months is that recommended by Dr. Black, Professor of Chemistry at the Otago

University (N.Z.), as a modification of the Etard process. The solution for the 1100-gallon vats is made up with 110lb. suphuric acid, 90lb. salt, 11lb. permanganate of potash; for the 700-gallon vats, 70lb. sulphuric acid, 60lb. salt, and 7lb. permanganate. This solution does not smell of chlorine, and has the advantage that the instant it is destroyed the operator can tell by the loss of the rich red or pink color. I must confess that when it was first mentioned, it appeared to offer no advantages over the ordinary methods of dilute chlorine solution, but here in actual operation it has been tested against these solutions, and the verdict is unhesitatingly given in its favor. Mr. Martin says it is easily made up, its reactions are obvious, nothing objectionable is introduced into your filter beds to clog up the filter bottoms like calcium sulphate will, and, above all, it does not kill those having to work with it by its smell. The time for extraction of the gold with Dr. Black's solution has been found to be on an average 130 hours, but if the gold is all fine then 50 hours are sufficient. Samples from the vats are panned off from time to time to see if all the gold is dissolved, the washings from the vats being tested with sulphate of iron to see if gold is present in solution. The great stack of tailings are said to be reduced to 3dwt. and less per ton, which is an excellent result for this class of ore, more than 500 tons per month being treated.

The solutions are led away into vats for precipitation, of which there are 10, having a capacity of 700gal. each. Sulphur dioxide is generated in cast-iron retorts from sulphuric acid and charcoal. The gases from the retort, consisting of carbon dioxide and sulphur dioxide, are passed into the precipitating vats through pipes for an hour or more until all the gold is thrown out of solution. It is stated that if the gas is passed too long the gold will pass right through the series of filters and escape. The gold comes down in a very fine state of division, and no attempt is made to settle it; on the other hand, as soon as it is found to be all in the metallic state the liquor suspending it is allowed to flow through a series of 24 jars, about 2 feet deep by 2 feet in diameter, filled with sawdust, thence through a series of 24 charcoal filters of the same dimensions. Very little gold passes the first few sawdust filters. The solution in which sulphate of lime is present, as in the Munktell process, will be found to keep the lime in solution when SO_2 is passed through it, whereas by other methods of precipitation a considerable precipitation takes place. When the gold has been caught by the sawdust filters the solution flows into a cemented tank 36 feet in diameter by 5 feet 6 inches deep; the tank contains some tons of scrap iron. Steam is blown in, which raises the liquor to near boiling point, causes a circulation of liquor, and assists materially in the precipitation of the copper. The solutions leave the tanks in from 48 to 72 hours deprived of their copper. The amount recovered in five months was 25 tons of precipitate, which would usually run 1½oz. gold per ton, 55 per cent. of copper. This was purchased on the ground by Messrs. Lempriere for £35 per ton.

The copper present in the roasted ore is about 1.7 to 1.8, and in the residues 1.2 to .5 per cent. A large quantity of arsenic is saved from the flues—about 50 tons per month; this is sold on the spot for from 10s. to 12s. per ton. It generally runs about 7dwt. gold per ton. A diagrammatic plan of the operations is given below.

Mr. Paul Martin, the assayer and chemist of the works, courteously supplied me with all technical information sought. The ore sample, after having been broken and crushed in one of Jaques Bros. combined laboratory stone-breakers and rolls, is sampled

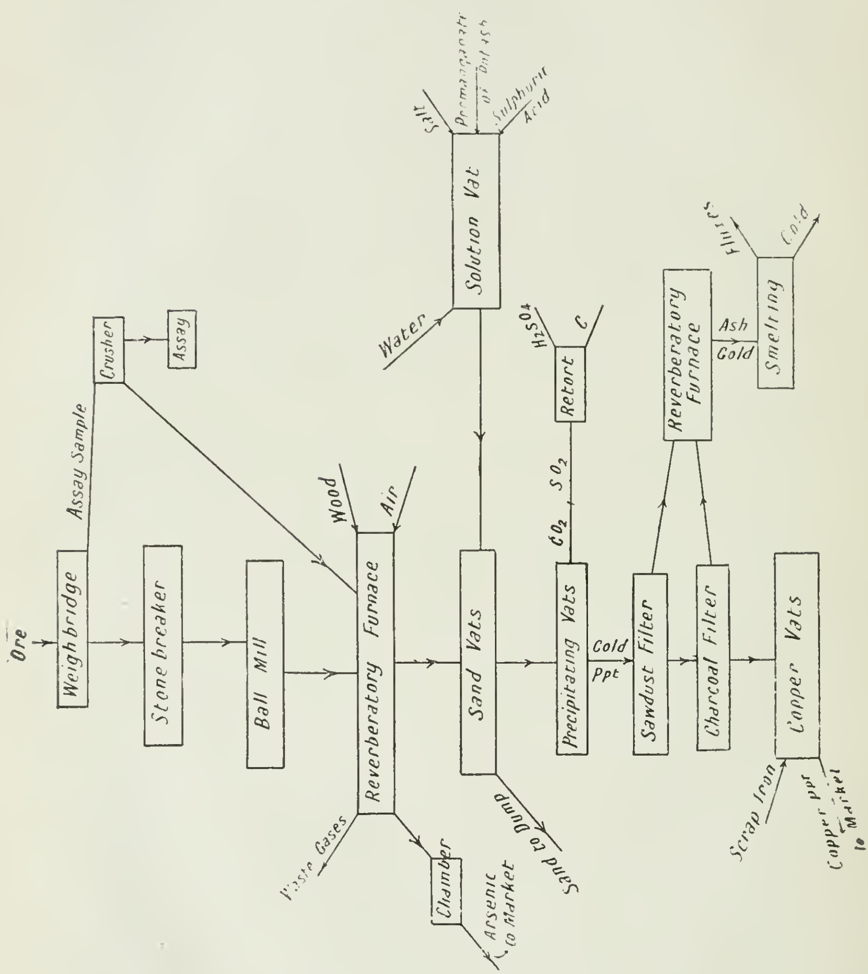


Diagram of Plant Arrangement.

down and assayed—500 grain samples are taken, roasted and fluxed with 500 litharge, 70 argol, 350 pulverised borax, 600 bicarbonate of soda. The buttons are cupelled straight away. Tailings assays are made on 1000-grain samples. Copper is estimated by the cyanide process; arsenic and sulphur gravimetrically. Sulphuric acid is estimated by a normal alkaline solution, and it is interesting to note that the strength of the acid is reduced from 1.2 to 1.5 per cent. to .7 to .8 per cent. after percolating through the ore. The cost of wood is 7s 6d. per ton, or, since 2 tons 5cwt. make a cord, about 17s. per cord. The roasting furnaces each consume 160lb. of wood per hour, and put through 20 tons in 24 hours, or the consumption of fuel is nearly equal to one-half the weight of the ore roasted. The furnace-men are paid 7s. per shift, and each man puts through his ton per shift. Thus the cost for those items amounts to about 11s. per ton, which is exceedingly good for hand-labor reverberatories on this class of ores. The cost of chlorinating amounts to about another 11s. per ton, so that the problem of treating the material may be said to be an event of the past.

Cassilis District, Gippsland.

For more than a quarter of a century the reefs in the neighborhood of Omeo have been worked in a more or less energetic manner, and though they are for the most part small, yet they are so well-defined and so rich as to give profitable employment to a great many miners. In the early history of the field batteries were put up in what was regarded as the most suitable sites. These sites, it is hardly necessary to state, were on running streams, so that the tailings might be got rid of. The amount of gold which has gone down those streams can only be conjectured.

In Cassilis, as in other places, the same evolution as regards treatment of ores has been gone through, and the present method as practised has led to a great many reefs being worked that would otherwise have been abandoned. While dealing with the upper portion of the reefs the gold was free and present in what is known as brown or oxidised ore. On the field itself the absurd name lava has been given to it, probably from a fancied resemblance to the decomposed dykes on the Bendigo goldfields. Some of this surface ore was very rich, part of the enrichment being accounted for by change of weight due to oxidation. The reefs themselves are very nearly vertical, and exist either in a quartz diorite or schist, a reef very often cutting through both classes of rock. The metallic minerals present occur, according to quantity, in the following order: — Arsenopyrite ($FeAsS$), pyrites (FeS_2), blende (ZnS), galena (PbS), chalcopyrite ($Cu_2SFe_2S_3$), small quantities of stibnite (Sb_2S_3), bismuthinite (Bi_2S_3). The gangue accompanying these minerals is mainly quartz. The reefs carry from 10 to 60 per cent. of these heavy sulphides and arsenides, and are as complex in composition as any in Australia. When the gold was free only the ordinary stamper battery was used, blanket tables being sometimes added. These blanketings were then either fed through the battery box again or rudely concentrated in tyes by running them down a launder, with or without ripples on the bottom, while a boy with a stiff broom kept sweeping the sand up against the running stream—a method similar to that practised by the Chinese in the Straits Settlements when concentrating tin ore. A rich product was obtained, which would sometimes go 25oz. per ton, but at the same time about 50 per cent. of the gold was lost in getting it.

The second stage of development was arrived at when grinding and amalgamating pans were introduced. Whenever a public crushing works was destitute of pans, then the owners of stone refused to send them ore. As a result pans were introduced almost everywhere, and the increased yields were hailed with such satisfaction that the millenium of treatment was considered to be reached. A hundred ounces from the batteries and four hundred from the pans was sufficient proof of the value of these appliances. Buyers of ore from Germany and other places came along and paid over £6 per ton on the ground for all the sand left after the pans were done with. This sand had to be bagged, carted for sixty miles over a rough road, shipped, and transhipped to Germany. Tailings which had not gone down the creek were assayed, and their value was such that an attempt at what is now known as the Warden Works was made to save this gold. Messrs. McCulloch and Eckberg purchased the plant and tailings. They introduced a number of Frue vanners to concentrate pyrites and gold. A reverberatory furnace was built on exactly similar lines to those in use at smelting works. A couple of barrels, holding about 2 tons each, were made use of for chlorinating in; filter beds and charcoal precipitating jars were introduced. After working for a short time and treating only a small portion of the heap the whole plant was sold to the Warden G.M. Co. Mr. Hazleton, who had been successful at Bethanga, was placed in charge. He found the plant so badly designed that he pulled down the furnace, and erected one more suitable for economic working, and, instead of the barrel system, he introduced the modified Munktell process. Mr. Walter Radford was in charge of the works at the time of my visit, and he supplied me with many details given in this article. The plant consisted of a 10-head battery, 2 Halley tables, 6 Frue and Triumph vanners, Berdan pan, roasting furnace, chlorination vats, and a 160-ton cyanide plant. The stone as broken out of the mines is tipped into a hopper in small pieces, whence it gravitates into a rather crude looking automatic feeder, which, however, works well with hard, evenly broken stone. It consists of a long shoot hung at an angle of 35deg. to the horizontal. Its lower end is suspended from two vertical rods, which pass through a beam about 3 feet above the shoot. Over the ends of each of the suspending rods, which project through the beam, a spiral spring a few inches long is slipped. A nut is screwed on over the top of this, which rests on the top of the spiral. The pitch of the shoot may thus be raised or lowered by means of these nuts. A vertical rod attached to the end of the shoot, kept in position by passing through a ring above, has a sliding piece of steel projecting under the tappet, which may be adjusted on the rod at any desired height, so that before the stamper is on the false bottom below it will be struck by the descending tappet. The blow is transmitted to the feeder, the jar being absorbed by the spiral springs on which it is hung, while the stone is shaken into the box, and the beating of metal on metal prevented.

The battery was run at 70 blows per minute or sometimes less, since the amount of heavy metallic minerals was so great as to require a large quantity of water to keep the plates clean. The plates were arranged in the following order:—No. 1: 18 inches wide; fall $\frac{3}{4}$ inch to 1 foot. No. 2: 9 inches wide, with same fall as No. 1. Nos. 3, 4, and 5: Each 3 feet, with a fall of $1\frac{1}{4}$ inches per foot. Wells were placed between Nos. 1 and 2 plates and also between Nos. 2 and 3, there being a drop of 7 inches into No. 1, and 3 inches to No. 2 well. The wells were made semicircular in section, so as to avoid the collection of heavy sand in those having a rectangular section. There was a drop from each plate to the succeeding ones.

The Halley tables had a fall of $\frac{3}{8}$-inch per foot. The blanket tables were 13 feet long, made with five steps, each having a fall of $1\frac{1}{2}$ inch per foot, and a drop of 3 inches to the next step. It was found that the vanners gave such a poor product after the Halley and blanket tables that they were seldom used. The battery was used on all classes of ore, the main work being public crushing. The average returns from surface ore would be about 50oz. amalgam from the boxes, 40oz. from the plates, and 10oz. from the wells, the amalgam retorting about 40 per cent. The mineralised ore would give from a parcel of 100 tons the following yields:—Boxes —no mercury being fed in—15oz. gold; wells, 10oz. gold: amalgam from plates, 100oz., giving 25 per cent. of gold; in all 50oz. gold. The Halley table concentrates would weigh 30 tons, giving 180oz. gold; the blankets 5 tons, giving 10oz. gold. The tailings and slimes were saved for cyaniding; these would run from 4dwt. to 5dwt. per ton. The ore was not crushed too finely, punched gratings having 225 holes per square inch being used. The wells were cleaned out every four hours, and the plates dressed. The skimmings and other small quantities of rich amalgams were treated in a Berdan pan, and the overflowing concentrates added to those of the Halley for chlorination.

The furnace used by the company is of the same type as used at Bethanga, and is 46 feet by 10 feet over all. The mineralised sand is fed in at the top of the furnace, where the bulk of the arsenic and sulphur is removed before it is worked over 15 feet of the hearth. The remaining space is mainly useful for the oxidation and conversion of magnetic oxide into the sesquioxide. The heavier the concentrates the cheaper is the roasting, since the burning sulphur supplies a large portion of the heat required. On the last floor a small quantity of salt is added, and the ore is withdrawn a few minutes afterwards, the test applied being the absence of magnetic particles when the cooled sand is acted upon by a horseshoe magnetic. It has been found that in some instances this is not infallible.

The ore is then cooled down and placed in the sand vats, of which there are three, with a capacity of 6 tons each. A water wash is sometimes applied: at other times a sulphuric acid wash. The chlorine solution is then run on. This is prepared by adding chloride of lime to a small barrel placed on a horizontal axis: water is run through, and the solution—or rather milky liquor containing the chloride of lime and other substances—is run into a vat holding 3 tons of liquor. Sulphuric acid slightly in excess of the quantity required to act on the chloride of lime is diluted in a corresponding vat placed by the side of the other one. The two solutions are at the same height. They are then allowed to flow through a pipe which junctions before reaching the sand vat on the next floor, and from the junction the liquors mix, giving chlorine in solution and a precipitate of part of the sulphate of lime. This arrangement was adopted by Munktell, it being considered that the chlorine was nascent, but it is not necessary to show this cannot be the case; at the same time it affords a good method for mixing the solutions without an undue escape of chlorine. Strong solutions are generally used, the amount being present from 0.15 to 0.3 per cent. of chlorine. The time required for the solution of the gold varies from four to ten days, almost wholly depending on the coarseness of that metal. The solutions from the sand vats are run into a settling vat capable of holding four tons. The liquors are allowed to stand in this for about 24 hours, when the calcium sulphate and other salts and substances which have come out of the sand vat will have subsided. The clear liquor is then drawn off into a precipitating vat of the same dimensions as the last, and the gold precipitated with an acid solution of sulphate of iron. The sulphate of iron is made on the works by dissolving scrap iron in dilute sulphuric acid. The precipitate is allowed to settle for 24 hours from solutions rich in gold, and for 36 hours at least for those poor in gold. The precipitated gold is obtained by decanting the clear solution on top and collecting the gold precipitated in a smaller vessel, filtering, washing and drying in the usual way. The dried precipitate runs from 50 to 80 per cent. gold, and the smelted bars will assay from 97 to 99.5 per cent. gold, the impurities being being mainly lead, antimony, arsenic and bismuth in small quantities.

The cyanide plant at the works consists of two mixing and neutralising vats, each 20 feet in diameter and 4 feet 6 inches deep, four sand vats 20 feet in diameter and 3 feet 6 inches deep, one filter vat, two zinc boxes with 10 compartments, each 28 inches by 15 inches and 30 inches deep, two solution vats 20 feet in diameter and 5 feet deep. The sand or tailings were screened and their acidity determined. This amounted to a very high figure for such pyritic material, as much as 35lb. of lime being required in some instances for one ton of tailings. The lime was slacked and the powder added evenly with the tailings in the neutralising vats, it being found more economical to do this than to wash the acid and acid salts out first with water. Water washes were then run on, and after some time the liquor came out with an alkaline reaction. After that considerable quantities of sulphides of arsenic and antimony still came through in the alkaline wash. As soon as these substances were removed the material was emptied into the sand vats; the air acting upon the ferrous and other compounds converted them into more highly oxidised ores, and thus prevented the destruction of the cyanide. The cyanide solutions were then run on, and about a fortnight was required for a 75 per cent. extraction on 12dwt. sand, the consumption of cyanide amounting to 1¾lb. per ton. The solutions were run through the filter vat on to the zinc boxes. Almost the whole of the gold was precipitated in the first three compartments of the zinc boxes. The zinc precipitates gave on smelting from 50 to 60 per cent. bullion, of a value of about £3 per ounce. The heap of tailings treated paid the shareholders handsomely, but it is a matter to be regretted that richer tailings, due to ineffective treatment, should have been sluiced down almost every river and creek in Gippsland. Some hundreds of tons of sand, which had previously been chlorinated, were also treated by cyanide, and these gave excellent extractions with a very small consumption of cyanide.

Messrs. Allsop's metallurgical works are situated at Tongio West, about two miles from Cassilis, and these gentlemen have introduced another method of treatment, which is of the utmost importance in fields where the cost of carriage is high, and where the ores are as refractory as on the field. Since the owners have solved many difficulties in the way of ore treatment, and have to compete against buyers from other places, the whole details of their processes cannot be published, but from an examination of their works and from an inspection of their books, courteously granted by Mr. J. L. Allsop, there is not the slightest doubt about the success of their methods, which are cheaper and give even better returns than those hitherto described. The two brothers are young Victorians, who gained their experience in Johannesburg. On returning to Victoria they were in the employment of the Australian Gold Recovery Co. An elaborate cyanide plant was erected at Tongio West for treating raw tailings by cyanide. The Siemens-Halske method of precipitation on lead foil used as cathodes, while iron plates formed the anodes, was introduced. After having successfully treated the residue left by the floods of a heap of tailings, the works were closed down, it being found that the cyanide solutions would not give a high extraction with the pyritic tailings from the mine with which they had to deal. Messrs. Allsop bought portion of the plant and started work on their own account. A furnace very similar to those used at Bethanga was erected, and arsenic chambers and flues connected to their stack. The ore or concentrates saved are all roasted, and then treated with cyanide in the ordinary way. The cost of roasting amounts to from 13s. to 15s. per ton. A lower temperature is attained when roasting to that given for roasting for chlorinating, and a large amount of sulphuric acid forms and is volatilised. This condenses to a large extent with the arsenic, and causes the latter, when withdrawn from the flues, to become as wet as mud by the absorption of moisture from the air. The ore, after having been roasted, the magnet being used for testing is withdrawn from the furnace and moistened; it is then mixed with a small quantity of slaked lime and water-washed until freely alkaline, after which the liquor is allowed to remain in contact with it all night. Owing to the shrinkage of the sand through being wetted a considerable space is left; this is filled up with more roasted ore, and a weak caustic-soda solution run on and all drained out. As is well known to cyanide men, lime gives a clear solution, while the alkalies often give muddy ones. The cyanide solutions are then run on, and the solutions are passed through a filter box. This consists of a box, having a vertical partition of filter cloth. The solutions run in on the top on one side of the partition drain through the filter, and are withdrawn through a pipe at the bottom on the other side. This filter never clogs, and, as the sediments which lie against it fall off as soon as the liquor becomes low, it may be used for a long time. The solutions are run through the zinc boxes in the usual way, the box having 16 sections. The zinc is moved up daily, and only a trace of gold in solution leaves the final compartment. The zinc used amounts to ½lb. per ton of ore. After considerable experience in the working of this process Mr. Allsop is confident that a fortnight would be a sufficient time to dissolve gold out of any material after having been roasted. Of course this method has the advantage of recovering silver as well as gold from an ore, and the bullion obtained refined by Mr. Allsop at his laboratory in Armadale.

Treatment of Cassilis Ore.

Description of Ore.—The ore, which is very refractory, consists of arsenical pyrites, iron pyrites, zinc blende, galena, antimony sulphide, magnesium and aluminium silicates, quartz, and traces of copper. Some of the oxidised ore, of which there is very little, has a very interesting composition; a specimen was shown to me by Mr. F. B. Stephens, the general manager, who stated that it consisted of hydrated arseniate of iron, orpiment, pyromorphite, lead oxide, quartz, magnesium, and aluminium silicates. Ore of this description assayed as high as 60oz. gold per ton, and is found in a leader which runs parallel to the main ore body, at intervals cutting through it.

Reduction and Treatment of Ore.—The mine, which is situated about 1000 feet above the plant on a hill, whose grade is about 1 in 2, is connected with the works by means of a double road tram-line, the loaded trucks coming down serving to haul the empty ones up, thus obviating the necessity of motive power for haulage. From the trucks the ore is delivered on to a grizzly, which separates the fine from the coarse, the former passing into a bin below, and the latter into a 9-inch by 16-inch Jaques rockbreaker, capable of crushing 10 tons per hour. From the breaker the crushed ore goes into the bin below, from which it is fed with the fines into the mortars. The feeding is done automatically by means of a hinged shoot, to which a shaking motion is given by a tappet keyed to the centre stamp, and so arranged that when the ore becomes too low in the mortar the tappet strikes a projecting arm, attached to the shoot, thus causing some of the ore to be shaken into the mortars. The mortar and stamps are of the general Victorian type. The stamps weigh 950lb., and are run at the rate of 90 drops per minute, with a fall of 8 inches. The power for driving the battery is transmitted from the engine by means of a pinion wheel connected on to another pinion wheel at the end of a counter shaft, which runs parallel to the cam shafts, for the whole length of the mill. Each 10-head of stampers has a continuous cam shaft, to one end of which is attached a pinion wheel, which is driven by another pinion wheel connected on to the counter shaft. There are 20 head of stamps divided equally between 4 mortars. The motive power is supplied at present by a high-pressure non-condensing engine, the steam required being supplied by two one-hundred horse-power boilers, one of which is a Cornish boiler and the other an underfired multitubular. The engine is to be taken out and a new 80 I.H.P. compound vertical engine put in its place. The screens used on the mortars are of 20 mesh woven. Other sizes, both in punched and woven, have been tried, but it was stated that the 20 mesh are the best and most economical. From the mortars the pulp passes over amalgamated copper plates, the dimensions of which are approximately 6 feet wide by 10 feet long. There is one mercury well and 3 ripples. The fall is from 1 inch to 1½-inch per foot. From the copper plates it passes on to the Wilfley concentrators, of which there are four, running at the rate of 240 strokes per minute. The galena is cut off almost free from the rest of the minerals at the back of the table, and reconcentrated on a Wilfley table kept solely for that purpose; when cleaned, they are wheeled to a small room, where they are ground in Berdan pans with mercury, three pans being used for this purpose. The amalgamating is done in charges, with the water standing about two-thirds way up the side of the pan. When the amalgamation is complete, the pans are cleaned out into settling tubs, and the mercury recovered by agitating and allowing the slimes to settle. The mercury is then squeezed through chamois leather or calico, and the amalgam thus got retorted and resultant gold bullion smelted. By this means about 48oz. of gold per ton is recovered from the galena. The slimes from grinding and the seconds from reconcentration are bagged and sent to the Illawarra Smelting Works (N.S.W.), where the gold and silver contents are paid for. The slimes average about 16oz. gold per ton, and 37 per cent. lead. The loss of mercury in the amalgamating process is said to be about one pennyworth per ton of ore crushed. The balance of the minerals are cut off from the quartz and silicates of aluminium and magnesium, bagged and trucked to the roasting sheds, where they are roasted in two Edwards' Furnaces (mechanical), each capable of roasting 4 tons of the concentrates per day. The output per 24 hours is small on account of the large percentage and nature of sulphides. The speed at which the rabbles are driven is: No. 1 (or the discharging rabble), 4 revolutions per minute; No. 2, two revolutions per minute: and the remainder at the rate of one revolution per minute. The first five rabbles are water jacketed, the balance are not. The power needed for driving the rabbles is supplied by an eight by twelve inch steam engine, running 150 revolutions per minute. This engine also drives the dynamo used for lighting the works, the lathe, sampling machine, cutting and punching machine, in the blacksmith's shop, and the bucket and push conveyors. The roasted concentrates,* as discharged from the furnaces, are conveyed from these to the chain and bucket elevator by means of a small push conveyor. The elevator raises the roasted concentrates up into a bin about 50 feet above the floor of the furnace shed. The slow passage of the roasted material through the air gives it ample time to cool, thus making it unnecessary to have a cooling floor. From the bin the roasted ore is trucked and discharged into the chlorinating vats.

The chlorinating plant consists of seven treatment vats (circular wooden) each 12 feet in diameter by 3 feet 6 inches deep (i.e., clear of the filter bed); four precipitating vats, each 8 feet diameter by 4 feet deep; and two storage vats, each 5 feet by 4 feet deep. This process used is a modification of Muncktell's. In this process the chlorine is generated by the action of dilute sulphuric acid on a solution of bleaching powder. The chlorine is liberated according to the following chemical equation:—

$$Ca\,(OCl)_2\,CaCl_2 + 2H_2SO_4 = 2CaSO_4 + 2Cl_2 + 2H_2O.$$

Two hundred and twenty lbs. of bleaching powder are used per

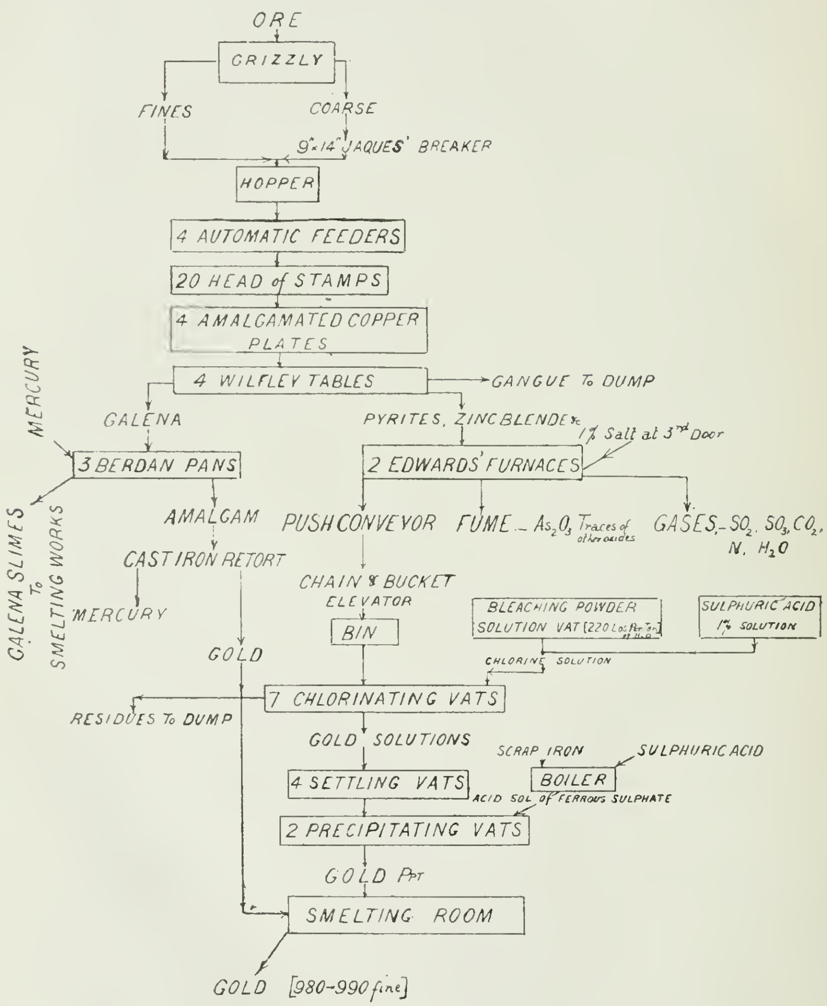
*Note.—About 1 per cent. of salt is thrown in on the last hearth.

eight tons of water, and a one per cent. solution of sulphuric acid. The two solutions are run on to the charge simultaneously, and in the desired proportions by means of a Y pipe connecting the two vats. The strength of the chlorine solution thus formed is about 0.15 per cent. The time required for treatment is about seven days. The chlorine solutions, after passing through the ore, pass into the two settling vats, from which the clear solution is drawn off into the precipitating vats, and the gold precipitated by means of an acid solution of ferrous sulphate ($FeSO_4$), made by treating scrap iron in a boiler with sulphuric acid. The ferrous sulphate solution is added in bucketfulls until by testing with gold chloride it is found to be in excess. The precipitated gold is allowed to settle for about 48 hours. The clear liquor is then drawn off from the precipitate by means of a rubber pipe, connected at one end to a pipe passing through the vat at about 3 inches above the bottom, and at the other end to a dish which floats on the surface of the liquor. By this means the whole of the clear liquor can be drawn off from the precipitate without disturbing it. The vat is then tilted on one side and the balance of the liquor, with the precipitate baled out by means of buckets into small tubs, where it is again allowed to settle and the clear liquor siphoned off. The precipitate is then treated with sulphuric acid, washed, thrown on to a filter, allowed to drain, and then dried in an iron pot. When dried the precipitate is heated to redness and nitre added. After this treatment it is smelted with borax. The gold precipitates contain from 70 to 80 per cent. gold. The purity of the bullion is about 980 to 990 fine in gold. The extraction by chlorination is from 85 to 87 per cent. of the assay value of the concentrates.

Mr. Stephens, in making some alterations and additions to the plant, proposes to shift the concentrators to a floor lower down, and to place, in their present position, hydraulic classifiers of an American type, which he has slightly modified. The sides of the classifiers, which are cone shape, are made of quarter inch steel plate fastened into cast iron at the apex. Drawings 3 and 4 show the classifiers in section and plan. The water is admitted from the top through a pipe leading down to within a few inches of the bottom; as the water issues from the pipe under pressure it strikes a baffle plate, fastened to a flange and about two inches below the end of the pipe, which causes the water to be deflected upwards. From the classifiers, of which there are to be four sets, the classified material is to pass on to the Wilfley tables to be further concentrated. The slimes overflow from the classifiers is to be passed over canvas tables and then to the cyanide plant. The cyanide plant is to have a capacity of 500 tons per week, consisting of five treatment vats, each with a capacity of holding 500 tons tailings. The precipitation of gold from cyanide solution to be effected by zinc turnings.

A heap of tailings stored below the plant, and containing about 16,000 tons, is to be retreated. The whole of the material is to be hauled up to the works and passed over amalgamated copper plates, and then through classifiers capable of treating 40 tons per day, and then on to Wilfley tables, from which a strip is saved. The balance of the material is then to be treated by the cyanide process. An electro motor is to be used for hauling the

DIAGRAM OF TREATMENT
Cassilis Ore



tailings and driving the concentrators. The distribution of power in present plant is approximately:—

Rock Breaker, 12 Horse-power.
20-head Stamps, 35 Horse-power.
4 Wilfleys, 5 Horse-power.
Edwards' Furnaces. Machine Tools, and Electric Lighting, 12 Horse-power.

SOME ANALYSES ON FURNACE PRODUCTS.

Raw Concentrates.

Arsenic	7.8 per cent.
Sulphur	25.5 per cent.
Iron	25.5 per cent.
Zinc	5.9 per cent.
Lead	1.88 per cent.
Copper	0.01 per cent.
Insoluble	33.4 per cent.
	99.99 per cent.
Antimony	.01 per cent.
	100.00 per cent.

Furnace Products.

Number of Doors—Numbered from the Discharge End—1 to 15.

	Raw.	14	12	11	9	7	5	4	3	2	1
	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.
Arsenic	7.8	1.5	.43	.304	.232	.293	a	.387	a	a	.136
Zinc	5.9	7.9	5.95	5.17	3.96	3.7	3.7	3.71	3.5	3.5	3.5
Sulphur in form of sulphate...	a	a	.37	.283	.728	1.386	1.2	1.396	1.338	1.4	1.4
Sulphur in form of sulphide...	25.5	11.67	4.04	1.96	a	1.73	1.52	a	a	1.41	a
Lead in form of sulphate ...	—	.379	.52	.68	1.62	1.85	1.87	2.18	2.09	a	a
Magnetics ...	—	52.7	74.1	80.2	62.8	15.2	7.5	6.6	1.6	1.4	.7

Analyses Magnetics.

	14	12	11	9	7	5	4	3	2	1
	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	p.c.	
Insol.	31.4	27.7	27.8	23.6	20.86	a	a	10.5	10.9	—
Sulphur	8.5	4.03	3.8	3.4	3.05	4.3	4.7	3.61	2.8	—
Iron in form of FeO	13.9	13.2	13.	2.9	1.1	a	.9	.4	.4	—
Total iron	39.9	43.9	45.3	42.7	46.2	40.9	44.8	35.88	35.9	—

NOTE.—a not estimated.

Maude and Homeward Bound.

How many have a sad remembrance of Glen Wills? Eleven years ago expert after expert disappeared into the Mount Wills country, and came back with such glowing reports that the surveyors' straight lines were the only limits to the prospective wealth of the fortunate leaseholders. Travelling over the same ground again one may see everything but tin, yet if the samples which were being obtained ten years ago were present in any abundance there are many lodes which would have returned a profit to the small co-operative party, provided there was a good central dressing plant. Just in the same way every reef is not good enough to support its own battery, yet will give good returns if the patches or shoots alone are taken out and treated.

Leaving the Glen and going down the creek one comes upon the remains of the Mount Wills South reducing and concentrating plant, and many will remember the "soft lode" in connection with this property. Only a short distance below cyanide works are treating some auriferous tailings, while almost adjoining the old Mount Wills South lode the lease of the Maude and Homeward Bound is pegged out. The country consists of schists penetrated by granitic intrusions, while dykes, probably dioritic, run through both schists and granites. The reef in the Maude and Homeward Bound has been traced along the surface from the summit of a hill about 1000 feet above the level of the creek. It was opened up by a tunnel, which lies about 400 feet below the summit of the hill. A portion of the ground has been stoped by the original holders, but the great bulk of it over the level has not been prospected. The stone is a bluish quartz, carrying arsenical pyrites, pyrite, a small quantity of sulphides of antimony, and also sulphide of silver in combination with antimony. These minerals are crystallised in small particles in the solid quartz, and this causes concentration to be difficult. The color of the stone is also due to finely-divided sulphides. The quartz itself is exceedingly tough, one might almost say leathery, and difficult to break, and causes the battery to give such a low output. The metallic mineral contents of the stone would probably be from 3 to 5 per cent.

Surface Works.—The surface works are as follow:—An ore hopper at the mouth of No. 2 tunnel takes all the ore. It is then trucked 400yds. on a horse tram; it is emptied into another hopper, and then sent down a hill 500 to 600 feet high by an automatic tram until it reaches the kick-up, when the truck discharges the stone into a hopper. The journey occupies a minute. The empty truck is then hauled up as another loaded one starts on its journey. The ore from the battery hopper is led on to an automatic feed box of Mr. Morgan's own device. It consists of a rectangular trough, open at both ends, 8 feet long, 20 inches wide, and 8½ inches deep, made out of ordinary hardwood, and set at an angle of from 25deg. to 30deg. The upper end is hinged, and receives the stone from the hopper; the lower is suspended by an iron strap, which passes round the end of the box. This is hung on an iron rod, the upper end of which passes through an ordinary sapling, where it is secured by a thumb-screw, so that the angle of inclination may be raised or lowered. The sapling is placed horizontally, and acts as a spring. Another stout bar attached to the lower end of the trough is led upwards through a loose guide, and on it is the usual sliding projection, which is set at the desired height under the central tappet. The feed may be regulated to a nicety. The merit this feeder possesses is simplicity, while its cost would not exceed a few shillings for material. The battery of 15-head was driven by a Pelton wheel.

A defect one continually sees in connection with these plants is that all the cams are placed on one side of the stems, leading, of course, in such a case as this, to a considerable useless end thrust. The speed was 66 drops per stamp per minute. The sand was about coarse enough for an iron punched screen of 225 holes per square inch; mercury was fed into the box. The pulp discharged into a well, and then over 10 feet of copper plate, then on to a short length of blanket table. After this it passed into a launder, and thence into spitzkasten, the slimes going to a Frue vanner, and the sand to 3 Halley tables. The overflow from the Halley's was led on to 5 Luhrig vanners. The blanketings were ground in a large Berdan pan with drags, mercury being added from time to time; the residues from these passed into spitzkasten, thence on to a Frue vanner. A fourth Halley was kept for cleaning up the concentrates obtained from the others. The addition of a mercury well at the lip of the box is not advisable. The amalgamator has no clue to the state of his box. By having an amalgamated plate the dryness or sloppiness of the amalgam at once shows the true state of affairs with regard to the amount of mercury fed in—the quantity can be regulated accordingly. The amalgam which can be caught in a well can be just as readily caught by a good plate. In almost all cases wells could be dispensed with altogether. If plate amalgamation is properly attended to the amount of gold caught in wells is meagre, while the loss of mercury is great. With wells of ordinary construction, even when the baffle boards are in, very little of the sand comes in contact with the mercury at all. All the devices which amateurs aim at in trying to bring gold in contact with mercury by forcing the sand through a bath of the metal are useless. A system adopted by Mr. D. White at Stawell is an undoubted improvement. A round bar of pure copper floats in a well, the stream of pulp flowing over it causes it to rotate, and present a fresh surface for amalgamation as the sand flows past.

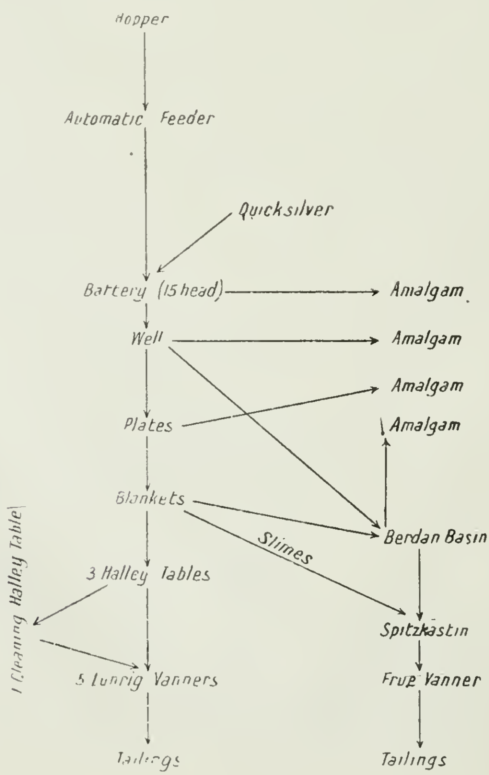
The yield per ton is from 18dwt to 19dwt. free gold. The concentrates are about 3 per cent. on the tonnage. The following table will serve to show the work done by the various concentrators:—

In a 30 ton parcel—	Percentage Saved.	Gold per ton. oz. dwt. gr.	Silver per ton. oz. dwt. gr.
3 Halley tables	65	5 12 0	22 17 0
1 Frue vanner	20	8 0 0	28 0 0
5 Luhrig vanners	15	4 0 0	17 0 0

or, say, in a 100-ton parcel the three Halley concentrates would be worth £1500, the Frue vanner concentrates £650, while the five Luhrig vanners would be worth £250. This demonstrates the value of that crude but effective concentrator the Halley, and though the Luhrigs have to take out what the Halley loses, and they were working well, it must be admitted that they are machines of small capacity. The Frue vanner was doing good work, but it was getting rich material. This vanner is an excellent one, but is out of place on most mines, owing to lack of attention. Unfortunately, battery boys are not to be trusted with such nicely adjusted concentrators, and though good work may be got out of them, the many oscillating parts, and, worst of all, the moving belt have relegated most of them to obscurity. Their successful competitor, the Wilfley, will last longer, since the wear and tear will be less, the capacity greater, and the adjustments simpler.

It must be remembered, in spite of what returns are, that gold is a most difficult metal to concentrate. In the present instance the pyrites are extracted almost perfectly, yet more gold is getting away past the concentrators than is caught on them. For instance, the stone yields from 18dwt. to 19dwt. per ton, which is about equal to 15dwt. pure gold, so that taking a 100-ton crushing the stone would yield 75oz. pure gold, the concentrates obtained about 18oz., while the sand or tailings, assaying 6dwt., contains 30oz. of gold. It is obvious, however, that searching cyanide tests should be made in order to minimise this loss. Since the foregoing was written the sands have been successfully cyanided.

The following diagram will serve to show at a glance the methods in use at the mine:—



The South German, Maldon.

So far as experience teaches in Victoria it is undoubted that reefs become poorer at a depth. Numerous and brilliant examples to the contrary might be quoted, but these are only meteoric flashes which enlighten the darkness. On the other hand, one must bear in mind that it is much easier to discover golden stone on the surface than to do the same underground. An acre of surface can be tested almost as readily as a fathom at deep levels. If it were possible to obtain the information it would be exceedingly interesting to find what amount of gold was won in Victoria from each hundred feet in depth. Maldon would serve as a most notable example of extraordinary surface richness, and at the same time show that payable gold was got at great depths. A hundred tons for 90oz. to the ton was recorded many years ago for a trial crushing of surface stone.

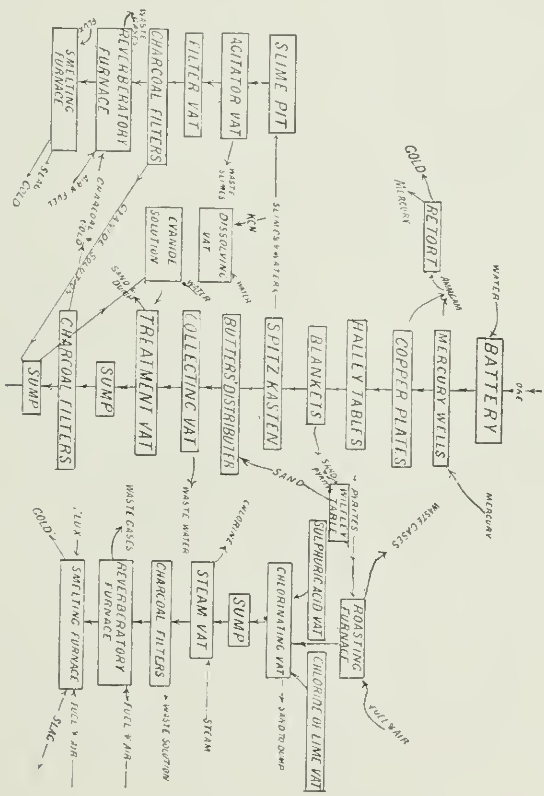
Maldon is also famous in possessing a mineral only found to occur in that district, and called Maldonite. It has the formula Au_2Bi, and is therefore a native alloy of gold and bismuth. The quartz from most of the mines has that peculiar glassy look so common when granite or a contact rock is the matrix. The percentage of sulphides and similar minerals is not high, but they are so associated with the gold as to make a good extraction with ordinary appliances impossible.

The best equipped plant in the district—and one might say in that part of Victoria, including Ballarat and Bendigo—is to be found at the South German mine. The Hon. W. B. Gray, who is managing director, is not a man who will look on in contented indifference if gold is getting away which he can save at a profit.

The works have grown around a battery erected many years ago, and since the site cannot be said to be in many respects suitable, except that it is near the shaft, much ingenuity has had to be expended in the transference of material from one part of the works to another. The material as delivered from the shaft is fed into a battery of 30 head of stamps, each averaging 900lb. in weight, with a drop of from 7 inches to 9 inches, and running 72 blows per minute. No stonebreakers nor automatic feeders are employed. The pulp is forced through punched sheet iron screens, having 225 holes to the square inch. It then passes through mercury wells, then over copper plates the full width of the battery box. No mercury is fed into the boxes, nor is inside amalgamation practised. The sand then passes over the well-known Halley's percussion tables, there being one for each box; after leaving the Halley it passes over blanket tables—the blanketings obtained being further concentrated by dressing them upon a Wilfley. The whole of the concentrates are then chlorinated. The sand, deprived of its coarse and part of the fine pyrites, is then elevated 30 feet by means of a sand pump. This contrivance acts well, and is much simpler than the elevator wheels and belts and buckets used in other places. After leaving the sand pump it passes into a spitzkasten, or pointed box, in which the slimes are separated from the sand. The slimes go into a series of settling pits, four in number, 35 feet long, 10 feet wide, and 3 feet deep. The sand is led away to four vats, 27 feet in diameter and 7 feet deep, each fitted with Butters' automatic distributor. These simply act as storage vats, it being found to be better practice to deal with the sand in other vats.

The tailings are removed to four treatment vats, each having a capacity of 65 tons. The solutions are supplied from three vats, and one vat is kept for dissolving or making up the strong solutions. The solutions are led into charcoal filters, of which there are 48. Half of these are cleaned up every month by burning in a reverberatory furnace and smelting the ash. The slimes are treated separately. They are dug out of the settling pits and emptied into two of Deeble's patent agitating vats, which resemble the settling pans used for amalgamating as regards action. Their dimensions are 18 feet in diameter and 4 feet deep, and the agitating paddle may be screwed up or down. The method of working these is as follows:—Cyanide solution from the sump is run in for a depth of 2 feet; the paddles are then set going, and the slimes tipped in until the pulp is within 3 inches from the top. This corresponds to about 15 tons. About 80lb. of lime are then added, partly for neutralising acid, but mainly for settling the slimes. Cyanide is then added to bring the solution up to a strength of about 0.15 per cent., and agitation continued for about 30 hours. From a number of tests made it was found that up to this time the solutions were growing richer in gold, but after this no further solution took place. The agitator is then screwed up and the slimes allowed to settle. This takes from 8 to 16 hours. The clear solution is decanted, run through a filter vat to arrest any slimes which may have been carried over; the clear solution is then pumped up to a storage vat, from which it gravitates in an even stream to the charcoal filters. The slimes still contain a considerable quantity of solution and also gold. This is decreased by running on some more solution from the sump, and agitating and decanting the second time; if necessary this process is repeated, and finally, a water wash is run on, the quantity being gauged to keep the amount of solution in circulation constant. A very high extraction is claimed for this process, or 92.5 per cent. from 9dwt. material; which certainly leaves nothing to be desired in this respect.

Since precipitation of gold from cyanide solutions by charcoal is very largely adopted in Victoria, and is almost unknown elsewhere, it may be of interest to supply a few details. The charcoal used is preferably a softer variety; this is crushed to about the size of beans and washed with water, all dust and dirt being removed. It is then placed in the precipitating tubs, which consist of wooden buckets. Each holds about 5cwt. of charcoal. An earthenware pipe takes the solution down through the centre of the charcoal tub; it then rises evenly through the body of charcoal, which is weighted down on the surface. The solution then flows through four similar precipitating tubs. It was found that, with a flow of 200 gallons of liquor per hour, the amount of gold precipitated in each tub was in the following proportion:—No. 1, 45 per cent.; No. 2, 25 per cent.; No. 3, 15 per cent.; No. 4, 9 per cent.; No. 5, 5 per cent., making in all 99 per cent. of the gold in the solution, which only contained 2gr. per ton after passing the last filter. It will be seen that the precipitation is practically perfect, but that a very large amount of charcoal must be used, or 60lb. of charcoal to every ounce of gold. It is worthy of note that a partial precipitation of the gold takes place, the first filters being as a rule rich in gold, and the last rich in silver. As I pointed out in previous articles, this is also the case when zinc is used as a precipitating agent. In any case the bullion from the charcoal is much purer than that obtained from zinc. Mr. Gray holds that charcoal is preferable to zinc as a precipitating and working material, and that it is cheaper. It has been found that charcoal precipitates every metal from solution with the exception of the alkalies and alkaline earths, so that the cyanide solutions are kept beautifully clean; it is further stated that charcoal—contrary to current opinion on other fields—does not decompose cyanide of potassium in solution. Even conceding all this, the space occupied by the



Diagrammatic Plan of Plant.

charcoal, the trouble in keeping it clean, the burning and smelting constitute serious drawbacks, which will probably prevent its general adoption.

The concentrates obtained from the Halley and Wilfley tables are roasted in a furnace of the MacDougall type. Occasionally a long-hearth hand-rabbled reverberatory is also used, but the mechanical furnace has been found to give a perfect roast. The ore is placed in vats, and Munktell's process of chlorination—that is, with chloride of lime and sulphuric acid to generate chlorine—is carried out. The weak chlorine solutions are found to be effective, and a high percentage extraction is obtained. The solution containing the gold is heated by steam to expel the chlorine, and then run through a series of charcoal jars, where the gold is rapidly precipitated. The charcoal is burnt in reverberatory furnaces, and the ash containing the gold smelted with suitable fluxes.

Edwards' Pyrites Works, Ballarat.

The Edwards' Pyrites and Ore Reduction Co. has gold extraction works situated at Bendigo, also at Sebastopol—a desolate looking suburb of Ballarat. The method used for gold recovery is by roasting and chlorination. The ore arrives at the works either as broken at the mine or as concentrates from stone which has been treated by amalgamation. The coarse material is put through a ball mill. The concentrates or pulverised material is then emptied into bins, each bin receiving ore according to its composition, so as to ensure suitable blending of minerals, which re-act on each other when heated, so that no undue proportion of any refractory substance be fed into the furnace. From the bins the ore is trucked over a weighbridge; a sample is taken at the same time, so that the actual weight, composition, and gold contents may be determined. After weighing it is delivered into the feed hopper of a mechanical furnace. On being discharged the roasted ore is damped, and emptied into small closed vats; chlorine is forced in, and after the gold has dissolved the soluble chloride is leached out with water and pumped into vats. The solution is heated by steam, and the warm liquor is run through charcoal filters, in which the gold is precipitated. Special trial lots have the gold precipitated with ferrous sulphate.

The novel features in connection with this plant are, first, the Edwards' mechanical roasting furnace; secondly, the vats; and, thirdly, the chlorine generator.

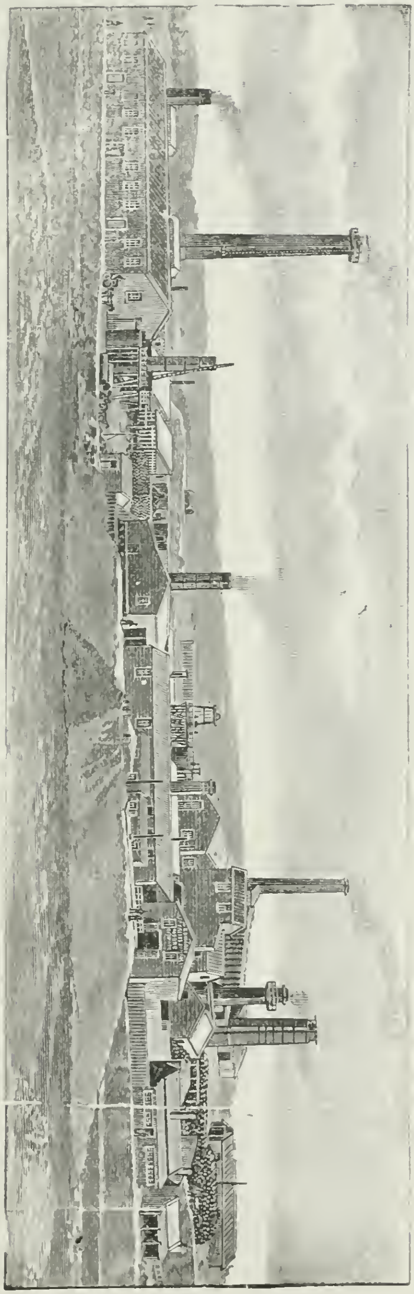
At Sebastopol, Mr. Edwards introduced a gas producer, so that by means of pipes the gas could be burnt at any point within the furnace, it being found with such ores as those received from the Great Boulder that the heat did not pass from the fireplace to the upper end of the furnace, on account of the sulphur contents being so low. By supplying producer gas about midway along the furnace the temperature necessary for roasting was attained and all difficulties overcome. Independent of considerations such as these, it can hardly be said that there is any advantage gained by converting coal into a product which gives less heat than the coal by means of which it was produced. In other words, if the heat could be applied in the same way it would be cheaper to burn the coal straight away for heating purposes rather than convert portion of it into a gas which has a lower calorific value.

It is now generally accepted that when sulphur burns the trioxide, not the dioxide, first forms, but since the heat generated is so great—

$$S + O_3 = SO_3 + 91.8 \text{ cal}$$

—that the temperature, due to the heat of formation and the added heat of the furnace, is so high as to dissociate the SO_3 into SO_2 and O. It is well known that when SO_2 is passed with air over heated spongy platinum that SO_3 forms, and the probable explanation is that the platinum, which condenses oxygen on its surface when heated to about 250deg. C., brings about an intimate contact of the sulphur dioxide and oxygen, and SO_3 forms, which will not dissociate if the temperature during the time of contact is not allowed to rise. Ferric oxide and other materials will bring about the same action. On the other hand, sulphur if burnt in quantity will produce SO_3 as well as SO_2, and the white clouds which arise from burning pyrites in a dry atmosphere are not SO_2, but are mainly due to SO_3. In spite of all that has been done with furnaces, it must be confessed that on the whole very little attention has been paid to the conservation of heat. It is true that many desirable re-actions have to be brought about at a fairly high temperature, and if the temperature were gradually raised undesirable compounds would form; while this is true in certain cases, yet in others, such as the roasting of almost pure pyrites, the temperature should be raised gradually (this being the result of Captain Richard's experience), so as to cause the greater part of the pyrites to pass through the sulphate stage. This again points to the principle applied by Hoffman and others in the construction of their kilns, where waste hot gases are used to warm up the raw material fed in. Again, if the ore to be roasted is brought to its final temperature—certainly, as a rule, not less than 800deg. C.—there is a waste of heat in allowing this to cool in the open air. The specific heat of sand, viz., 0.1881, is not very high, but since the weight discharged is great the heat units lost—which may easily be calculated—will be found to represent a large amount of energy which should have been conserved. The amount of air supplied to roasting furnaces is an indefinite quantity, and if exact experiments were carried out would be found to be largely in excess of actual requirements. Thus fuel is burnt, and to a great extent wasted, in warming air which escapes hot after doing no work, in heating sand which is discharged hot from the furnace, and in heating rabbling appliances; much heat is also lost by conduction and radiation. The end of improvements in roasting-furnace design has not been arrived at, yet much has been done, and more will be done by those who take into account both physical and chemical factors.

The type of vat adopted by Mr. Edwards is practically that of the Plattner type—each holding less than two tons—excess of depth being given no doubt to save chlorine, which always fills the vacant space below the filter bed. The vats are not lead-lined; the cover has a lead coating on the under side; the rim of the vat has a groove cut round it; in this is pressed a couple of strips of rubber, side by side; these project about half an inch. The lid is lowered on to these and a perfect joint is formed; by means of bolts a greater pressure may be brought to bear in fastening the cover down. Chlorine is produced from manganese dioxide salt and sulphuric acid. The manganese ore is ground in a ball mill, no further trituration being considered necessary. The generator used is one of Mr. Edwards' design; it consists of a steam-jacketed hemi-cylindrical vessel, made of boiler-iron or steel, with a thick internal lead lining. The cover has openings for the introduction of the chemicals; these openings may be readily closed by small lead lined lids which may be screwed down tightly. The whole vessel is supported on trunnions, and a rocking motion may be imparted to it which serves to prevent the caking and packing of the material. The gas is passed through a wash bottle securely fastened



General View of Works.

down. This simply indicates the rate of flow of the gas as it passes into the vats, under a slight pressure. Solution of the gold takes place rapidly. When dissolved a water wash is run on, and this chlorinated water is allowed to stand in contact with the sand for some time. The chlorine gas not absorbed by the descending water is driven before it through an outlet pipe into the next vat, which is then cut off from the first. Chlorine is then passed into the second vat from the generator. The first vat is water-washed, suction being used to assist filtration; the gold solution is pumped up into a sump; steam is passed in to displace the chlorine and warm the liquor. The solutions are then run through charcoal placed in earthenware jars, the charcoal being coarse (not like the finely-graded material in use at Mount Morgan). The charcoal is burnt from time to time in short reverberatory furnaces, and the gold recovered from the ash by smelting in the ordinary way. The whole proceeds as carried out appears, so far as outside indications go, to be exceedingly simple, yet of all the firms in existence many years ago in Victoria this is one of the few which has been successful. Altogether it may be taken as a modified Plattner process, which has been adopted in place of the barrel and other systems. A small smelter has been erected, but is rarely, or never, used.

Jaques Bros., Richmond.

The ore treatment works of Jaques Bros. are situated at Coppin-street, Richmond, Melbourne. This firm is well known for its rockbreakers and various forms of mining machinery well adapted to our local conditions. At their general manufacturing and engineering works they have erected a neat testing plant, which should be capable of doing as good work as any of its kind in the State.

Their reduction plant comprises three head of stampers, each stamper weighing about 3cwt., a set of rolls and stonebreaker combined. The concentrating plant consists of a 5-compartment Luhrig jig and two Luhrig vanners. The roasting plant consists of a patent automatic furnace, 30 feet in length by 4 feet 6 inches wide, divided into three floors, with drops between each floor. The rabbles are horizontal arms attached to vertical spindles, which pass through the crown of the arch and are driven by gearing above. Each set of rabbles works the ore on its own hearth, which is circular, by simply cutting in under it and allowing the sand to flow over the back of it, thus exposing it in a falling stream to the oxidising action of the heated air passing over it and through it. As soon as the ore is deemed sufficiently altered in the first compartment a sliding shutter is opened and the partly roasted sand is worked on to the second hearth. There it is kept for some time, and when deemed sufficiently roasted on that hearth, it is worked on to a third, nearest the fireplace, where it is finished off. The rabbles on this hearth are water jacketed and may be driven by an oil engine. Coal was used for roasting instead of wood, and heat was conserved to a certain extent by keeping the crown of the furnace 13 inches above the hearth. In passing it should be stated that this furnace reminds one of the McDougall furnace. A form of the latter as used in Victoria consists of a vertical cylinder having a series of horizontal arches across it. A vertical shaft, with horizontal arms attached, runs through the centre of the cylinder. The ore is stirred by the revolving rakes on top of each arch, and may be admitted to the next floor by the withdrawal of a shutter, or by openings in alternate arches at the centre or near the side. It was found that this led to dusting, so McDougall constructed a furnace with a long hearth, with rabbles working in circular tracks all the way down it, each rabble revolving in an opposite direction to the one on either side of it, while the circles described by each rabble cut into that of the next. In this way the material was brought down from one end of the furnace to the other, but since it is obvious that if each rabble or rake brought down a fraction of the ore above it at every revolution, then there would be a danger of portion of raw ore handed in from first to last without having been oxidised. It will be seen that Messrs. Jaques' furnace is somewhat on the same lines as McDougall's vertical furnace, and yet has not the drop which would lead to excessive dusting. The ore subsequent to roasting is treated according to the Plattner process in small vats, each holding about 1½ tons. There are three of these vats. After roasting the sand is cooled and damped with a very small quantity of water—just enough, in fact, to say it was damp. Some dry sand is sieved into the bottom of the vat to absorb excess of moisture in the filter bed. As soon as this ceases to absorb water from below the damp sand is fed in, pressed around the sides, and chlorine gas fed in through a pipe under the false bottom of the vat. In a few hours the gas has reached the surface of the sand, and the cover of the vat is screwed down, a seal being made by a couple of projecting strips of rubber being let into the rim of the vat. These are squeezed against the plane under surface of the lid. After some time the gold will have dissolved. The time is seldom less than 36 hours, but if not extracted in at least 72 hours then it is of little service leaving it in longer, since the rate of solution is excessively slow for the last particles. This process, which was the first one ever made use of practically by Plattner, has much in its favor. First, since gas is passed through the damp and, the water present is saturated with the gas, and the gold is much more quickly dissolved in a saturated solution than in a dilute one. Further, should there be an absorption of chlorine, this is quickly made up for by the excess of gas between the particles of sand. The second point in its favor is that, being in a closed vessel, it is not decomposed by the action of light as would be the case to a considerable extent with open vats; while its third advantage is that the absorption of chlorine is indicated by the time the gas takes to reach the surface of the ore. In some cases as little as from 3lb. to 4lb. of chlorine suffices for a ton of ore, but these cases are exceptional. The disadvantages, and these on a large scale outweigh the points in favor of the system, are, first, the smallness of the vats; secondly, the trouble of generating the gas and keeping it going; thirdly, the trouble of giving a preliminary acid or water wash to some ores; fourthly, the large quantity of gas absorbed by lime and magnesia, too; and finally, the almost impossible task of reducing operations to routine on a large scale where so many vital points require attention during day and night.

There are six precipitating vats, 4 feet by 3 feet by 2 feet 6 inches. When the gold has been dissolved water is run into the vats, and the soluble chloride of gold run into the precipitating vats. It is there precipitated with a solution of sulphate of iron, allowed to settle, the liquor drawn off from the gold sludge; the latter is then run out, filtered, and the mud-like material dried and smelted into bars of almost chemically pure gold.

Messrs. Jaques also have a small cyanide plant, consisting of two one-ton vats, with solution vats, zinc boxes, and the necessary other appliances. There is an excellent chemical and assay laboratory attached to the works, all under expert supervision.

NEW SOUTH WALES.

BROKEN HILL.

I desire, in connection with the following Broken Hill chapters, to acknowledge the assistance given by the managers of the mines and their staffs, also the excellent reports published by the Mines Department of New South Wales, notably the "Mineral Resources of New South Wales," by Pittman, and the "Geology of the Broken Hill Lode," by Jaquet. Other references will be made in due course.

Until the year 1844 the interior of Australia, except where rivers ran, was practically unknown, but the explorer, Sturt, who in that year started from the Darling River, passed the spot now known as Broken Hill. This lies in New South Wales, about 120 miles north of the Victorian border, and about 35 miles from the South Australian. For over 20 years after Sturt's expedition the Barrier Ranges, at the southern extremity of which Broken Hill lies, were only known to a few adventurous stock-owners, who pushed their way further and further back. The discovery of gold in Victoria and New South Wales unsettled men's minds, so that when, in 1867, it was reported that gold had been found in the Barrier Ranges, miners from Burra Burra, in South Australia, rushed off to the new land of promise. The tale of the sufferings and privations of many of these unfortunate men will never be told. In an arid, semi-tropical, and almost waterless country many perished, while the survivors found no gold, and returned with such harrowing descriptions of the place that it was shunned as a land accursed; yet had some of these men possessed a knowledge of mineralogy they might have amassed wealth surpassing that of any gold mine-owners. Later on the pioneers with their flocks and herds settled on outlying districts. Salt bush grew luxuriantly, and in occasional seasons a magnificent sward of grass turned the desert into a garden, while water was conserved in dams. When the country had become sparsely settled a discovery of argentiferous galena was made in 1876, at Thackeringa, which is about 20 miles south-west of Broken Hill. Mr. P. Green, of Wilcannia, a place over 140 miles away to the north-west, raised some 36 tons of ore, got it carted to Burra Burra, and sent it to England. Unfortunately, it was jettisoned on the voyage. The set-back caused by this loss was four years, for it was not until 1880 that Green raised another lot of 100 tons. This was sent to England, and two years afterwards the report came out that it contained about 65 per cent. of lead, and 35oz. silver per ton. About this time gold was discovered at Mt. Brown, about 150 miles to the north. This caused a rush, and during this time the value of the Thackeringa mines becoming known, drew much attention to the place. Shortly afterwards Umberumberka was discovered, and, in 1883, Silverton, Apollyon Valley, and the Pinnacles were found to contain rich argentiferous ore. Individual prospecting was stimulated by the finding of large masses of chlorobromide of silver, and chloride of silver. Many of these contained from one third to one-half their weight of silver. These lumps were found on or near the surface, and were locally termed slugs, in contradistinction to the nuggets of the gold miner. By a strange turn the term has been applied to the surface gold in Western Australia.

It can scarcely be wondered at that the station hands of the district caught the mining fever. A boundary rider on the Mt. Gipps Station, which included a large extent of Broken Hill country, pegged out the biggest outcrop he could find. He considered he had struck a great tin mine. It is somewhat remarkable that Mt. Morgan was looked upon by Donald Gordon, its discoverer, as a great silver mine. Philosopher Smith was disappointed when he heard that Mt. Bischoff ore did not contain silver but tin. Mt. Lyell was worked as a gold mine, while it is reported that the ore which was discovered when the Great Boulder mine was floated was almost worthless. Assays were made of the Broken Hill outcrop for tin, but with negative results. Other prospectors had climbed to the summit of the massive metallic-looking outcrop, had napped off pieces only to look upon them as worthless. After Rasp had pegged out what is now known as Block 12, he took two contractors on the station in with him. These were Messrs. Poole and James. The manager of the station, Mr. Geo. McCulloch, was then informed of the great tin deposit, and blocks 13, 14, and 15 were pegged out, and subsequently 10 and 11 on the south end, and 16 on the north. Thus, with the exception of the Central and the South and one or two minor mines on the north, the whole of the valuable leases were included in the original pegging out. A small syndicate was formed, consisting of station employes named George McCulloch, George Urquhart, Charles Rasp, James Poole, Philip Charley, David James, George M. Lind. Each member paid in £70, and it was decided to sink a shaft. By some ill-chance the shaft was put down on about the poorest part of the lode, and only low grade carbonate of lead was discovered on the surface, while the shaft material was unpayable. The money raised was spent. Lind sold his interest to McCulloch and Rasp. Urquhart next retired. It was decided to raise the number in the syndicate to fourteen, in order to get some more money. Soon after this was accomplished, towards the end of 1884, chlorides were discovered in the shaft at 100 feet, and rich ore found in other parts of the mine. Just before these discoveries the syndicate shares were at a discount. The romantic tale is told of how McCulloch, finding three syndicate shares too risky to hold, played euchre with Cox, at Mt. Gipps Station, to see whether £100 or £150 should be paid for a share. McCulloch won, and lost a share that six years afterwards would have returned him nearly a million times as much. Another well-known holder bought three shares for £320; he sold one for what it cost him, another for £200, the third share he retained—each share was worth, with bonuses and dividends, £1,250,000 at the end of six years. What must have been the chagrin of Lind and Urquhart, who each might have become multi-millionaires at the rate of over £1100 per day by simply holding their shares.

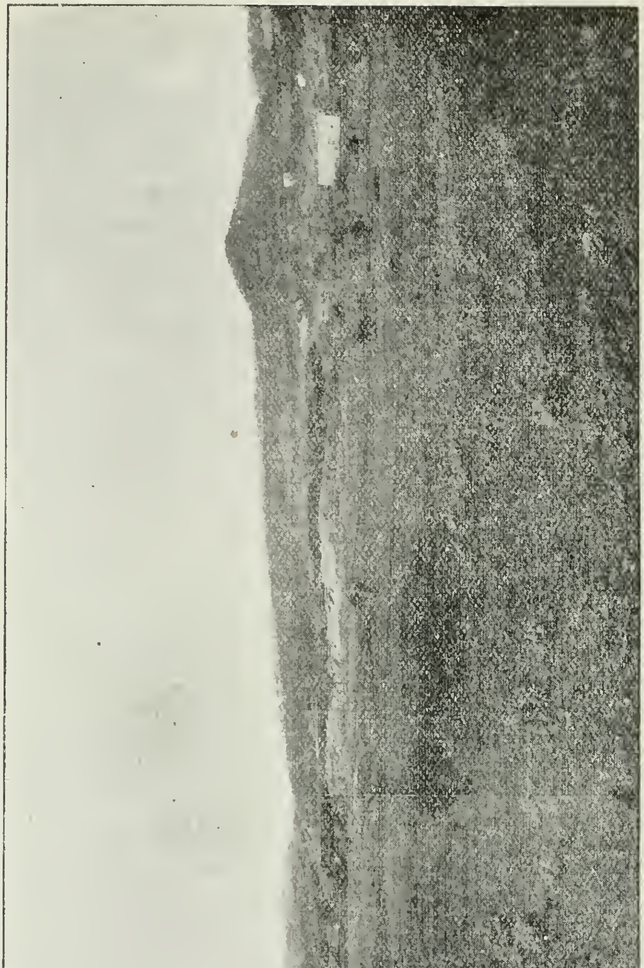
Mr. Wm. Jamieson was manager in those early days, and an aboriginal, named Harry, employed on the mine, discovered chlorides in the kaolin. In a short time 46 tons of kaolin were treated, and yielded a ton of silver. This was exhibited in Melbourne. The magnitude of the mine led to the subdivision of the shares, and on the 10th August, 1885, the present Broken Hill Proprietary Co. was floated. The company was to consist of 16,000 shares of £20 per share, the fourteen shareholders were to receive 1000 shares, each paid up to £19, while the 2000 shares were offered to the public at £9 each, and considered to be paid up to £19. There was thus a liability of £1 per share on the whole of the shares issued. Three thousand pounds were to be paid to the original shareholders for expenses previously incurred. Three and a half years later the immense value of the property was becoming apparent, and the shares in the company were increased from 16,000 to 160,000, each £20 share being subdivided into 10 shares of £2 each, and considered paid up to £1 18s.; yet the rise in the value of shares within another year caused the company to increase its scrip issue from 160,000 to 800,000, and proportionally reduce their value to 8s.; also to issue 160,000 new shares, all of which were paid up.

In 1886 smelting operations were started, and at the end of that year four 30-ton smelters were erected, and Mr. H. H. Schlapp, from Colorado, was appointed metallurgist; Mr. S. R. Wilson being mining manager. In 1887, Block 14 was sold. The new company had 100,000 shares of £5 each. These were issued as paid up to £4 10s. The shareholders in the parent company were given 96,000 shares, and 4000 were subscribed for by the public to provide the working capital.

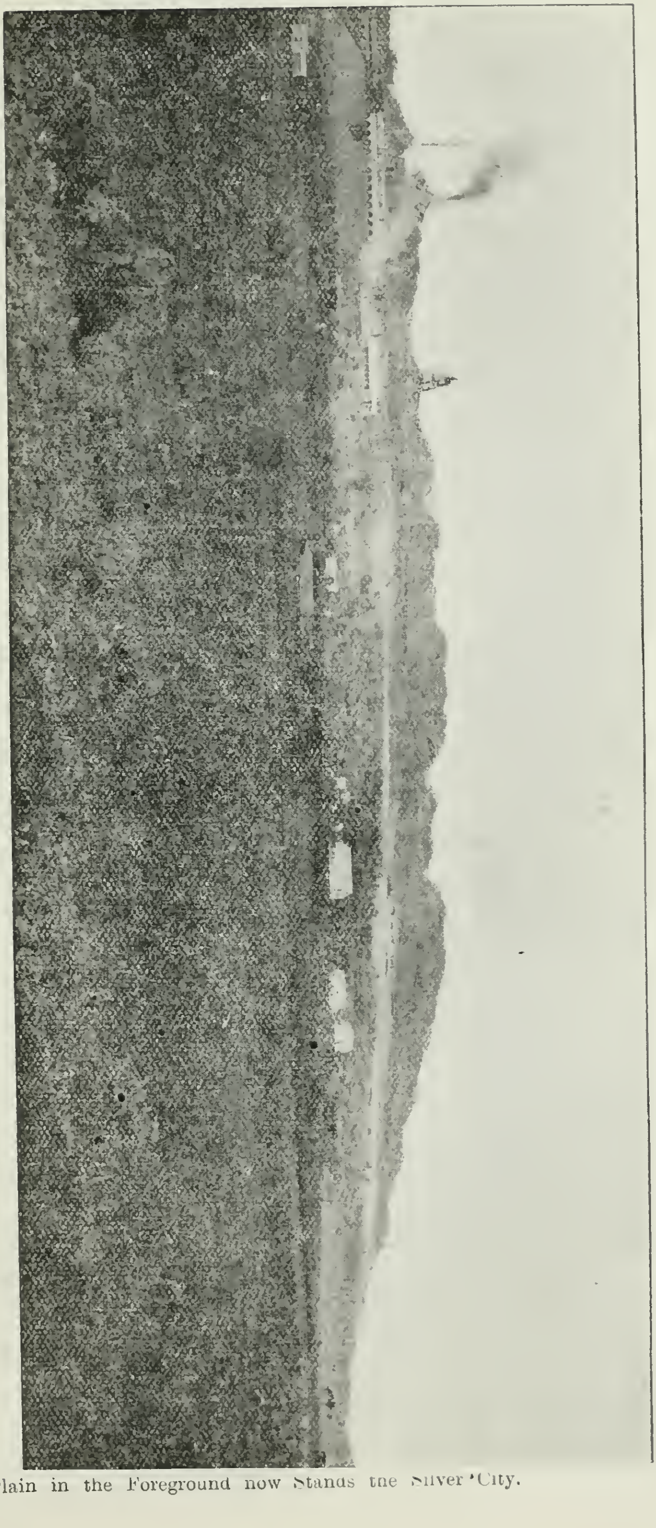
During the same year Blocks 15 and 16 were sold to the British Company. The shareholders of the parent company received shares to the value of £400,000, and £576,000 in cash. In the following year Block 10 was sold; of the 100,000 shares issued, the parent company received 96,000 shares, valued at £10 each, and 4000 shares were issued to the public to provide the working capital. These shares were issued as paid up to £9 10s., the money actually contributed only being equal to 3s. per share, or £15,000 in all.

The parent company thus held blocks 11, 12, and 13, or a distance of 60 chains along the line of lode, and 20 chains across it. When the Silverton boom was at its height, the Government of South Australia ran a railway line from Petersburg to the border, and it was well that the Broken Hill lode turned out to be a big producer, for Silverton and the adjoining fields are practically deserted. A line connecting the South Australian line with Broken Hill, known as the Silverton Tramway, carries practically all the material sent for some hundred miles along the South Australian line.

It is very difficult to get all the figures relating to yields and values from the various mines on the line of lode, but the following have all paid dividends:—Starting on the line of lode from the south end, Blocks 5 and 6, or the South Blocks, 7 and 8 Broken Hill South; 9, the Central, owned by the Sulphide Corporation; Block 10; Blocks 11, 12, and 13 Proprietary; Block 14; Blocks 15 and 16, The British; 17, North Broken; 39, Junction; 40, Junction North. These mines extend along the line of lode for about 2½ miles, and at some levels the lode has been worked for great distances across from one end to the other. This single line of lode has, up to the present, produced £50,000,000 worth of gold, silver, and lead—an amount nearly equal to a fifth of the value of the gold



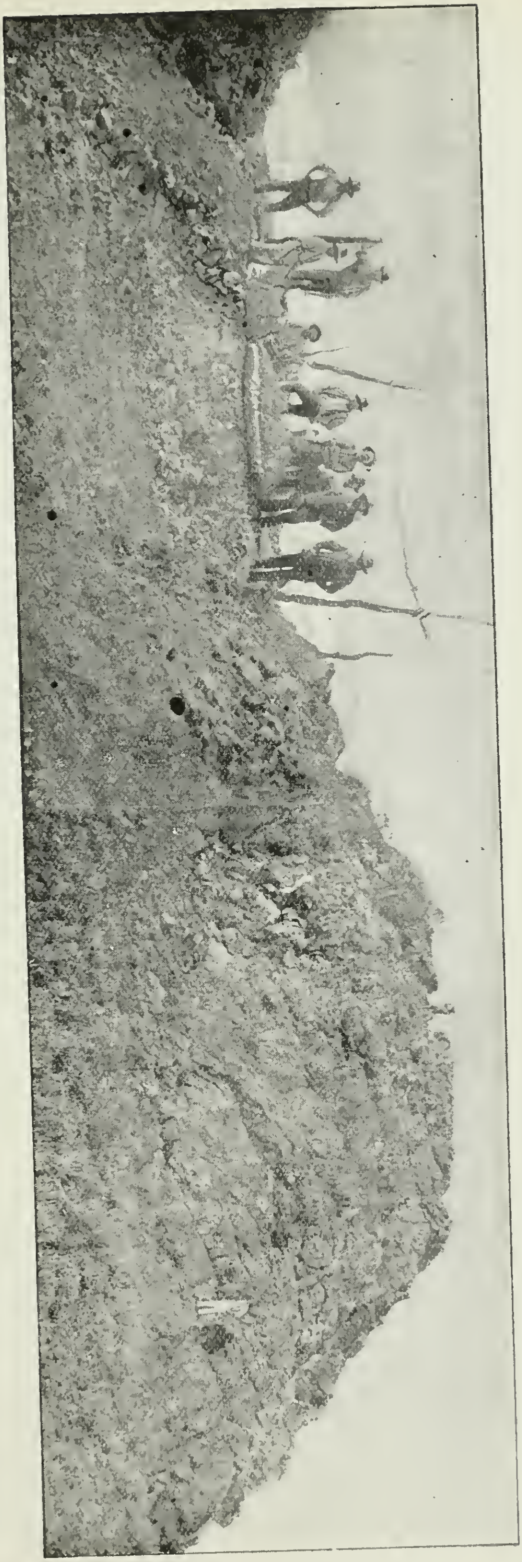
The Beginnings of Broken Hill. On the



Plain in the Foreground now Stands the Silver City.

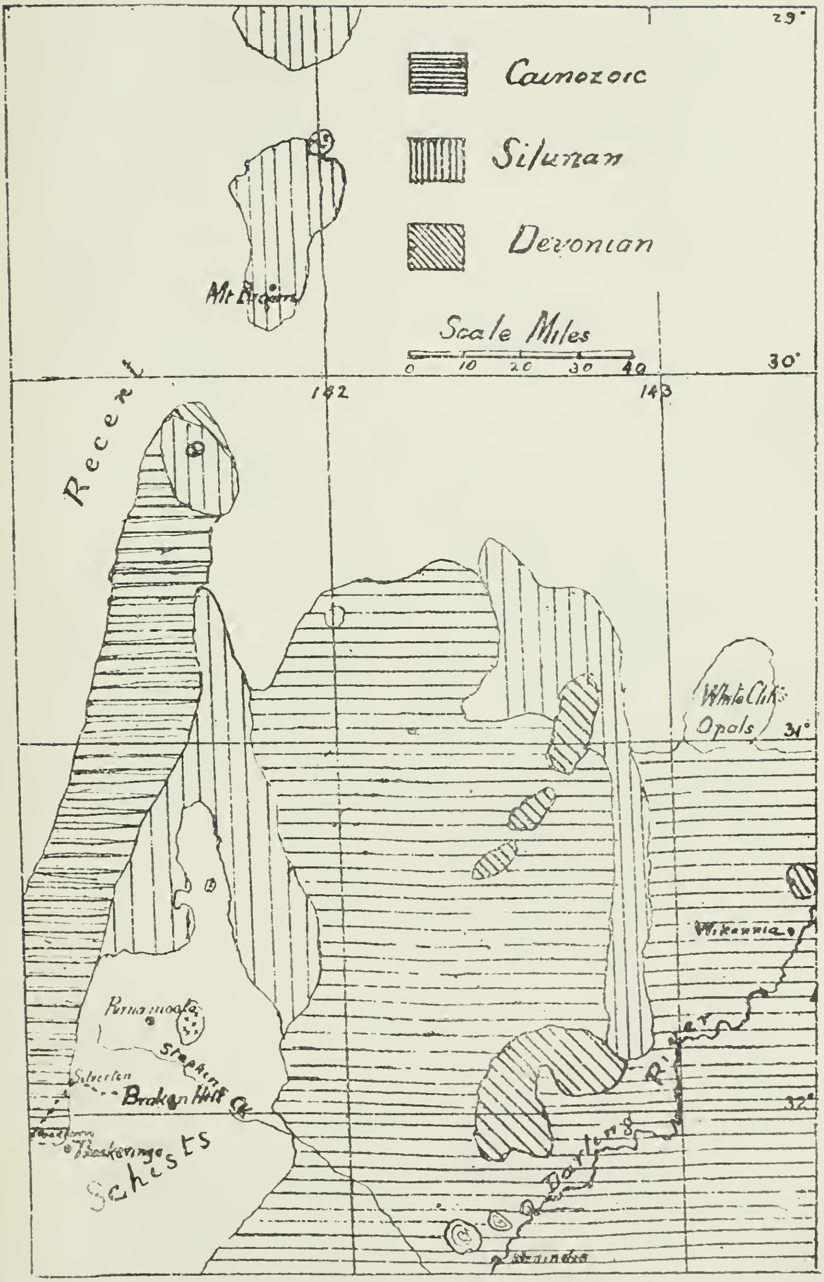
won in Victoria, nearly equal to that won in New South Wales, or Queensland, and greater than the total value of the gold won, up to the present, in Western Australia. The dividends paid have amounted to more than £12,000,000, and yet, in the 20 years the mines have been producing, no shaft is down 2000 feet, and the workings producing ore in the parent mine are not below 850 feet, and still it produces one-twelfth of the lead obtained in the world annually.

The height of Broken Hill above sea-level is between one and two thousand feet, but this elevation seems to have but little influence on the climate. In summer the temperature rises to over 100 degrees in the shade for days together, while, in winter, the days are clear and the air dry. The objectionable element is dust. Sometimes it is present in minute particles, sometimes the heavens are darkened for hours, and even days together, but it is never absent. On a fine, sunny day a dusky red horizon may be seen all round. The hills are bare and warm-tinted, while well-known objects have distance conferred on them by being viewed through the fine red haze of desert dust. Only calm days can be enjoyed on this field; when the wind blows, clouds of dust are swept across the desert for hundreds of miles, coarse sand is driven with such violence that all the polished glass one sees lying around the town is frosted and fretted with sand and grit. Paper and light rubbish is blown right out of the town, and litters the country for miles around. Before the prospector came, stunted mulga bushes covered the slopes, and sage-green tufts of saltbush hid the red-brown soil. Now not one shrub is to be seen for miles round the Hill, and the scattered tufts of saltbush left alone redeems the place from being a desert. Here and there, in the distance, one may see a winding track of stunted gums, showing the dry bed of one of the creeks where water sometimes runs. The rainfall of the place is exceedingly variable; as much as 16 inches fell in 1889, but in the previous year only 3 inches fell. Taken altogether, it is a dry place, for the evaporation per annum would need to be expressed in feet, instead of inches. Water has been conserved by building the Stephen's Creek reservoir; but it is a pity, considering the necessity of a plentiful supply of clear water, that the Darling River was not tapped. Broken Hill is a fairly old town now, yet owing to the imminent danger of shortage of water, the trees and shrubs planted years ago had to be neglected, and only the hardy pepper trees have survived. Looking down on the town from the hill, one sees fine buildings and streets, but the background of low hills and bare ridges, almost destitute of vegetation, makes the place look desolate, and the bright green of the umbrageous pepper tree is the only relief the eye finds in all the landscape. To add to the drawbacks of the place, mosquitoes have increased and multiplied, and are always buzzing about at night in search of human blood, and summer flies are ravenous. The bare houses, with their baked backyards, and the hopeless attempts made to grow a few common flowers, make the place a very unenviable one to live in, in spite of the brave fronts of many of the buildings and shops. When mining is done, unless some extensive irrigation and water conservation scheme is introduced, the place will revert to desert. In no town in Australia can one see so many men propped up against walls, or aimlessly wandering about: the women rapidly age, and even the young



The First Shaft on Broken Hill.

children have old faces. Beyond a man's daily work there is nothing for him to do, while families only exist in such a place as this. The days of lead poisoning are practically over. When the smelters were at work tons of metallic lead were daily discharged into the air above, only to be precipitated as poisonous compounds on the country around. Fowls, which picked up the surface soil, and cats, who cleansed their fur, soon succumbed, while many children were leaded in this unhealthy town. Miners, working amongst the dust of carbonate ores, became leaded, and even now there are many human wrecks left as relics of the boom days of Broken Hill. These days are practically over; leading, typhoid, and similar diseases have not vanished, yet the mortality has been greatly reduced—with more perfect sanitary arrangements there is room for a still lessened death rate; but on the whole, the town would be uninhabited were it not for a strip of country a couple of miles long, by a few hundred feet wide, which has produced more wealth for the depth worked than any similar strip in Australia. Probably, in another generation, the end of mining in the district will be as far off as it is at present.

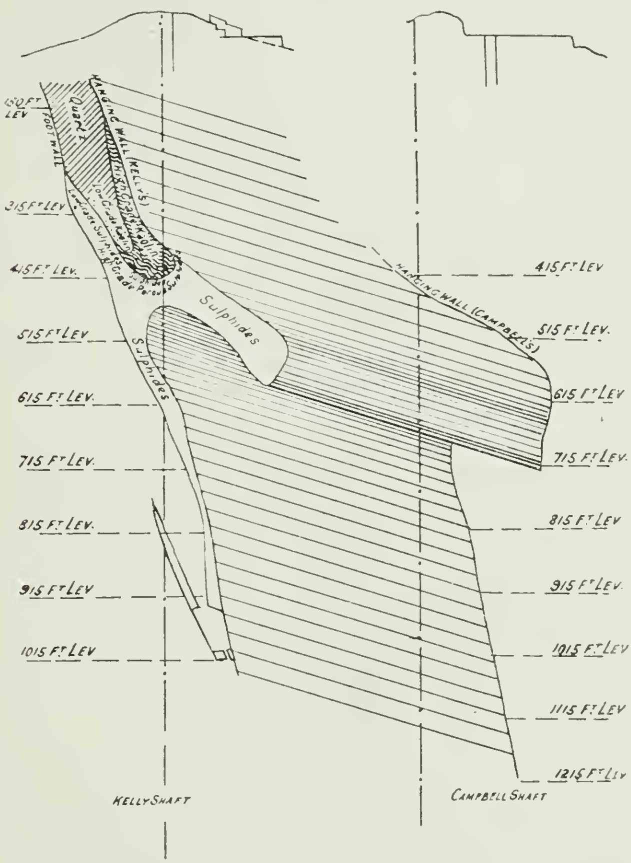


Locality and Geologic Plan of Broken Hill.

Geology of Broken Hill and Work Done.

After passing the border line between South Australia and New South Wales, the tramway runs over the southern extremity of a great level stretch of country, known as the Seventy-Mile Plain. This averages about 20 miles wide and 120 miles in length. This plain is of comparatively modern origin, consisting, as it does, of clays, gravels, and other rocks derived from the wearing down of the underlying older rocks. It may coincide with some of the Victorian gold leads. On the eastern side of this the older rocks outcrop over a wedge-shaped area, the width of the base extending from Silverton, eastward, for about 30 miles, and the apex of the wedge being about 80 miles to the north of this line. On the eastern side of this again lies a wide area made up of the same tertiary or post tertiary drifts and clays which make up the Seventy-Mile Plain.

The older rocks, at the lower end of the wedge-shaped area, are made up of crystalline schists, penetrated by diorite, and having considerable outcrops of granite, while at the upper, slates, sandstones, and limestones, all very much altered, prevail. Near the base line mentioned, some of the hollows between some of the intervening ridges are filled with a nodular dolomitic limestone, sometimes to the depth of several feet. The country for, at least, 30 miles wide by 40 miles in length, contains numerous lodes, carrying lead, zinc, and silver, while at the northern extremity of the schist country, near Mt. Euriowie, stanniferous deposits occur. In addition to these minerals, platinum has been found near Stephen's Creek reservoir; bismuth, to the south of this, and numerous copper lodes to the south of Broken Hill. The lodes first discovered starting from Thackeringa, and going to Silverton, Umberumberka, Apollyon Valley, and Purnamoota, follow a line a few points to the east of north for nearly 40 miles. Broken Hill is about 15 miles to the east of this line, but the general trend of the lodes is the same. It is evident there is a large mineral field here, but an ordinary lode is so puny against the massive lode at Broken Hill that it can receive but scant attention. Broken Hill itself is a ridge about two miles long and 200 feet in height; it dies away at both ends, and has a plain on each side. The rocks, of which it is made up, consist of highly altered slates and garnet sandstones. Gneiss and micaceous schists also outcrop abundantly near the southern extremity. The strike of these rocks is parallel to the length of the hill, and the hill itself is an anticline, the dip of the rocks practically following the sides of the hill. The lode follows the crown of the hill. The surface consisted of a massive black outcrop of manganic ironstone, which reached its greatest height over Block 12. This varied in width from 20 to 100 feet. Almost the whole of this was, more or less, impregnated with cerussite, embolite and iodyrite. Beneath this iron cap occurred the oxidised ores, which were known at the Hill as the carbonate ores, dry, high and low grade ores, respectively. The first always contained cerussite, but included was a considerable portion of silicate of alumina, and also oxides of iron and manganese: the dry, low grade ore had much the same composition, but less lead, and from 5 to 40oz. silver per ton. The dry, high grade ore consisted of kaolin, containing garnet and quartz, carrying native silver, chlorides, bromides, and iodides, and running up to 300oz. silver per ton. Below this level came the friable sulphides. These consisted of somewhat loose aggregates of galena, blende, and gangue, consisting of silica and garnets. The composition of this was somewhat variable, but, as a rule, the lead contents were in excess of the



Projection of the Lode in Block 10.

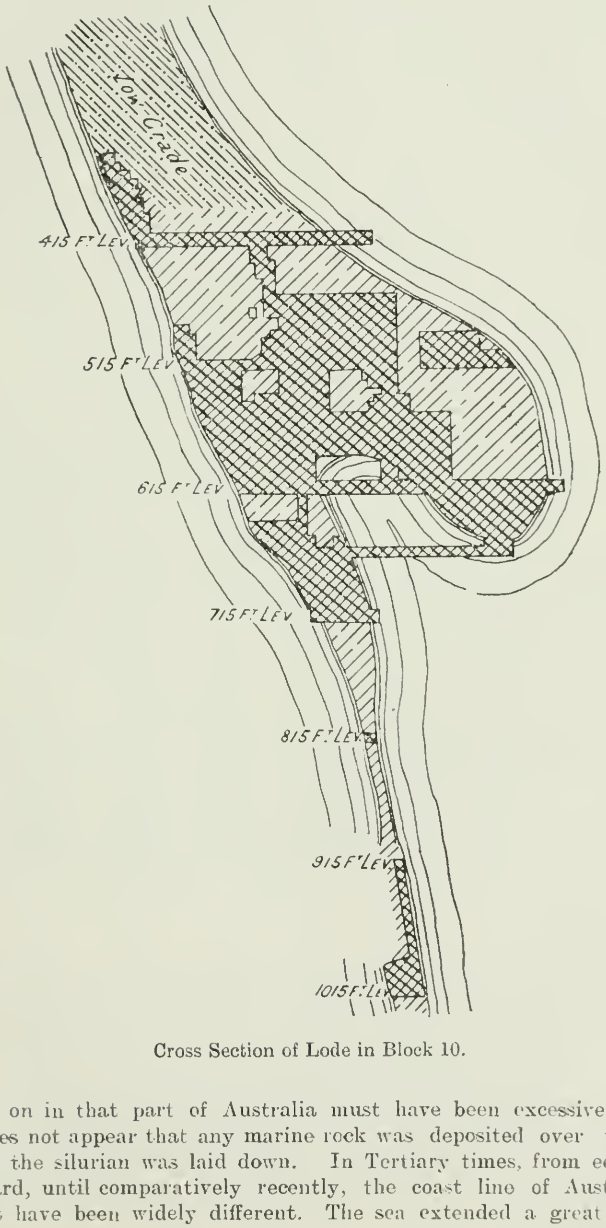
zinc. Below this came the compact sulphides, or an intimately mixed mass of galena and blende, with a gangue of quartz rhodonite and garnet. In the early days of the sulphides this was spoken of as a 25 per cent. proposition, or 25 per cent. lead, 25 per cent. zinc, and 25oz. silver per ton, but the grade of the ore has gone down until now it is more nearly a 15 per cent. proposition.

It was found that, after sinking for a certain distance, the massive lode at the outcrop bifurcated: men from the Bendigo (Vic.)

goldfield, and amongst them, Mr. Hebbard, the present manager of the Central mine, pronounced the lode to be a saddle, similar to the saddle reefs at Bendigo. This idea was ridiculed by the most of the managers on the field, but on a geological survey being made by Mr. J. B. Jaquet. Mr. Pittman, in April, 1892, pronounced it, on the evidence then available, to be a saddle lode, and his contention is upheld by Mr. Jaquet. The latter's report is an exhaustive ore on the work up to that date, but since it was published nearly ten years ago, there is no doubt a great deal more information could be included in an up-to-date report. The theory is commonly accepted now, but it is quite possible that, while the upper portion of the lode agrees in many respects with a saddle reef, when more work is done it may be found that the filling of a single hollow in an anticline. by a lode, does not mean that this lode will possess any other point in common with the series of saddle reefs so well known at Bendigo. The upper configuration has many points of resemblance with saddle reefs. For instance, these widen out over the arch to many times the thickness of the legs; in these caps or crowns of the saddle the large deposits of ore occur. A longitudinal section of the saddle follows the top of the anticline along the strike in a wavy line, the wave extending over great distances. This departure from the horizontal is known as the pitch of the reef, and it would seem as if the Broken Hill lode had a pitch from Block 14 both ways, that is the saddle falls northward, towards the Junction mine, and to the south, towards Blocks 13, 12, 10, the Central and the South mines.

From an inspection of the plans of the various mines I do not consider that the Broken Hill lode is a true saddle, but that it is a lode, the materials of which it is composed coming from below and filling cavities which opened up at the time of its formation. The undoubted saddle, so well described by Mr. Jaquet, is simply due to the cavity between the upper strata of the anticline being filled laterally from a nearly vertical vein. When saddle reefs, as ordinarily understood, occur, the space left between two strata at an anticline is filled with lode material, giving the cross section a crescent-shaped appearance. Both legs thin out before the syncline is reached, one leg generally descending to a greater depth than the other. In no case do they pass through the strata and descend as ordinary reefs; and, although the cap may extend upwards for some distance, yet it does not break through any extent of the upper strata. The saddle reefs are found one below the other, yet wholly unconnected, as far as veins or lode matter is concerned. Now, at Broken Hill, the so-called western leg of the lode has usually abruptly terminated in a rounded stump, while the eastern leg has persistently gone down almost vertically. The excellent diagrams of Block 10 mine, shown in a paper on "Reminiscences of Broken Hill," by Captain Warren, abundantly prove this. The shoots of ore at this mine pitch southwards, at an angle with the horizon of about 30 degrees.

As a consequence of the belief in a different type of lode, I do not agree with Mr. Jaquet that the formation of this lode was due to any process of lateral secretion, except in a subsidiary way. The present Broken Hill mine, at the time the ore was deposited in it, was probably covered with some hundreds, if not thousands, of feet of rock. Mineral solutions, at some period, circulated freely through a large belt of country, some depositing their contents in suitable receptacles. It is improbable that any reached the then existing surface. This most probably occurred at the time of the alteration of the sedimentary rocks into metamorphic ones. Assuming that time to be about the Devonian, the amount of erosion which has

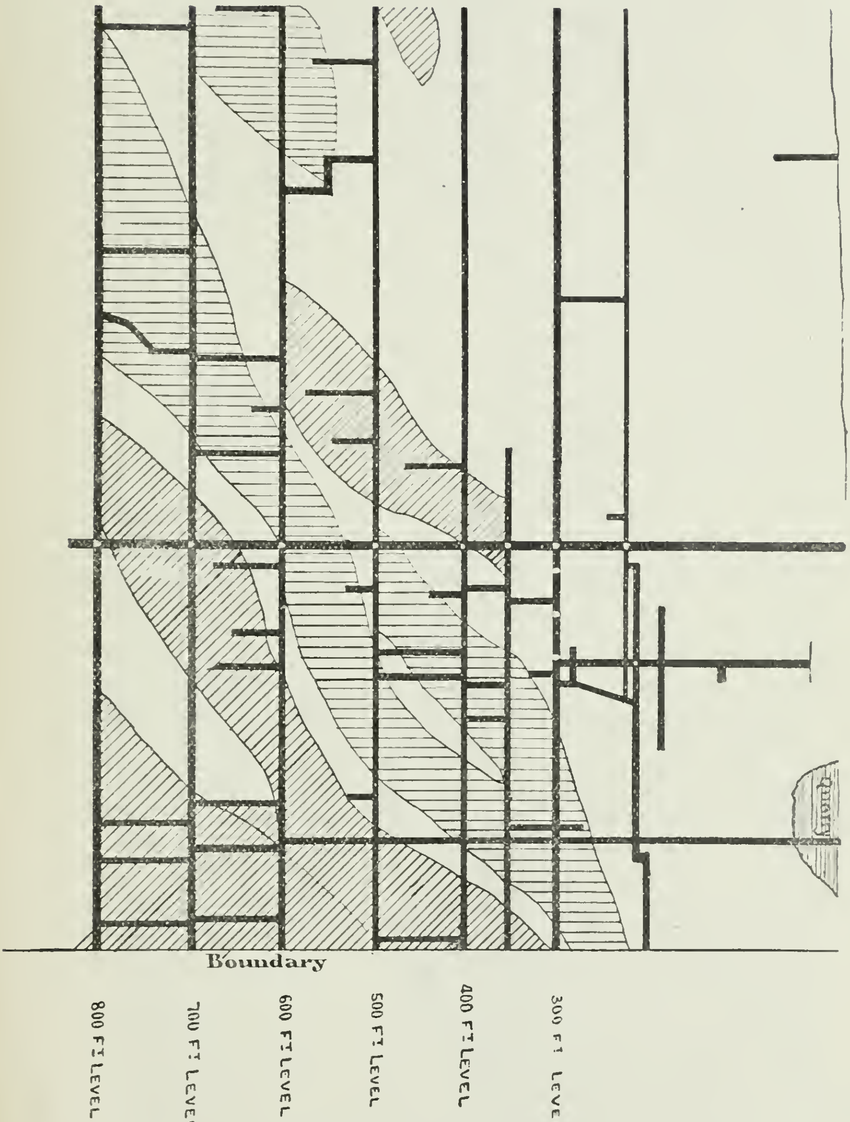


Cross Section of Lode in Block 10.

gone on in that part of Australia must have been excessive, for it does not appear that any marine rock was deposited over these since the silurian was laid down. In Tertiary times, from eocene onward, until comparatively recently, the coast line of Australia must have been widely different. The sea extended a great way into South Australia. The north-west of Victoria, and a part of New South Wales was also submerged, so that climatic conditions at the Barrier Ranges would have been widely different from those at present, and the denudation much more pronounced. It is probable that hundreds of feet have been removed from these ranges. Since pliocene times, however, the conditions would have been similar to those at present. There does not seem to be any reason for giving a different origin to the cap of the lode than that required for most other lodes of a similar nature. The composition of the ore below gives the clue. The galena is almost pure sulphide of lead, but the blende is comparatively low in zinc, and is an isomorphous compound, consisting of sulphide of zinc, iron and manganese. Now, assuming all these to oxidise the galena becomes carbonate, sulphate, or oxide of lead, the zinc sulphide becomes sulphate or carbonate, the iron becomes sulphate or oxide, the manganese suffers a like change. As a rule, a certain amount of free sulphuric acid is also formed, moisture is attracted, and the more soluble salts creep towards the surface, as liquids will rise in blotting paper. The evaporation, near the surface, causes the action to be slowly continuous. Iron becomes converted to oxide; manganese to a stable oxide. The amount of manganese and iron present would, in some degree, measure the amount of decomposition. Once the surface cap is formed the action will go on, but as the country wears down the cap will become a mass of rugged, metallic, rock-like masses. The iron, the manganese, the lead, and the silver will all tend to remain, their compounds being fixed, but the zinc oxide or carbonate, even when they do form, are extremely soluble in acids, or decomposed by acid salts, so the tendency is for the bulk of the zinc to remain in a soluble state, and hence to be washed away. As bearing upon the rapid decomposition of sulphides exposed to air-carrying solutions, and their solution, an analysis of water from a shaft about five miles south-east from Broken Hill was made by Mr. J. C. H. Mingaye, at the Government Laboratory of New South Wales. The solution contained over a pound weight of solid substance per gallon, and contained iron, alumina, cadmium, lime, magnesia, potash, soda, as well as zinc, cadmium, copper, cobalt, nickel as sulphates and chlorides, as well as free sulphuric acid; the amount of the commercial metals present, in grains per gallon, being: — Copper, 8.40; zinc, 10.67; cobalt, 21.82; nickel, 6.71. In a rainy district it would be easy to account for the loss of the zinc, but even at Broken Hill the rainfall has been sufficient to distribute soluble salts over wide areas. The kaolin, and its ores, are secondary products, whose origin is similar to that for the others.

The length of the outcrop was about a mile and a half. The depth at which the legs of the saddle commence varies, but in all cases in the Proprietary mine, according to Mr. Jaquet's report, is less than three hundred feet, but it pitches away rapidly in Block 10 to nearly 600 feet at the boundary with the Central mine.

The depth to which the oxidised ore extended at Broken Hill varied. In the Proprietary mine, it practically ended over the No. 3 level, from Block 14 boundary, over Block 13, and the greater part of Block 12; on the south end of the latter block it extended, in places, to No. 4 level, while, in Block 11, in one place it went down to nearly No. 5 level. Owing mainly to the oxidation of the

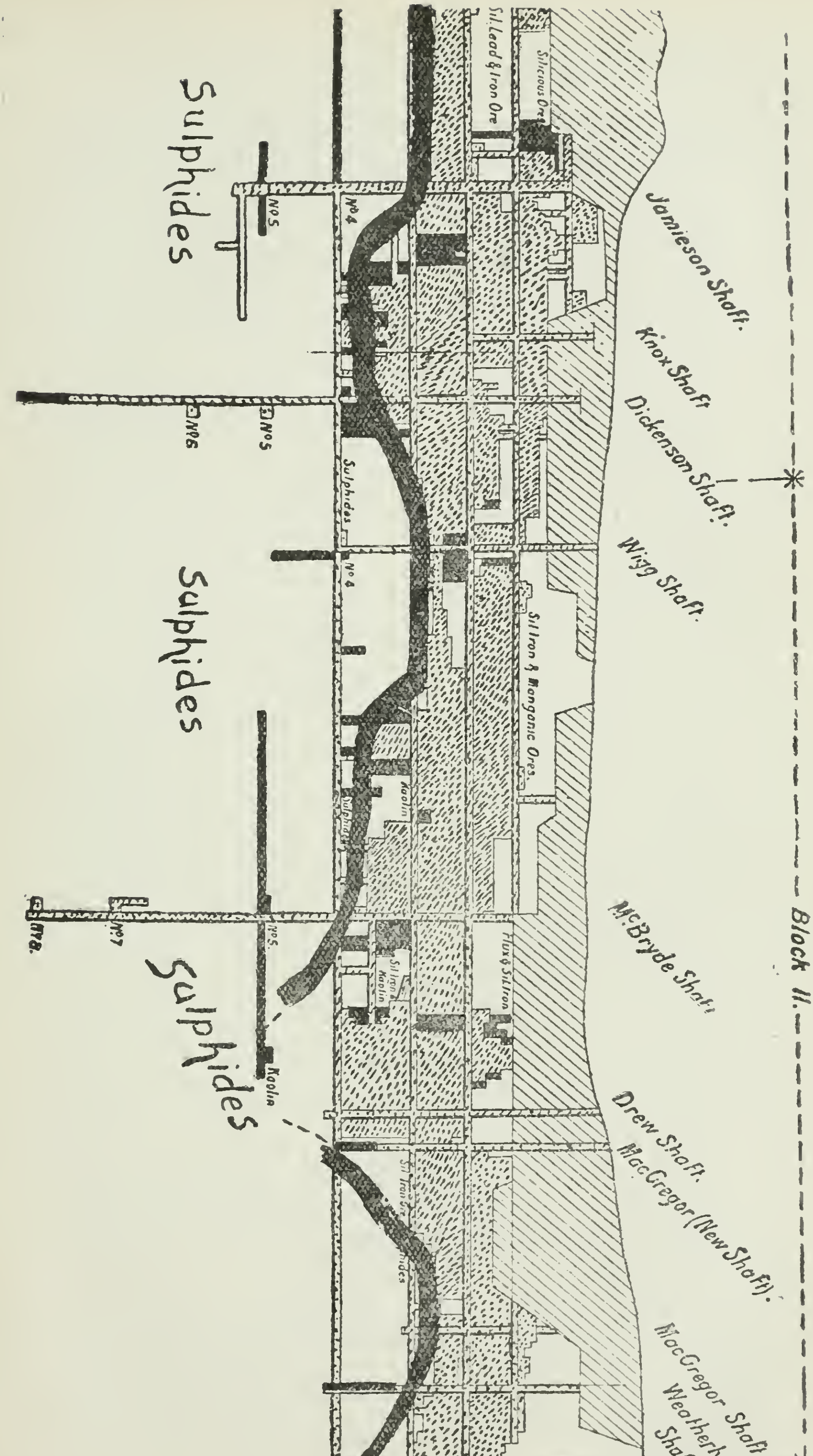


Longitudinal Section of Block 10.

metallic contents, and the removal of the more soluble ones, a great concentration of silver occurred in places, while, in others, almost pure carbonate of lead remained. The upper layer of sulphides also were often exceedingly rich, but when the solid unaltered sulphides were struck the values have decreased enormously, and, so far as I can learn, there are no continuous rich veins of argentiferous material. The whole lode is made up of platy galena and blende in a gangue, consisting mainly of rhodonite, garnet, and quartz. In some mines the rhodonite and garnet is almost absent; in others, very little free quartz occurs, but in all cases the metallic minerals and gangue are all admixed evenly together, and not in the banded structures common on other fields. Gold, also, invariably occurs, and in the case of the Proprietary mine, is profitably separated from the silver. On visiting the Hill, some years ago, I was shown, by Mr. Stewart, a fine specimen of auriferous ironstone, showing gold freely, which had just come out of the upper workings of the Proprietary mine. The gold appeared to be exceedingly pure, and free from silver.

WORK DONE.

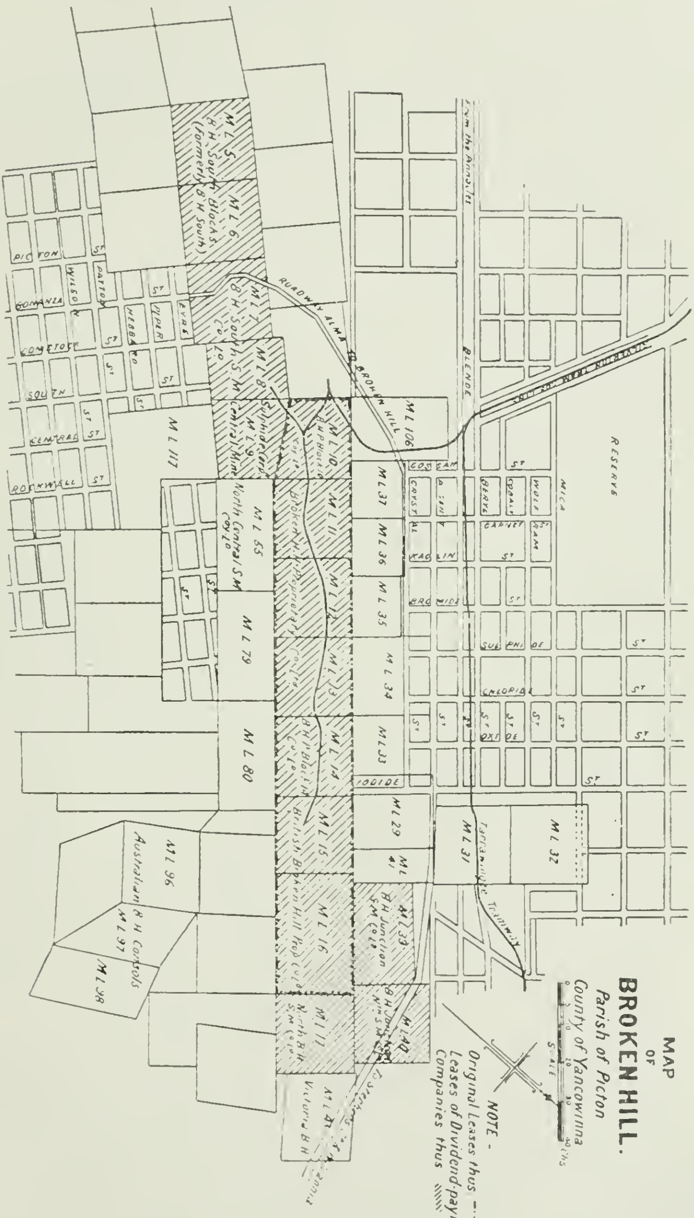
With regard to the companies on the field, it is a somewhat difficult matter to obtain information from their reports alone. The only one which pretends to publish plans of their workings is the Proprietary Company. In this a longitudinal section is given, which looks very well with its greens and reds, but which is about of as much use as the remains of the shafts marked on top of it. A plan of the various levels is also given, but this gives almost as little information as the section. It can be only inferred that the cross sections, at various places, would not have looked so well. From the plan it would appear that the oxidised ore has all been removed, with the exception of a very small portion in Block 11, and a small portion near Block 14, by the open-cut, from the 200 to the 300 feet level. The sulphides have been worked from end to end at the 400 feet level, but a block of ore up to 200 feet wide, and 900 feet in length, is now being broken out in Block 13. At the 500 feet level sulphides have been worked from Block 10 boundary to the middle of Block 13; a block of ore, 1000 feet in length, 40 feet in height, and about 40 feet wide, is being broken at this level; at this level, also, an indeterminate quantity of ore is penned in by the barriers which bound the fire still smouldering. At the 650 feet level ore has been proved to exist for 1600 feet from Block 10 boundary. The ore has only been worked over a portion of Block 11, the width being about 30 feet. At the 800 feet level a drive of 1400 feet has been put in, and the sulphides are being mined under the centre of Block 11. The lode here is also about 30 feet wide. Prospecting revealed a body of ore of considerable width to the east of this. At 1000 feet a level is being driven under the ore bodies proved in the level above. If the prospecting done is a guide to the presence or absence of ore bodies, then it is evident that the so-called leg of the saddle goes down narly vertically, or with a slight underlay to the west, and that the western leg has cut out. It is also evident that the pitch of the ore is carrying it rapidly out of the company's ground into Block 10. The ore in sight, as indicated by actual blocking out operations, is said to be over 4 million tons, or sufficient to keep the present plant going for another seven years. So far as quantity is concerned, there is a



The Sulphide Lines.—Above, are the Oxidised Ores. Below the Black Lines, the Sulphide great probability of these ore reserves being materially increased, but it must not be assumed that the quality is equal to the quantity. At the present time, a very small fraction of the value of the ore is profit, and it needs but a small diminution in the metal value to make these vast bodies of ore unpayable, with present methods of working. Were it not for the prospects of better metallurgical work, the outlook would be a precarious one for shareholders. The ore bodies, to the north of the big mine, have been worked with good results in the upper levels, but the same cannot be said when the low grade ores, at low levels, were struck; on the south are mines which have worked for many years with success, notably the Block 10, but absolutely no useful information can be gained from the general manager's report. It can only be assumed that considerable ore bodies exist to justify the erection of a concentration plant, based on the old wasteful methods of working. The next mine, the Central, contains immense bodies of sulphide ores; the ore in sight is said to be sufficient to keep the present plant going for 20 years. In this case the ore was covered with 300 feet of rock, since it dipped rapidly through the Block 10 ground. The South adjoins the Central, and, in this case, the ore reserves are more than a million tons above the 800 feet level, or more than a ten years' supply for the present plant. The mines to the north of the parent one are—first—Block 14. It is shut down at present, and is probably the one worked to the least depth on the field. Judging from an inspection of the company's plans, there is room for many other ore bodies. Unfortunately, the grade of the ore is a shade too low to be payable with present methods, so that the company is doing the right thing in endeavoring to improve metallurgical methods before rooting out ore, and losing half its value. The British mine, Block 15 and 16, comes next. This is now being worked at a slight profit, but judging from the banging and clanging of its stone-breakers, they must be running empty half their time. This ground, like Block 14, is only worked to about 600 feet. Extensive low grade bodies of ore have been discovered, and prospecting operations, by means of shaft sinking and dimond drill boring, are now going on. The ore treated at present is low grade, running about 16 per cent. of lead and zinc, and 11oz. silver per ton. The North Broken is also shut down, yet it has large ore bodies proved, and is supposed to have a quarter of a million tons of low grade ore in sight, the proposition being about 15 to 20 for lead and zinc, and 10oz. silver. There are two ore bodies, one with silica as gangue, the other with rhodonite.

The Junction adjoins the Block 16 of the British on its north-western boundary. This company, too, is marking time, but there is a chance of discoveries of large ore bodies still. The mine is by no means worked out. The ore at present in sight is mostly low grade.

The Junction North joins the Junction on its north-western boundary, and the North Broken on its north-eastern. Its shaft is down over 1000 feet. A large body of ore, running up to 15 per cent. lead, and up to 10 per cent. zinc, with 10oz. silver, remains to be stoped out. The gangue is mainly rhodonite. The last mine on the end of the field, practically at the north end of the ridge, is the Victoria Broken Hill. So far, there has been nothing sensational discovered, and it would seem as if the famous Broken Hill lode either has thinned out or pitched beyond the depth at present attained.



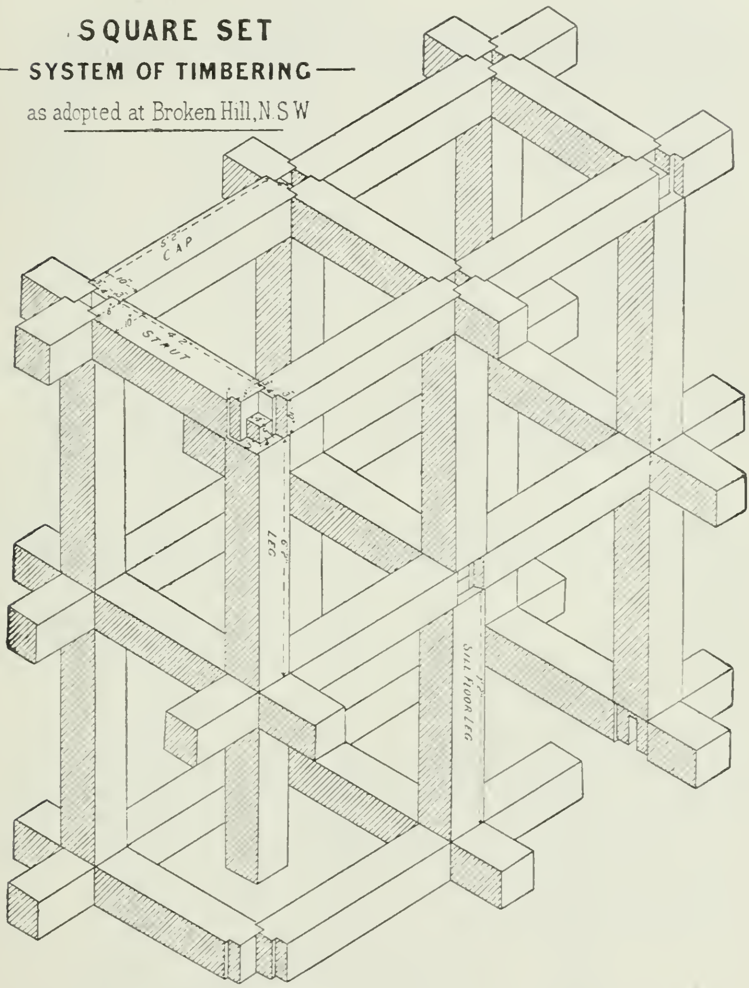
Methods of Mining.

In the early days of Broken Hill, the position was like that of the coal fields of England when coal-mining started. The quantity of coal was so great that galleries were cut through the beds and pillars of solid coal left. It was considered that these would never be needed. The "pillar and stall method," so started, has been continued, but the pillars are also taken out as well as the stalls. Although, at Broken Hill, it was known that the bulk of the standing ore would have to be removed, the question of systematic mining was not seriously considered in the early days, and when it was it cannot be said to have been particularly creditable to most of those concerned. The square set system was introduced from America. Oregon logs were sawn down to a section 10 inches square; and the long timbers sent up to the mine were then cut and notched into the sizes shown in the illustrations . The method of building this up from wall to wall, and to the stopes overhead, was exceedingly simple. Passes, shoots, and gangways were easily constructed, and the whole of the freshly-timbered stope was as neat a form of support as could be wished. The accompanying illustration shows the various dimensions for a comparatively wide lode. It seems hard to believe that any faith should be put in the stability of this system of timbering alone, for, with any diagonal pressure, it is bound to collapse like a house of cards. Let a single set go, and the whole structure crumples. When used over the wide lodes of Broken Hill without any filling, disaster followed disaster, so that it is now rarely used as a skeleton structure, but all the sets are filled up solidly except those required for ventilation or passage ways for mullock, ore or men. It is surprising that oregon should still be so universally used at the mines on this field ; forests of it have been already buried, and ship after ship arrives with it from the States. The cost of it is about 16s. per 100 feet super. It certainly has the advantage of being light, and very strong for compressive strains; it is easily worked, and easily handled below. Probably, these advantages have led some of the directors of the mines to state that it is essential for the square set system of mining. This, however, is not so, for Australian hardwood was used more successfully at Mt. Morgan (Q.), for a great many years. The great disadvantages of oregon are, that it is exceedingly weak, as regards crushing across the grain, and will crumple up like pulp when any great weight comes on it. In some of the mines, I saw it used for check and block supports, and as such, without filling, it is practically useless. It will not bear any great bending strain, so that when used for drives and roadways, where there is apt to be a lateral strain, the sets have to be strutted, after the manner shown in the accompanying drawing, and even then, set after set may be seen crushed in. Another serious drawback is the danger of fire. Already two serious fires occurred in the Proprietary mine, and the danger is never absent where large quantities of the timber is used. Its life in a mine ought to be as long as that of the mine, if kept dry and well ventilated, but in damp, ill-ventilated places it will perish like all soft wooded timbers. It is a great pity that more enterprise is not shown by our own timber men; some of our hardwood timbers in Victoria—notably the Woolybutt—are almost as light as oregon when dry, and of greater strength in compression: much greater as regards crushing, and very much greater as regards breaking strains.

SQUARE SET

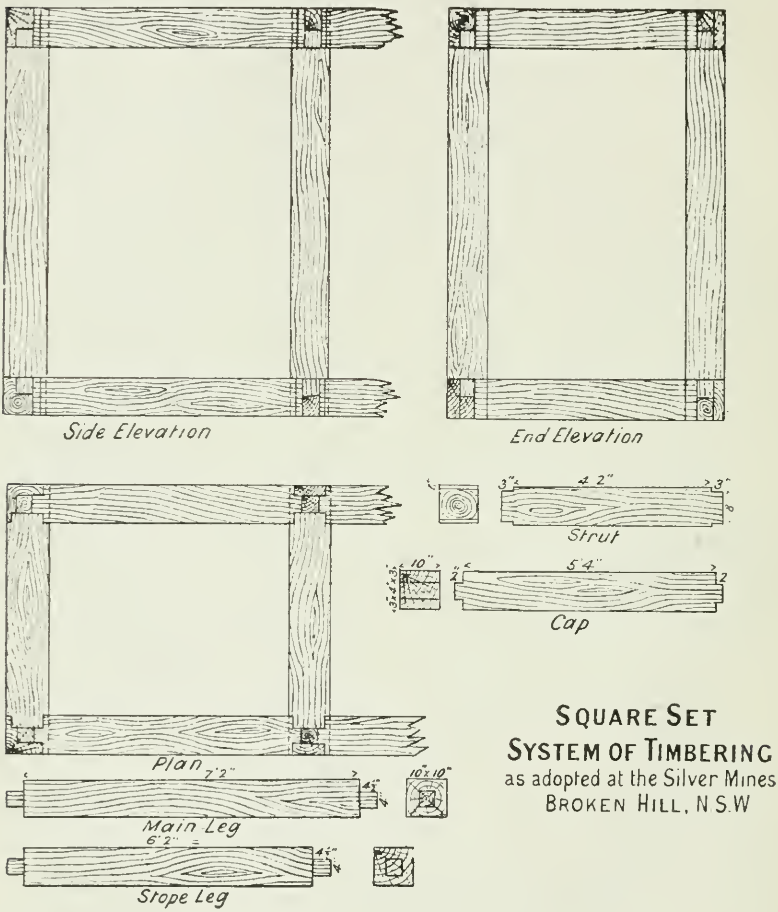
— SYSTEM OF TIMBERING —

as adopted at Broken Hill, N.S.W



The cost of timber swallowed up a large amount of profit in the early days, when the ore was ten times as rich as it is at present, and Broken Hill would be a call-paying proposition if such methods were persevered in. The first step in advance, as regards economic mining, was the start of the open-cut along the line of lode. The material taken from this was either oxidised ore or mullock: the latter went underground, and is now used as filling, whether the square set or any other systems of timbering is adopted.

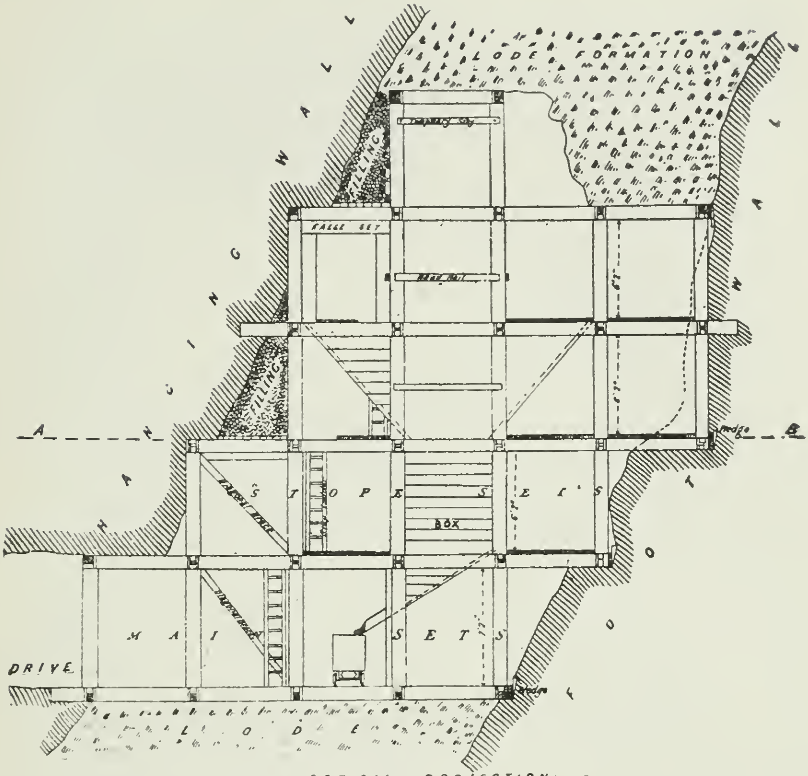
The open-cut extends over the whole length of the Proprietary mine, or for a distance of three-quarters of a mile. The width at the surface varies from 100 to 350 feet, and the depth from 200 feet over most of the workings, to 300 feet near the boundary of Block 10. Were it not for the relief of pressure afforded by taking away this enormous amount of material, some of the ore near the surface would never have been won. Had it been made part of the original



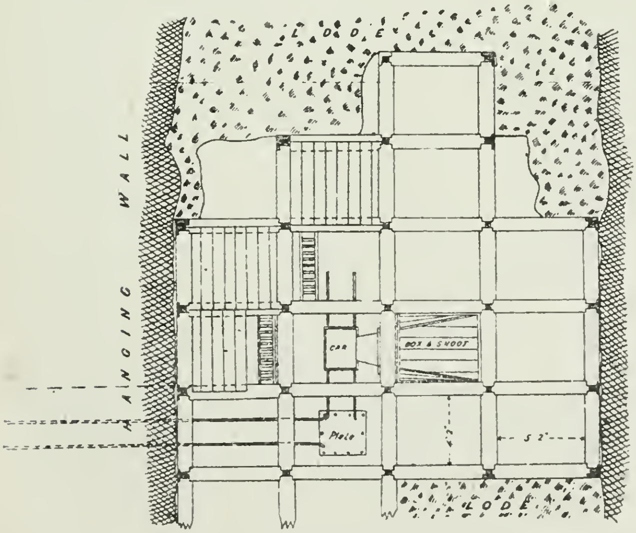
SQUARE SET SYSTEM OF TIMBERING
as adopted at the Silver Mines
BROKEN HILL, N.S.W.

Showing sizes of timbers.

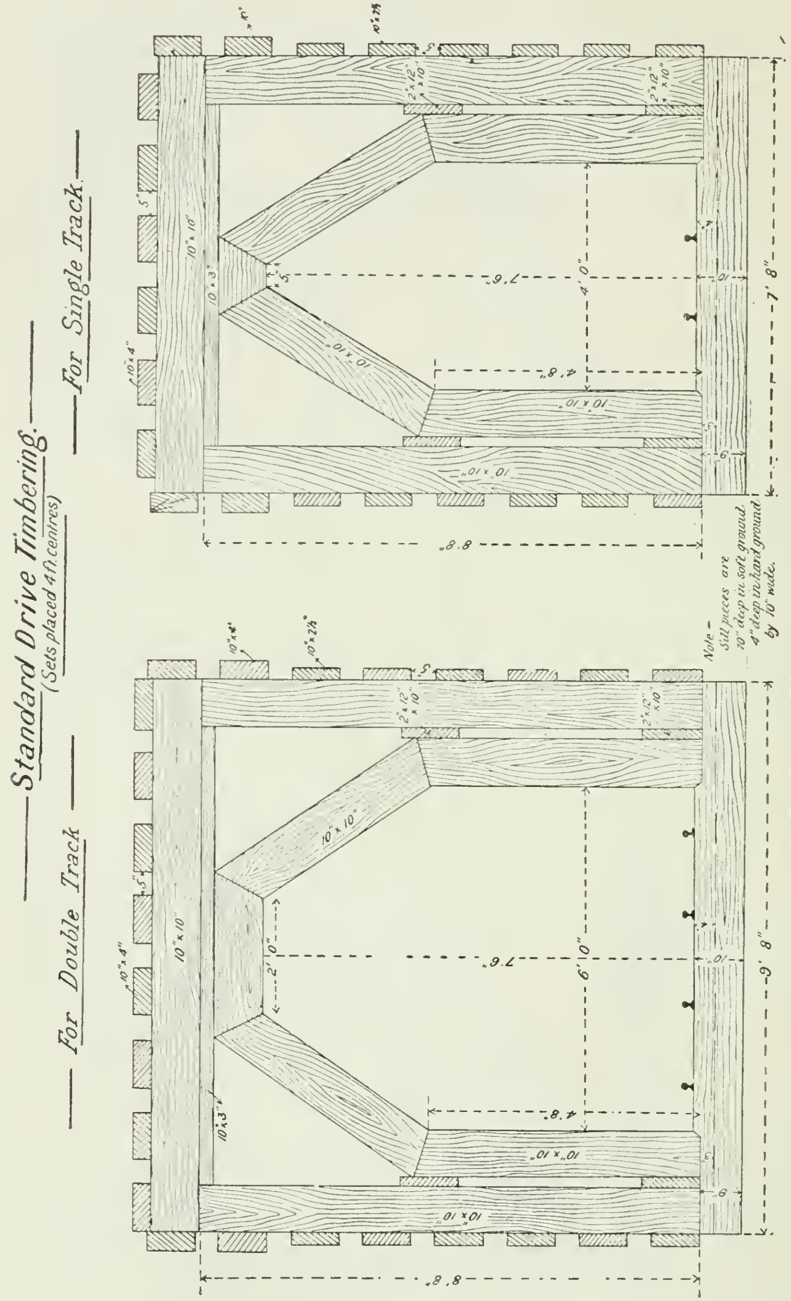
scheme of mining it enormous savings would have been effected, and safer work done. As it was, the timber recovered from the old workings was practically useless. This open-cut was not removed. as at Mt. Lyell, which, by the way, is but a small affair compared with this, but by keeping a batter depending upon the material worked. Owing to the somewhat irregular slopes and contour of the quarry, the estimation of quantities removed is a somewhat difficult problem. I was informed by Mr. Worsley, the surveyor



VERTICAL PROJECTION



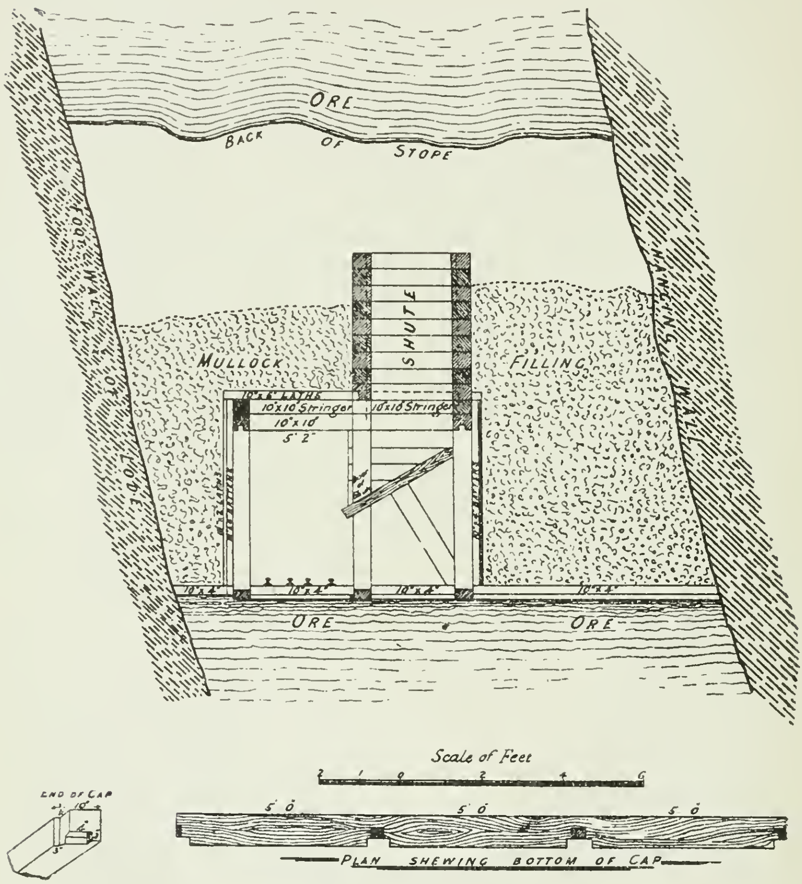
Horizontal Section A—B.
System of Timbering as adopted at Broken Hill, N.S.W.


Standard Drive Timbering.
(Sets placed 4 ft. centres)
For Double Track
For Single Track.
Note – Sill pieces are 10" deep in soft ground. 4" deep in hard ground. by 10" wide.

to the mine, that frequent sections are taken, carefully plotted, and the areas determined with a planimeter.

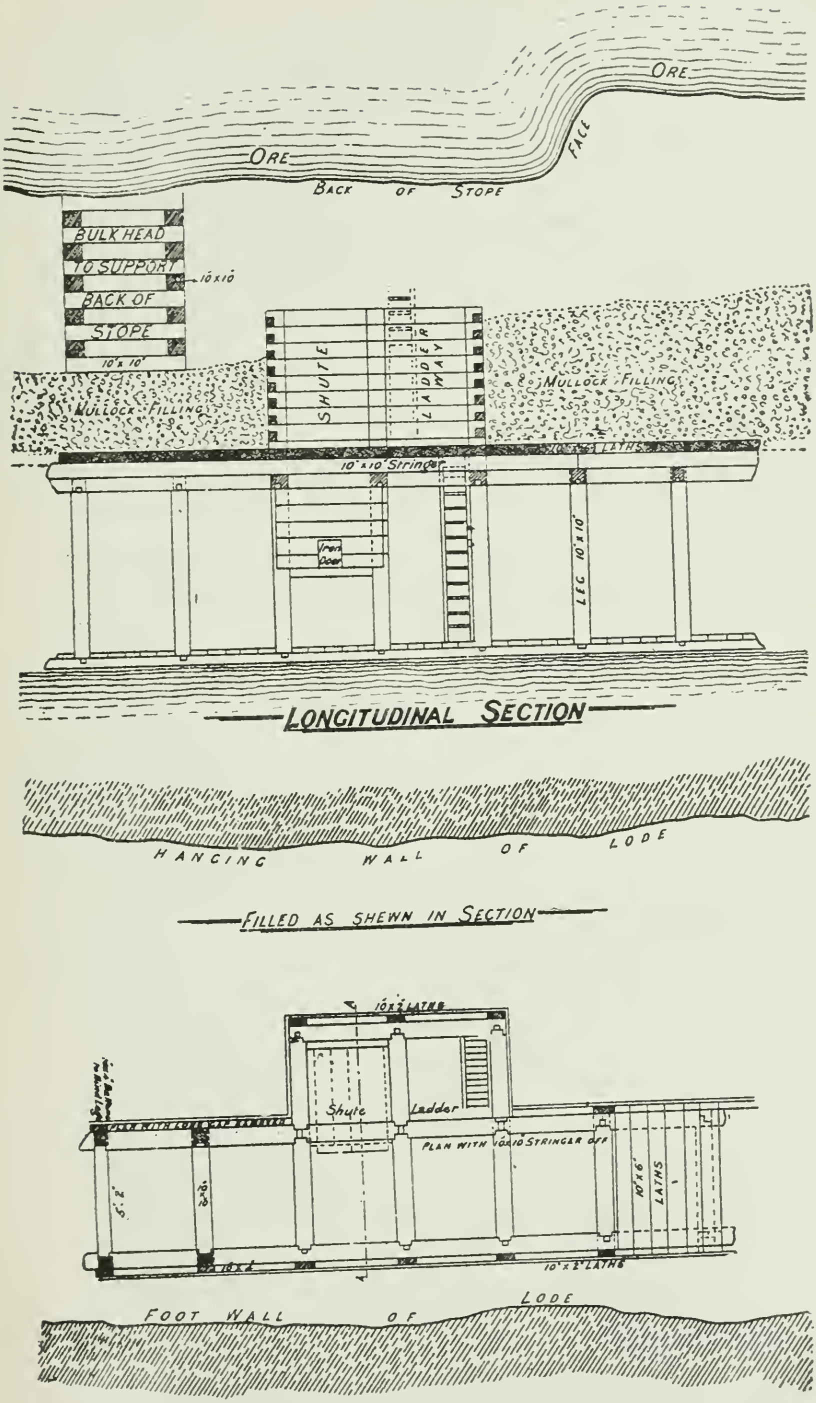
The ore has been practically all removed from the open-cut, and the mullock now obtained by increasing the batter is all sent down below. Previously it was hauled up by means of an inclined tram, the full truck being hauled up as an empty went down the line. Most of the material was raised with a "Flying Fox," a term applied to a carriage which runs on a fixed rope, and has attached to it a skip, which may be raised or lowered as well as hauled along the ropeway. It is generally made use of when ore has to be raised, and then carried laterally. A fixed cable is passed over pillars, or posts, and then securely anchored at both ends, after the manner of the chains, or ropes, of a suspension bridge. In this case, however, the ropeway between the posts is stretched taut, so as to be nearly horizontal. A carriage, called a bicycle, runs on this ropeway. Two grooved pulleys of this run on the rope, and two lower ones are attached to the framework below. A rope is attached to a movable pulley-block, then passes up, and over, one of the fixed lower pulleys on the bicycle, down, and round, the movable pulley, and up over the other lower pulley of the bicycle, thence back to a fixed pulley. On unwinding this, assuming the bicycle is on any part of its track, the skip, which is attached to the movable pulley by a hook, descends; while on hauling this rope, the skip is raised vertically. As soon as it is raised, the same rope draws the bicycle along the fixed cable, and when it is drawn to its discharging place, a self-acting catch steadies it, and the skip is lowered and discharged. It is then hoisted, run back again and lowered, and another skip, which in the meantime has been filled with material, attached in its place. The engine used for hauling works a winch, having a loose drum; it is provided with reversing gear. These appliances are made the most use of at the Port Pirie Smelting Works, and a complete journey, and return, from the H.H. plant, with a full load of two tons, including hooking on, hoisting, hauling to the smelters, lowering, discharging, hoisting, returning, and lowering, and unhooking, only took 1½ minutes.

Underground, the solid sulphide ore is much more compact than was the ore in the upper levels, and great flat roofs are left with little or no support, certainly much less timber is used than one sees almost everywhere else. There are several systems for removing it; the first being called, by Mr. Beaumont, from whom I have borrowed the excellent diagrams accompanying this, the underground open-cut system. Drives are put along the foot and hanging walls; these are connected by crosscuts. Winzes are sunk from upper levels and connected with these drives. From the bottom of the winzes a face is started in the ore. When the ore is removed from around it a double compartment timbered-frame is built up near the initial drive. One compartment serves to send ore down, the other is for men, and provided with a ladder-way. As the ore is removed and sent down the newly-formed pass (the first strip, of course, being broken out and trucked away), mullock is sent down the winze and distributed over the floor. The drive or drives are thus wholly surrounded by mullock, the double compartment frame rising as the ore is stoped, and mullock takes its place. The whole weight of the mullock thus thus comes on the drive timbers, which are made specially strong, especially over-head, to receive it. The ore is then stoped over a wide face, and bulk-head supports, or oregon beams, laid, as shown in the figure, are built up from floor to roof, in places where the rock is deemed loose. These timbers are recovered as the stone is removed. Drills are kept going in the stopes, and huge blocks are broken down: these sometimes amount to many tons weight. They are then drilled by hand, and popped into blocks of a convenient size to handle. At the Proprietary mine they have introduced a



pneumatic hand-drill, or a popper. It is on the same principle as the caulking hammer. The whole machine only weighs 10lb.; it is connected with the air-pressure pipe in the same way as the larger drills, it works with the same clatter as an ordinary drill, and in Broken Hill ores, a man will drill about 18 inches per hour. Such a drill would be eminently adapted for small lodes. I was informed, by Mr. Slee, of the Proprietary mine, that a man can hold it easily, the jar being more apparent than real.

At the Proprietary mine, a modification of the last method des-

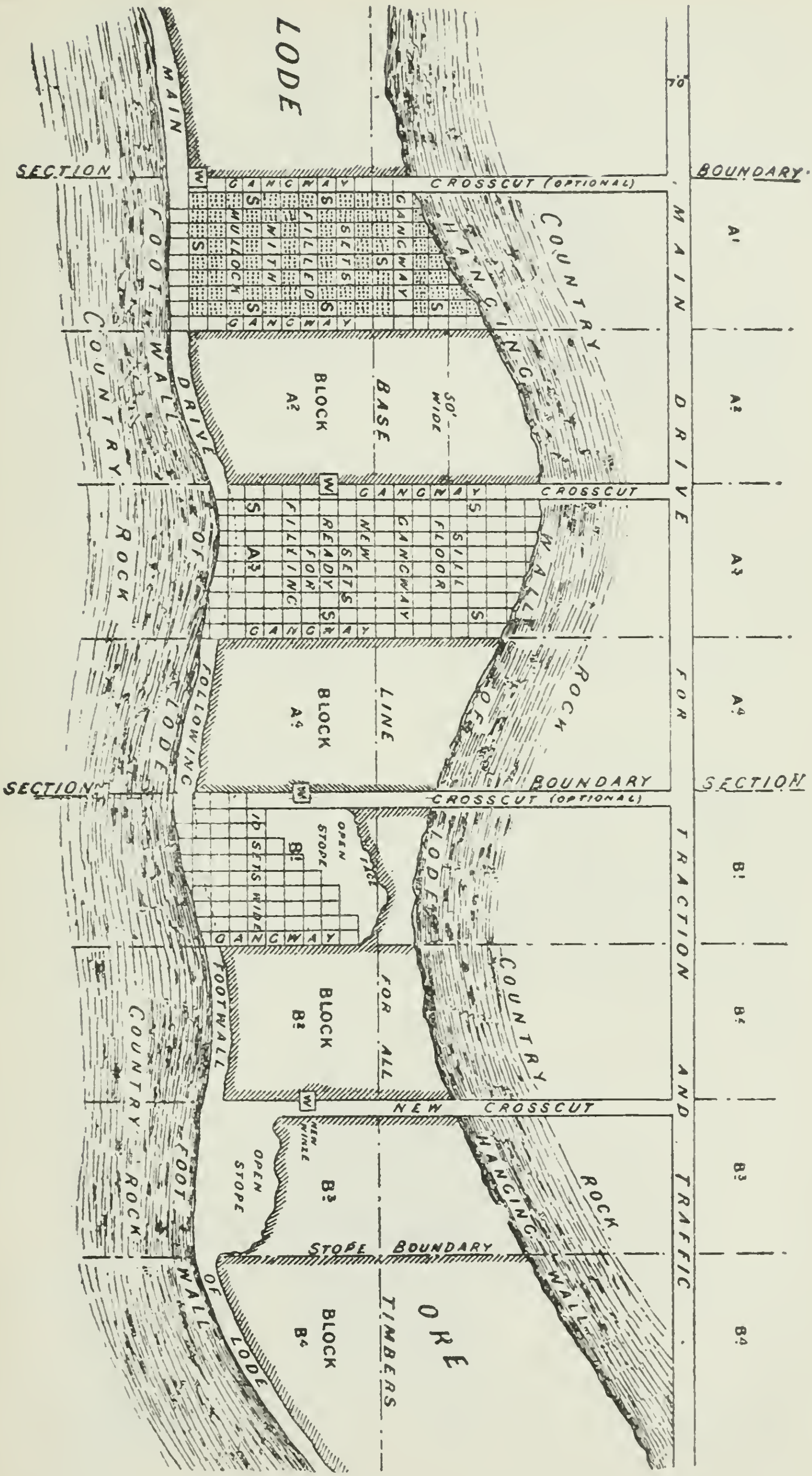


Plan of Sill Floor.

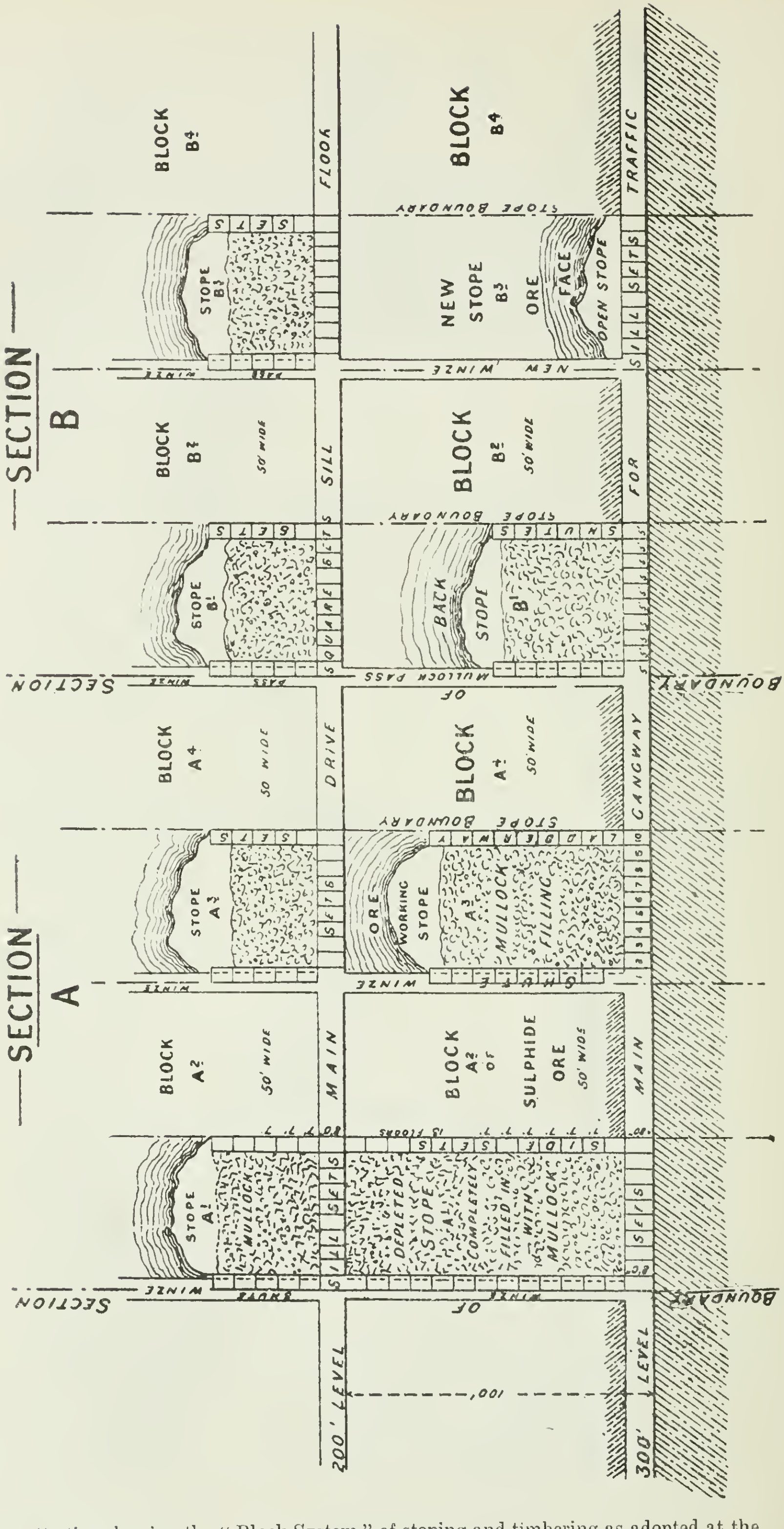
Plan and Section, Showing the "Open Stope" System of Timbering and Filling as Adopted at the B.H. British S.M Company.

cribed, and known as the sloping stope system, is also used. In this the drives are put in as before, the winze is sunk through the ore, and stoping commenced. The ore is removed in such a way that a section is an arch, and then an isosoles triangle, having the winze at its apex, and the sill floor for base. Mullock is sent down the winze as before, and filling proceeds as the ore is removed—the ore itself, however, in falling from the stopes, slides down the inclined wooden surface placed over the mullocked up floor into the shoots. The stope is always worked from the winze downwards, so that very little space need be left between the filling and the roof. The flooring boards can be raised, more mullock sent down, the boards replaced, and stoping again started. The length of the stope will depend on the stability of the ground. The next block is stoped out in a similar manner, but in order to prevent the filling of mullock from running into this a timber partition is built up of vertical pieces, five feet apart, with laths placed horizontally. These are recovered as the work in the second stope proceeds. A combination of this method with square sets is also in use. Some of the square sets may be used for shoots, while the rest are mullocked up. The sets are put in to correspond with the angles of the stope. Still another system has been introduced by Mr. Delprat, the object being, in all cases, to have a minimum of open space left in the mine. In this case a drive is put through the centre of the lode parallel to the walls, then a crosscut is put in from the centre of the block to the walls. This is enlarged to a 7 feet by 7 feet opening, and the ore taken out; another parallel strip is then taken out, and the last one filled with mullock, and so on until the whole bottom strip is taken off. After this an upper strip is taken off in a similar manner, and the process repeated until all the ore between the levels is won. By this system the working gallery is the only open place in the workings, and if any settlement takes place the filling soon takes the pressure up. The large stope at the 650 feet level, which has always been difficult to work, owing to the presence of floors and head, is now being worked on this principle.

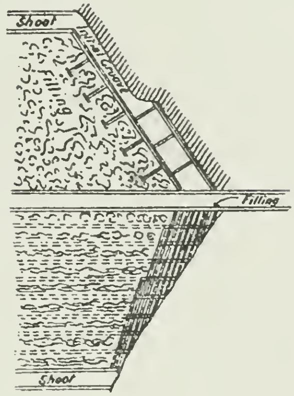
The Central mine possesses immense bodies of ore. It is stated there are at least six million tons down to the 800 feet level, or more than 20 years' work at the present rate of progress, that is, nearly a quarter of a million tons per annum. Since there was no outcrop at the surface of the Central, and there is about 300 feet of barren mullock down to the 300 feet level, the size of the ore bodies on this small lease may be imagined. The lode is from 300 feet to 400 feet in width, and the problem of removing it is a difficult one. Owing to the great amount of overburden theopen-cut method is looked upon as being out of the question. Whether this is so or not, the immediate needs of the company necessitate a more direct method. Mr. C. F. Courtney, the general manager, decided to remove the ore in vertical strips, 50 feet in width, leaving pillars of the same width of solid ore standing. The vacant spaces will be mullocked up as ore is being extracted, and the solid pillars will then be removed. Mr. Hebbard, the mine manager, estimates by this means that the whole of the ore may be safely removed—a result, up to the present, not achieved by any company working large bodies. Mr. Beaumont names this the "block system." The illustrations given make this clear. A main drive for traffic is driven in country on the hanging-wall side of the lode. Crosscuts are put in every 100



Plan showing the "Block System" of stoping and timbering as adopted at the Central Mine.

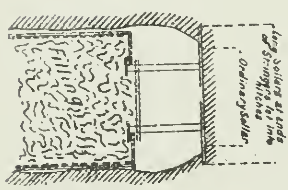


Section showing the "Block System" of stoping and timbering as adopted at the Central mine.

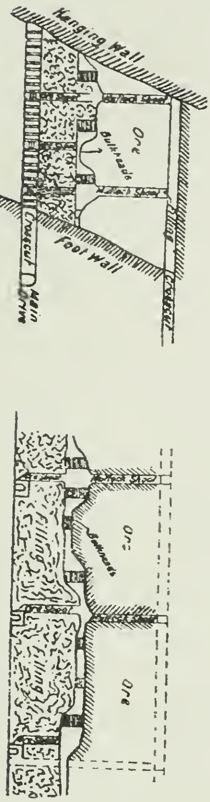


Section of Stope in Course of Work.

Section of stope Completed and Partially Filled.



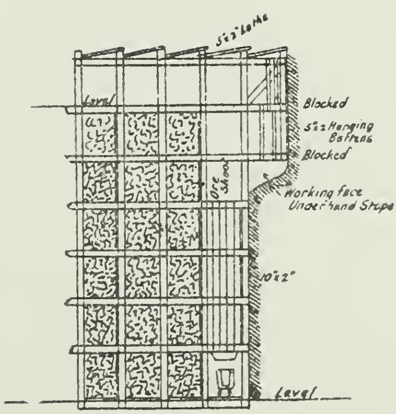
Transverse Section.



Stoping in Horizontal Layers. Adopted in Hard Ground.

Transverse Section.

Longitudinal Section.



Section of Stope on the Square Set System.

Sketch Showing Various Methods of Stoping and Underground Timbering as Adopted by the Broken Hill Proprietary Company.

feet to meet the foot-wall. A main drive is put in along the contour of the foot-wall. The lode is then marked off into 50 feet blocks—each block having 10 sets of 5 feet wide. The whole stope is then out from foot to hanging wall, and the whole space of the first strip removed filled with square sets, leaving a gangway on either side, and where necessary across the block. The other spaces are filled with mullock. A run of square sets is placed on each side as the stope rises, while the central strip is filled with mullock. Laths prevent the mullock running into the open sets at the side which form ladder-ways and gangways. Winzes for sending down mullock are placed at the side of each alternate strip, half in the solid pillar, and half in that removed. This winze will, therefore, serve when the other portion has to be removed. The position of the winzes (W), gangways and stopes may be clearly followed from the illustrations. It will be seen that, while one half of the ore may be easily recovered, the balance will not be won so readily. Much will depend on the shrinkage of the mullock, and on any movement in the walls. It will scarcely be possible to remove the second set of pillars by means of open stopes. The pillars left in the first case support the stopes, but as soon as the first cut is made under the second set of pillars, the whole weight will come on the sets. The strips must, therefore, be taken across the pillar. The lateral adhesion to the mullocked pillars will be small. If it were possible to remove the first strip with a decided upward batter, leaving the solid pillar with a section like a wedge, there would be a greater chance of removing the second pillars by stoping. In such a matter as this nothing should be left to chance. The bad work done on Broken Hill has been mainly due to the policy of mining for the present, leaving the future to look after itself.

The work done by Mr. Samson, at the British mine, deserves more than passing mention. A drive is carried through the lode. This is timbered, and stringers are run longitudinally over the tops of the legs. On these laths are laid transversely; by this means the cap pieces have no weight on them at all, and simply act as spacing timbers. The usual shoots and passes are put up from this drive, and stopes are taken out and filled up with mullock, as in the other cases, bulk-heads being used to support any unsafe looking ground. When the next stope is taken off, these bulk-heads are recovered, to be used over again. By this means the stopes are ultimately filled with mullock, and no timber is used, except that for the necessary shoots and passes. It is found that the drives remain in perfect order after being left for years.

Ore Treatment.

Smelting the oxidised ores constituted the main metallurgical practice at Broken Hill in its boom days. Very few concerned in the management of the mines on the Hill appear to have mapped out any systematic policy either with regard to future mining or treatment. As an instance of gross recklessness, nothing could be more glaring than the position of most of the metallurgical plants; in some cases they were placed right on the lode, while in others plants were put over the position of the lode at lower levels. It was well known that the lode was of immense size, and that its underlay was to the west. The smelters at the Proprietary mine were so placed, while on Block 10 endless trouble has been caused through the settlement of country over the lode interfering with the machinery. The same reckless disregard for the future was shown in smelting operations. "Eat, drink, and be merry, for to-morrow you die" was accepted evidently as the best maxim from day to day until the fatal hour almost came, and would certainly have come had it not been for the few who faced the problem and partly solved it. When on a visit to Broken Hill some years ago, dry ores were treated not only by smelting, but also by amalgamation processes, and also by lixiviation by hyposulphite solutions. The system of amalgamation was crushing in a fifty-head stamp battery, having two pans to each five-head for grinding and amalgamating, followed by a settler for each two amalgamating pans. The whole plant worked continuously, but I should not judge that extractions could have been very good.

The lixiviation plant at the Proprietary mine also dealt with a very large quantity of dry ore, and even now the huge black dump of manganic ironstone remains. The system as introduced by Von Patera depends upon the fact that chloride of silver is soluble in solutions of hyposulphite, or more strictly, thiosulphate of sodium. Unfortunately the bulk of the silver halogen compounds were iodides and bromides, or admixtures of compounds of these with chlorides. It was also found that such compounds did not dissolve as readily as the chlorides. It became necessary to convert these compounds into chlorides by roasting with salt. The cost of treatment was thereby increased from 6s. 1d. per ton to 19s. 8d., but a much better extraction was obtained. I was informed at the time the plant was working, by Mr. G. M. Roberts, that considerable losses took place in roasting during the transition of the iodide to chloride. The amount of salt used varied from 5.5 to 6.6 per cent. of the weight of the ore. The ore was crushed through a slotted screen $\frac{1}{8}$ inch wide and $\frac{1}{2}$ inch long. The dry material was fed automatically into a White-Howell revolving furnace, and discharged at the hot end. The chloridised ore was then transferred into 50 ton vats and washed. The ore carried about 4 per cent. of lead; this was in the form of chloride and sulphate. Part of the chloride dissolved, and this was passed over scrap iron, which precipitated the lead and the silver it contained. The lead precipitate ran about 60 per cent. lead, and carried 500oz.

silver per ton. It was found impracticable to wash the whole of the lead chloride out with cold water, so after two hours' washing the ore was treated with a one per cent. solution of hyposulphite. It was found to be almost impossible at first to obtain good extractions, the filter beds clogged, and there was a limit to the extraction no matter what strength of solutions were used. The cause of the trouble was ascertained, after a great deal of work, to be due to the formation of crystalline hyposulphite of lead. This material coated the silver compounds and prevented further attack, as well as choked up the solution channels. To overcome this, after the ore had been ten hours under treatment with hypo., carbonate of soda was spread over the surface of the ore. The solutions run on dissolved this, and carried it through the ore, where it reacted with the lead compound—

$$PbS_2O_3 + Na_2CO_3 = PbCO_3 + Na_2S_2O_3.$$

In this way sodium thiosulphate was formed and carbonate of lead—the latter as a pulpy product. The reason given for adding the soda at this stage is that it is held that there are a number of molecules of hypo liberated at the very places where their action was before retarded. The solutions containing lead, gold, which becomes soluble owing to the chlorine generated from the salt in roasting, silver and copper were run into a large vat. Lead was precipitated with carbonate of sodium; after the lead had subsided the clear solutions were run into a second vat, where sulphide of sodium was added. This precipitated the gold, silver, copper, and the remaining lead.

The chemistry of the process is briefly as follows:—

$$\text{Roasting—}PbCO_3 + 2NaCl + 2AgI + O_2 = Na_2PbO_3 + 2AgCl + CO_2 + I_2.$$

Other reactions also bring about the liberation of chlorine—

$$\text{Solution—}AgCl + Na_2S_2O_3 = AgNaS_2O_3 + NaCl.$$

$$\text{Precipitation—}2AgNaS_2O_3 + Na_2S = Ag_2S + 2Na_2S_2O_3.$$

While the chemistry of the process appears simple enough, there are many details which require close attention; for instance, if in precipitation more than the requisite quantity of sodium sulphide is added, then instead only of having hyposulphite in the liquid, which of course is used over again, a certain amount of sulphide is present, which would react at once with any chloride in the next vat, and prevent solution. This actually happened during the absence of the superintendent of these works when I was at the Hill some years ago, and much trouble was caused before the solutions were rectified. The carbonate of lead precipitate was fed into the smelters, while the sulphide slimes were roasted and sent to the refinery. The product contained about the following percentages of metals:—Lead, 6.5; copper, 26.0; silver, 22.0; gold 5.5oz. per ton. The dry ore treated ran about 12oz. silver per ton, and the average extraction was about 78 per cent. The amount of chemicals used per ton of ore were—Hypo., 3.8lb.; caustic soda, 2.25lb.; sulphur, 1.5lb; soda carbonate, 21lb.

Many smelters were at work on the Hill when the oxidised ores were being treated; ironstone was abundant, limestone was first obtained from the secondary deposits around the flanks of the neighboring hills; subsequently a line was run to Tarrawingee, where a metamorphic limestone was quarried; coke was obtained from England. When the sulphide problem had to be tackled, the values would not admit of the extravagant treatment of the early days, so that the smelters were all removed to near the coast. There would have been but little sense in running fuel and fluxes for miles to the smelters and carrying away the bullion, whose weight averaged 60 per cent. of that of the concentrates treated. As previously stated, the sulphide ore consists of an even admixture of crystallised galena and platy zinc blende, with a gangue of quartz, sometimes crystallised, sometimes opaline, garnets and rhodonite. Where the ore occurs in large bodies, it has a compact granular structure. Its sub-division in crushing appliances is thus rendered comparatively easy, also admitting of a fairly perfect separation of the various mineral ingredients.

An analysis of a sample from Block 10 is, according to Mr. J. C. H. Mingaye, as follows:—

Moisture...	2.065
Iron...	5.675
Lead ...	18.755
Copper ...	.244
Arsenic ...	.057
Antimony...	Trace
Cadmium ...	Strong Trace
Bismuth ...	Nil
Sulphur ...	20.426
Alumina ...	2.161
Lime ...	Nil
Magnesia...	2.339
Carbon dioxide...	.350
Gangue...	18.500
Silver...	30oz. 4dwt. 8gr. per ton
Gold ...	3dwt. 6gr. per ton.

It was currently stated that it would not be possible to treat the sulphide ores by concentration; it was held that a large proportion of the zinc-lead sulphides were not admixtures, but isomorphous sulphides, and that no process of mechanical separation could effect a separation of this so-called compound. It is well known now that such is not the case, and as a matter of fact many of the grains of blende and galena are coarse, certainly up to one-eighth of an inch in diameter. Smelting with high proportions of zinc was not looked upon as possible. Amongst the foremost to dispel these illusions were Captain John Warren, of Block 10 mine, and Mr. J. S. Greenway, metallurgist to the Junction, and subsequently to other mines. Concentrating mills were erected to deal with the sulphide ores from one end of the field to the other. The upper sulphides were partly oxidised, friable in character, and as a rule richer in lead and silver, and poorer in zinc, than the compact ones down below. While these were being treated Broken Hill had a big revival, and in a single year turned out minerals or products sold for more than the gold raised in any State of Australia for the same year. When these sulphides were exhausted, and the mines produced lower grade material, the fall in the metal market was sufficient to practically shut down most of the mines. Others struggled on and made a small profit by dealing with immense quantities, but it must now be seen that ordinary concentration alone must only form one part of even fairly successful treatment. With regard to smelting, any appreciable quantity of zinc was looked upon as fatal to economical work, and slags up to 20 per cent. was looked upon as impossible. Mr. Greenway was able to show that he could produce these and smelt them profitably. The concentrates actually treated contained about 40 per cent. of lead, and 20 per cent. of zinc. This was roasted at a low temperature, no effort being made to expel the whole of the sulphur, nor to sinter the product. The sulphur remaining after roasting amounted to about 8 per cent. Curiously enough, the sulphur was eliminated in smelting. The slags produced were highly basic, running as a rule as follows:—

Silica (SiO_2)	24.8
Ferrous oxide (FeO)	28.7
Zinc oxide (Zn O)	15.9
Manganese oxide (MnO)	4.4
Lime (CaO)	14.1
Alumina (Al_2O	3.0
Lead oxide (PbO)	3.3
Undetermined	5.8
	100.0

It is probable, in view of the discoveries leading to the use of the Huntington-Heberlein process, that some of the reactions of the H.H. converters went on in this, but that the fuel also brought about the reduction of the lead.

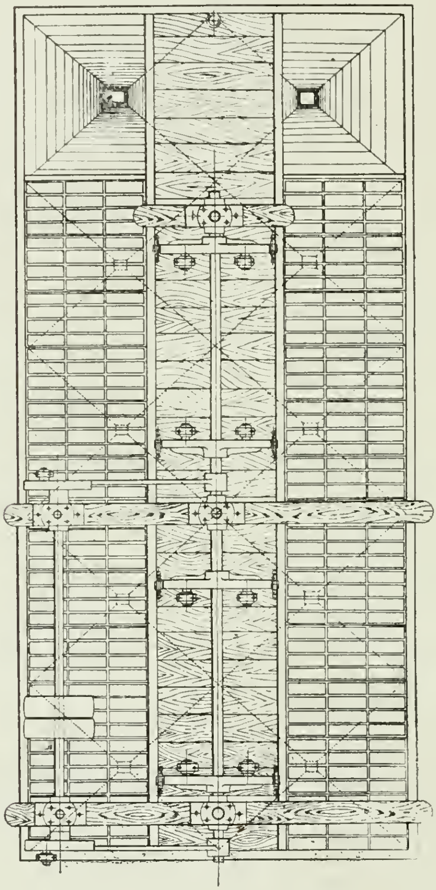
While both concentration and smelting methods then adopted served to prolong the lives of some of the mines, yet both in themselves were wrong, and the further they were persisted in the worse has become the position of the mines.

Dr. Schnabel, in his elaborate report, recommended nothing new, but the essence of his report was that the zinc product was to be separated from the lead. The method recommended by him for doing this, namely, the roasting of the sulphide ore and extraction of the zinc as sulphate, was impracticable. The companies thus were no better off. Concentration methods give frightfully wasteful results. The smelting with high zinc contents make the work difficult, and destroys for ever the chance of obtaining a valuable metal—zinc. It may be said that apparently the whole of the chemical possibilities for the separation of these metals had been exhausted; some had been commercial failures; others, such as the Ashcroft process, were almost successful, and still others, now before the public, will probably never be tried at all. The methods which have been undoubtedly successful in dealing with the ore are the magnetic processes, and that depending on the action of slightly acidulated solutions, now commonly known as the Potter process, and salt-cake process respectively.

Concentration at Broken Hill Proprietary Mine.

The work as carried out on the large mine is typical of that done on the field. Variations may be found in other mills, but the general practice is the same. A somewhat detailed description of this plant will stand for most of the mines.

The ore, which is broken below by contract, is selected as far as possible, worthless gangue going for filling. It is further inspected at the brace for mullock, any valueless material being sent below. Trucks are weighed periodically. Each has a capacity of about a ton. The ore is raised and run along an elevated tramway into a tumbler. This consists of a framework for holding each truck; at each end of the frame, and at right-angles to the length of the truck are circular bands of steel resting on friction rollers. By seizing the frame on one side the tumbler containing the truck is inverted, and the contents fall on to a grizzly. The empty truck is pushed out and another takes its place. This device, which is commonly used on the field, allows of a truck being made without opening doors, or provisions for kick-ups. The grizzlys are made of bars about 12 feet long, ½ inch wide above and ¼ inch below, spaced about an inch apart by means of circular washers. The angle of inclination is about 50 degrees, and the total width about 6 feet. The fines pass into a bin below the grizzly, the coarse passes on to four stone-breakers of the Gates type, No. 4. These break the material down to 1½ inch gauge. The product from the breakers goes into the same bin as the fines from the grizzly. The crushed material is sub-divided between the two large concentrating plants: the former, which has a mill of the gradual reduction type, does better work than that in which the ore is returned to the same rolls until fine enough for treatment. It will therefore only be necessary to describe the former. The amount which goes to each plant is about equal or about 1000 tons per day, or in all, from 12,000 to 13,000 tons per week. From the bin the material is trucked and passes on to a hydraulic lift; it is then re-elevated on to the original level and tipped into a hopper which supplies the rolls. The ore is evenly distributed from this hopper to rolls Nos. 1, 2 and 3 by means of automatic feeders. These consist of a roller feeder at the lower end of the hopper, the roller being about 10 inches in diameter, with a 15-inch face, and rotating about one revolution in 1½ minutes. This gives a very even feed. By raising or lowering a slide over the roller a greater or lesser amount may be fed in. At this point a jet of water strikes the ore, and as wetted it is fed into the rolls. The rolls have a diameter of 2 feet 6 inches, with a 15-inch face, and are set ⅜ inch apart, and run at the rate of 37 revolutions per minute. The product from these rolls passes into corresponding trommels, which are conical in shape, resting on horizontal axes. They are made of sheet iron, with holes of 3-32nd of an inch. All material of this size and finer are led away for classification and concentration. The coarse material from the trommels 1, 2 and 3, falls on to a belt conveyor, which carries it to the boot of a bucket belt elevator. It is raised by this to the top level again, and falls over a perforated screen into a hopper below. From this hopper it is delivered by a roller feeder into No. 4 roll, which has the same dimensions as the others, but runs at the rate of 45 revolutions per minute, with rolls 5-16th inch apart. Since No. 4 rolls have all the coarse products from Nos. 1, 2, and 3, the perforated launder



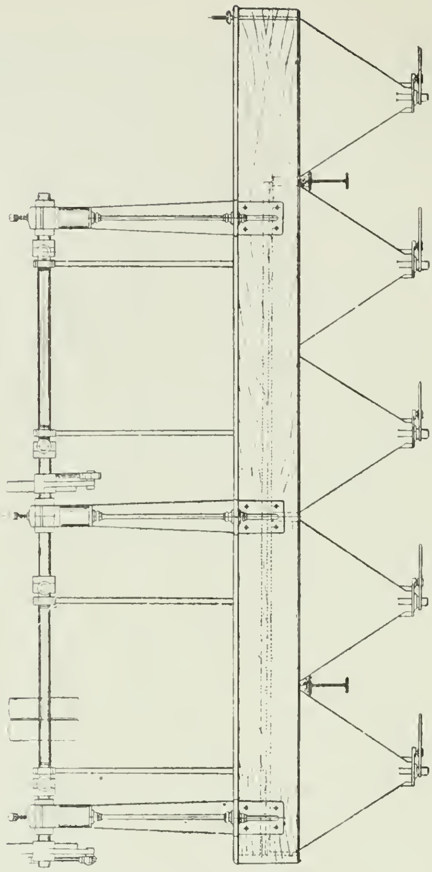
May Patent Compound Jig.—Plan.

on the upper floor removes some of the finer material, which passes to a hopper supplying Nos. 5, 6, and 7 rolls.

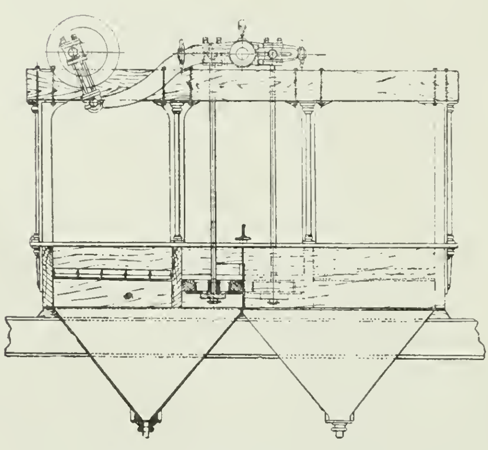
The product from No. 4 rolls passes into a trommel. The finer material passes to the classifiers and concentrators as before, the coarse being re-elevated to the upper floor. From thence it passes into a hopper, which supplies Nos. 5, 6 and 7 rolls. These are of the same type as before, but run 80 revolutions per minute, and are set 1-16th inch apart. The products from these pass into corresponding trommels, the fine material going to classifiers and concentrators, the coarse material from 5 and 6 are returned and sent through the same rolls, the coarse from No. 7 trommel goes to a hopper, whence it passes into Nos. 8 and 9 rolls, running 80 revolutions per minute, and set in contact also to a No. 5 ball mill running 25 revolutions, and crushing to 3-32nd inch. The products from 8 and 9 rolls pass through corresponding trommels, the coarse from these being re-elevated and returned until all passes through. So far it will be seen that this system is one of gradual reduction of the material, so that the largest grain does not exceed 3-32nd of an inch in diameter.

Corresponding to the rolls are the coarse jigs. There are nine of these to take the product from the rolls. At the head of each is a hydraulic classifier, which washes all the fine sand and slime out, all of which are further classified in spitzkasten; the coarse material passes into a double May jig having 5 compartments on each side; the products from the first and second hutches are concentrates, which go to the smelters, those from the third and fourth hutches of the jigs are reground and rejigged in fine jigs, while the product from the fifth compartment goes to the dump. The hydraulic classifier at the head of the jigs is a small conical box 2 feet diameter and 2 feet 6 inches deep. A jet of water under pressure washes the slimes into an overflow launder, while the coarse sands pass through into the jigs. The May jig is widely adopted by the plants on this field, and has proved very successful. Each double jig, as before stated, has 10 compartments, five on each side, the fifth in each case being for tailings. Each hutch is 3 feet 6 inches by 2 feet 6 inches, and has a woven wire bottom 6 inches below the water level, having 6 meshes to the inch, stiffened by cross-bars below and grids above.

Each compartment is a pointed, or inverted pyramidal box, the base of which is 3 feet 6 inches along the jig and 4 feet across it. The angle of slope of the sides is 50deg. from the horizontal across the box and 57deg. along it. The sides of the jig are extended vertically for about 18 inches above the tops of the pointed boxes. A longitudinal partition divides the five boxes on one side from those on the other up to this level. Longitudinal partitions placed on either side of this central line, and at a distance of about 15 inches from it extend to the same depth as the sides of the jig. These serve as the plunger compartments. Each plunger extends the whole length of the box, or 3 feet 6 inches, and is 14 inches wide. The plungers have clack openings 2 feet 6 inches long by 6 inches wide. This is to give a quick upward motion to the material on the screen, but to allow of a slower return, the heavy minerals finding their way through the sieve bed into the compartment below, the lighter materials passing on to the next screen. The clack consists of a board, loosely bolted to the bottom of the plunger, the clearance being about $\frac{5}{8}$ inch. The plungers of the coarse jigs are run at the rate of 165 revolutions per minute, with a $\frac{3}{4}$ inch maximum throw. At the Proprietary mine the size of the material dealt with is 3-32nd inch. The throw given to the plunger is accomplished by means of the eccentric motion of the driving shaft being transmitted through a lever having as a fulcrum a longitudinal axis. Each lever drives a plunger on either side of the central axis; the throw of the plunger may be increased or diminished by moving the head of the plunger rod



May Jig.—Side Elevation.



May Jig —End Sectional View.

further away from or nearer to the axis. Each jig requires about 2 horse-power, and will treat as much as from 6 to 7 tons per hour.

The fine jigs are of the same type as the coarse, but are driven at a higher rate of speed, and deal with material crushed down to 1-32nd inch.

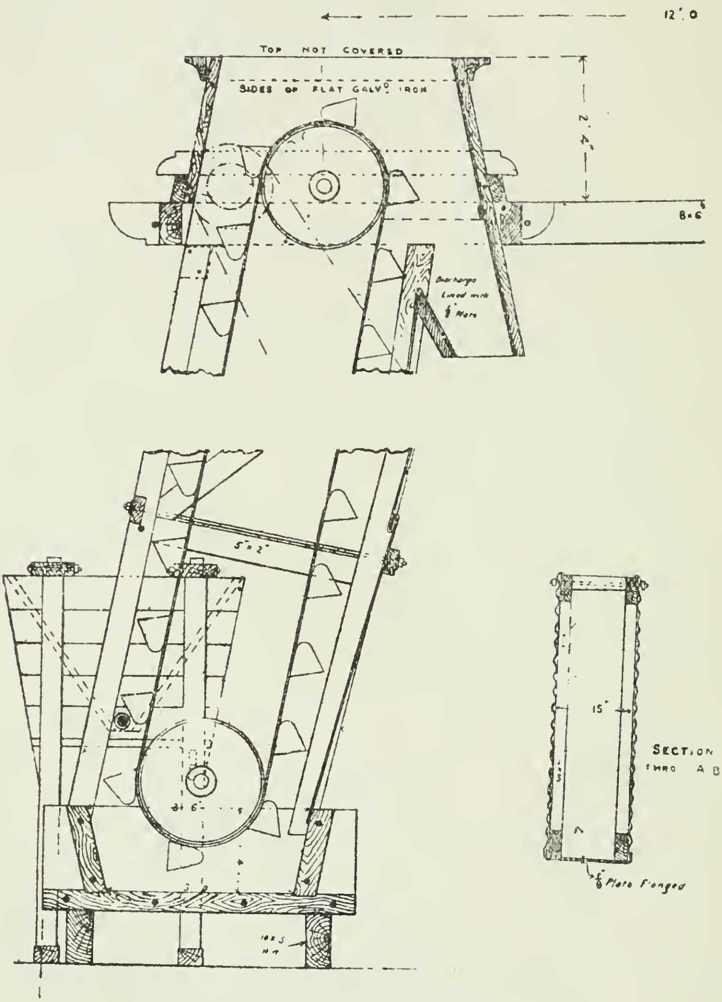
The products from the third and fourth compartments of all the jigs go to 5 elevators, thence to 19 Heberle grinders; the products then pass to 5 hydraulic classifiers, the overflow going to spitzkasten, the coarse material on to 5 May jigs. The product from the first two hutches of the jigs are concentrates, the last tails, while the middles pass to 3 elevators having spitzkasten at the boots of each, the product from these flowing away for future treatment, the coarse middle product being elevated and sent to 3 No. 4 ball mills, having slot screens 1-32nd inch wide. The products go to 5 elevators, and are returned to the same jigs.

The Heberle grinders are mainly used for crushing jig middlings. The machine consists of two vertical revolving discs placed slightly eccentric to each other. The sand is ground between the vertical faces. In order to feed the machine one axis is made hollow, and the ore is fed through this, and escapes between the discs. These are pressed up towards each other so that a product of almost any degree of fineness may escape at the periphery. The object of these machines is to crack up and detach mixed products such as galena and gangue without unnecessary pulverisation. It seems doubtful whether this is the most economical means of doing the work.

The elevators, which are widely used, are of the belt and bucket type. The belt has buckets attached to it about every 15 inches, the buckets being so arranged that their surface is horizontal when the belt is inclined, the angle of inclination of the latter being about 80 degrees. Flanged drums about 2 feet in diameter are placed at the top and bottom for the belt to travel over, the top drum being driven so that the belt travels about 250 feet per minute. Near the boot of each box, from which the buckets pick up their material, is a separating box arranged so that only sand or thick pulp passes through for the buckets to elevate; the water separated at the top of the box containing slimes is pumped to other settling boxes. The elevator is enclosed in a framework covered with galvanised iron; thus leakage or splash is avoided.

In the first stage of treatment the products from the rolls were treated and passed through the coarse jigs; in the second stage the products from the third and fourth hutches from these jigs were recrushed and sent through the fine jigs. The next stage of treatment consists in treating the sand and slimes from all the jigs, classifiers, and spitzkasten. This material is led by launders into spitzkasten classifiers. These are made in one box, widening and deepening at the overflow ends. The bottom is sub-divided into five pyramidal compartments, each having a spigot. Any coarse material from the first compartment goes back to the fine jigs; the intermediate sized material goes to seven Wilfleys, and the fine sand to 28 Luhrigs. The heads from the Wilfleys and Luhrigs go to the smelters, the tails to waste, and the middles to a classifier on the lowest floor. The material from the first spigot goes to four Wilfleys, and the fines to eight Luhrig vanners. At this stage only heads and tails are made, the middlings being returned.

The overflow from the classifiers containing the true slimes passes into a large spitzkasten for further settling slimes, the overflow water from this serving for the whole of the Luhrigs and the Wilfleys; any overflow from this passes into a tank, from which

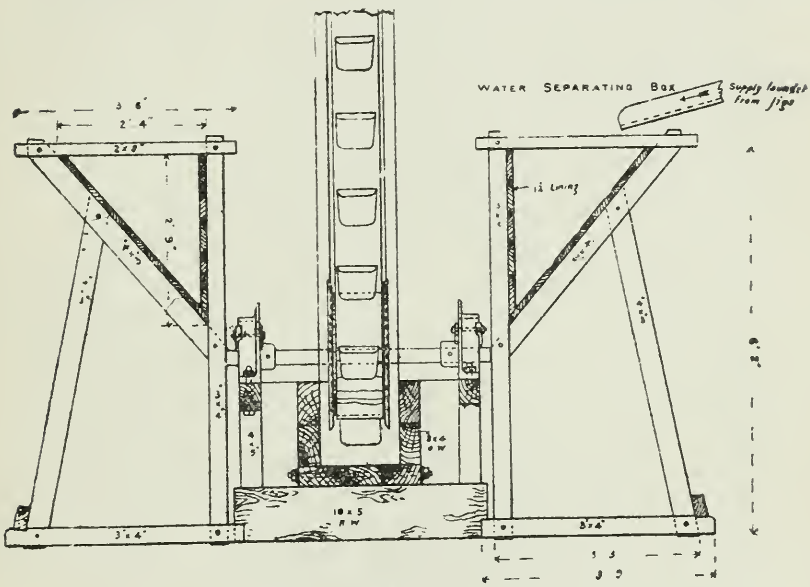


Arrangement of Elevators.—Side Elevation.

clear water may be drawn for supplying wash water for vanners. The slimes are run out through spigots into settling tanks, from which they are trucked, and emptied along the railway line. Here the slime is subdivided into sods with spades, allowed to dry, and the sun dried bricks are trucked down to the sintering works.

It may be said that true slime treatment is not attempted at these works; the products treated on the Wilfleys and Luhrigs are only coarse or fine sands; the dividing line between heads and tails, or rather heads and middles, is not easy to determine, and the final product in all cases contains a considerable amount of lead. The amount of this metal, as well as silver and zinc, will be dealt with in a future article. The concentrates are got rid of in all cases by being run down launders into large bins, whence they may be at once run into trucks. The tails or middlings are got rid of by being dumped direct into trucks, and are carried to the tailings heaps.

The capacity of the various machines may be put down as— Each May jig is capable of dealing with 1000 tons per week, each Wilfley 16 tons per 24 hours and up to one ton per hour, each Luhrig about 6 tons per 24 hours. The Luhrigs are fitted with Balata belting, which has been found to be much better than that



Arrangement of Elevators.—Front Elevation.

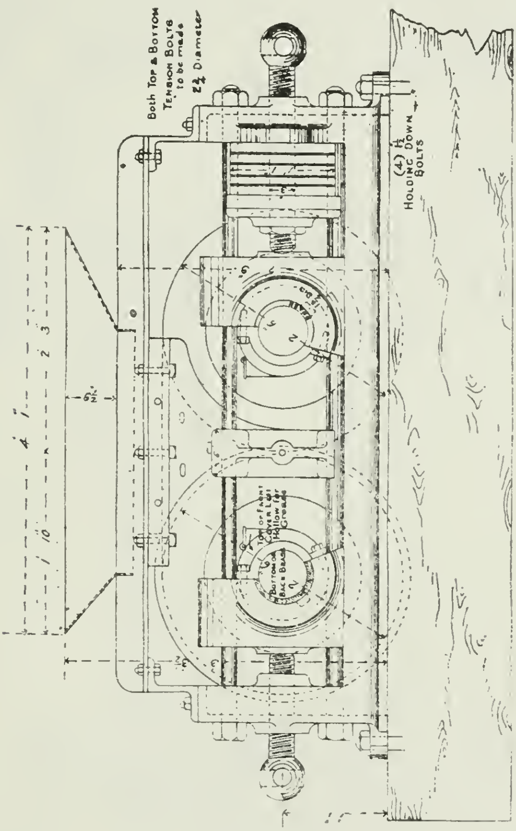
previously in use. It is interesting also to note that these and the Wilfleys are run at a lower rate of speed than is usual in other places—the Wilfley at 216, and the Luhrig at 150. Mr. Hilden, the superintendent of the mill, informed me that by running at higher rates of speed the return stroke does not synchronise with that of the spring, consequently there is a jar, which impedes concentration, and leads to excessive wear and tear.

The products from the mill have the following values:—

From the first compartment of the coarse jigs ...	Lead	70 per cent.
,, ,, Second ,, ,, ,, ,, ...	,,	56 ,, ,,
,, ,, Fine jigs	,,	40 ,, ,,
,, ,, Wilfleys and Luhrigs	,,	50 ,, ,,
Average assay of all products	,,	55 ,, ,,

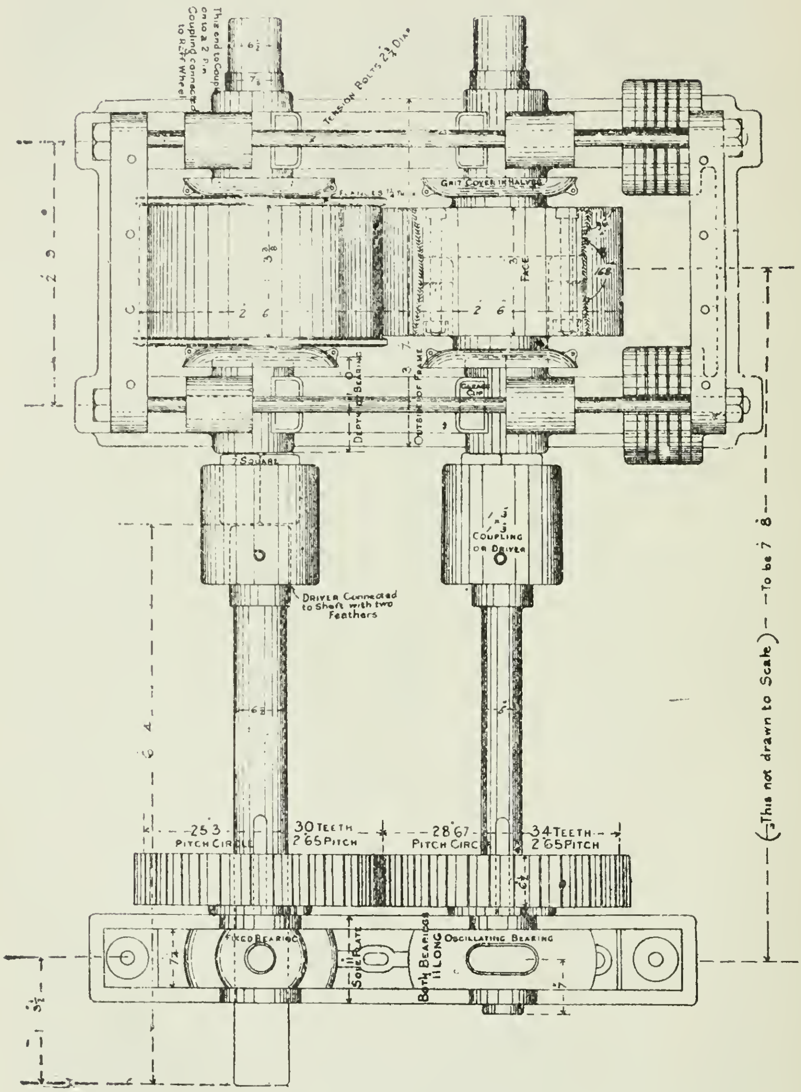
Block 10 Company.

Block 10 was one of the few mines which never went in for elaborate smelting plants. The rich ores in the upper levels were sold, and when the sulphide problem appeared to be almost insoluble Captain John Warren perfected his ore dressing plant and paid dividends when other mines shut down their smelters and even ceased producing. Unfortunately the remarks made about placing machinery on the line of lode apply in this case. The concentration plant now running, according to Mr. Stanley Low, the general manager, is causing endless trouble through bearings getting out of line, due to the surface creeps. A new plant, capable of treating 3600 tons per week, has been erected at a cost of over £50,000, which will do but little better work than the one now being relegated to the scrap heap. A description of the old mill would be superfluous. The new mill is admirably placed on a rocky knoll about a quarter of a mile from the mine. Material will be transported to this by means of an aerial ropeway similar to that used on the Lyell field. The ore as raised from the mine will be dumped into a bin, and, after passing over a grizzly, will be sub-divided, the screenings going to a bin below and the coarse material going through two gyratory rock breakers supplied by the Austin Manufacturing Company of Chicago. These are placed near the base of the poppet head on an independent foundation. A great deal of experience has been gained at Broken Hill concerning these machines. When the jaw breakers were commonly used, some very hard material would lead to their bursting so often that these breakages were taken as a matter of course. When they did break they were useless. To minimise this evil an excellent plan is adopted at the Central mine; the body of the breaker is made in two parts and held together by an iron or steel band or strap. This is of such size that all ordinary rock may be crushed; but if a hammer-head or piece of unbreakable metal gets in the strap will burst. The broken ore is transferred into bins placed at the head of the new plant. The mill is divided into sections, so that should any part break down the other machinery can run on uninterruptedly. There are four automatic feed rolls, each 10 inches in diameter, with an 18-inch face running at the rate of 4 revolutions per minute. Each one delivers the material into a conical trommel 5 feet 9 inches long, 2 feet 3 inches at the smaller, and 4 feet 6 inches diameter at the larger end. This runs at the rate of 12 revolutions per minute. The trommel is covered with 14 gauge iron, punched with round holes $\frac{1}{8}$ inch in diameter. The coarse material passes along the trommel to the rolls, the fines passing to classifiers, thence to the jigs. There are 4 feed rolls of the Cornish type. These are used on many of the mines at the hill. The main difference from other rolls is that one roll is provided with flanges, the other fits into these, thus preventing any lateral escape of material. Each roll has a diameter of 2 feet 6 inches, with a 15-inch face on the plain, and a $15\frac{3}{8}$-inch on the flanged, the width of each flange being $1\frac{1}{4}$ inch. On another mine I saw the rolls 2 feet 6 inches diameter for the flanged, and 2 feet for the plain. Each roll is made in several pieces. The shell or tyre of the plain roll is cylindrical externally, but tapers from the centre towards the outside, the angle of taper from the cylindrical form being about 8deg. The core pieces are of different construction from the usual type of Krom rolls for one core piece is 1 foot 9 inches in diameter at the axis, to which it is attached, and is 1 foot $2\frac{1}{8}$ inches in diameter over more than half its total length, and 20 inches at its widest end. This then tapers at the same rate as the interior of the



Cornish Rolls Elevation.

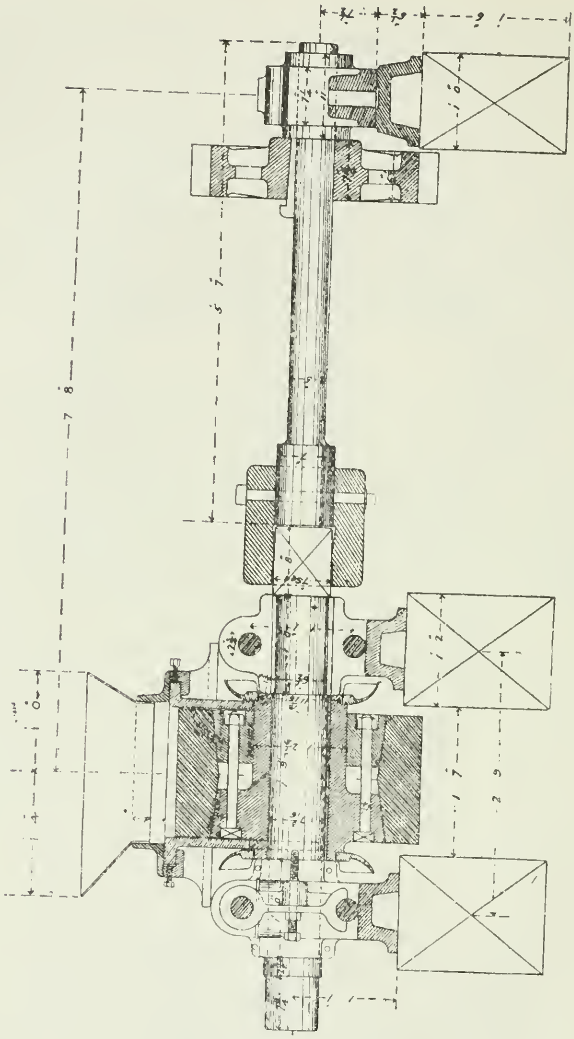
shell for six inches along it. The other core piece is in three pieces, each one-third of a disc 6 inches wide, with a taper on the outside to fit the shell, and having a cylindrical bored surface inside of the same diameter as the smaller turned outside portion of the first core. The inner core is first placed on the shaft, which is 7 3-16 inches diameter. The steel shell is placed loosely over this, and the sectional core pieces put in and brought up by means of two bolts in each piece. At another place I noticed the shells keyed on to the cores with wedges of solid jarrah timber. On this mill the rolls are driven by gear wheels at the rate of 15 revo-



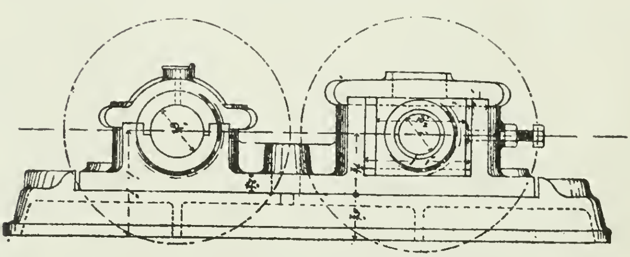
Cornish Rolls Plan.

lutions per minute. After passing through these rolls the crushed material from each pair of rolls enters two cylindrical trommels set at a slope of 1 inch to 1 foot. Each trommel is 6 feet in length, and 22 inches diameter, and revolves 20 times per minute. The screens on the trommel are punched with 2½ m.m. holes. The fines passing through unite with those from the first conical trommel and flow together to the classifier at the head of the jig. The coarse material is returned with a raff wheel to the same set of rolls. This style of treatment is adopted at several places. The raff wheel is a large wheel about 14 feet in diameter, and one foot wide. On the inner circumference of this are a number of compartments made by flanging the inner rim on both sides for about a foot and putting divisions across, these last being so arranged that they will lift the stuff up near the top and drop it into a hopper above the rolls.

It is somewhat remarkable that this system is persevered in with such a costly plant. The returns from the trommels will include some of the hardest material; this will be in size much smaller than the ore being fed in, so that it is not possible to do the best work by this system of total reduction in one machine. The gradual reduction process, as used in the new mill at the Proprietary mine, is a great improvement on this irrational method of ore reduction. The whole of the material passing through the rolls ultimately goes to the classifiers, which are small conical boxes having an upward jet; the fines overflow and are carried down a launder to a slime separator. The coarse passes into one of May's jigs of the same type as that described in a previous article. The products from the first and second hutch are marketable, from the fifth are tailings, while those from the third and fourth will be retreated. The plungers of coarse jigs here are arranged to run 180 strokes per minute. The products from the third and fourth hutches are run from launders into the boot of a bucket and belt elevator, which travels at the rate of 250 feet per minute. The material raised is discharged into a No. 4 ball mill, running at 50 revolutions per minute, arranged for wet crushing. This material is crushed down to 25 mesh. The product from the mill passes to a slime separator, thence to the fine jigs. These are of the same type as the coarse, only they are run at 200 revolutions per minute. Concentrates are obtained from hutches 1 and 2, tailings containing blende go from No. 5, while the products from 3 and 3 are retreated by sending them back to the ball mill, and again dealing with them in the same jig. The fine material from all slime separators or spitzlutten are run into spitzkasten; the coarser material is run on to Wilfley or Krupp tables, having percussive strokes of from 210 to 220 per minute. A series of slime thickening boxes runs right across the whole building. In addition to these, there are a number of four-compartment spitzkasten, having a width at the delivery end of 2 feet 6 inches, and at the discharge end of 6 feet 9 inches, and a length of 15 feet 6 inches on top and 10 feet at the bottom, the first box being 5 feet, the second 3 feet 6 inches, the third and fourth about 3 feet. From each one of these compartments the thickened plup is led on to a separate Luhrig or Warren vanner. The Warren vanner possesses the good points of a Luhrig and Triumph vanner. It is like the Luhrig because the slope of the belt is lateral, and

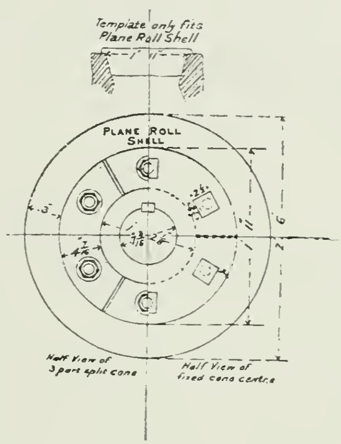


Cornish Rolls Section.



Cornish Rolls.

the travel is from the head to the tail of the machine. It is like the Triumph in that the end shake is given by an eccentric motion and beautifully balanced by discs on the shaft. The travel of the belt is also capable of being regulated by means of a lateral friction pulley after the manner of a Triumph. The feed is delivered near the head of the machine; a water trough delivers clean water along the length of the higher side of the vanner. The clear water running down the belt sweeps the light material off into the tailings compartment, the coarser and heavier material is carried along further as the belt moves towards the tail of the machine; but the percussive shake agitates the coarse material and allows the fine heavy material to get down to the surface of the belt, where each particle clings. The coarser material is washed off at the middle of the table, the fine rich concentrates travel down



further, and are swept off by a jet of water into a compartment for the heads. At the present mill the system is to treat the material which first goes on to the vanners as heads, middles and tails; the heads are saved, the tails go to waste, and the middles are sent on to another set of vanners, where heads and tails alone are made. All tails are trucked away to the dump; the concentrates are sent to bins, and discharged wet into trucks. I learnt that the jigs, of which there are four in the present plant, each gave $6\frac{1}{4}$ tons per shift of concentrates, or 25 tons in all. The Warren vanners, of which there were 18, gave 16 tons, while the retreatment vanners, of which there were 12, gave an additional $4\frac{1}{2}$ tons. The tailings going to waste are said to average from 6 to 16 per cent. lead. The average grade of ore treated at the mine for the last six months, according to the last report, was 13.8oz. silver, 15.2 per cent. lead, and 17.4 per cent. zinc. The recoveries were, on nearly 70,000 tons, 65 per cent. lead, 36.83 silver, and no zinc. In fact the zinc was recovered with lead concentrates, where it is not only waste, but actually detrimental to smelting. Mr. V. F. Stanley Low thus classifies the work:—

	Tons	Silver. Oz.	Assay Lead. %	Zinc. %
Concentrates	10,930	32.5	63.4	8.6
Vanner middlings...	850	12.2	8.5	23.8
Slimes	4,198	18.1	16.3	19.7
Jig Tailings	12,567	8.3	4.7	18.8
Jig Middlings...	33,583	10.0	7.1	21.4
Vanner and overflow tailings ...	5,881	9.7	6.4	19.6
Vanner extension tailings	1,663	10.0	6.6	19.7

The milling costs are given as 4s. 9.48d. per ton, and ore raising as 11s. 3.21d. per ton.

Mr. Stanley Low, the general manager of the mines, states that he expects a plant practically identical with the new plant at Block 10 to give 20 per cent. concentrates per ton of crude ore, 20 per cent. jig tailings, 7 per cent. slimes, 7 per cent. vanner tailings, and 45 per cent. zinc middlings. Unless the values of the products are also given the statement has little meaning. He also estimates the horse-power of such a mill as that described, laid out in four independent sections, as follows:—

		Total H.P.
2 Gates breakers	40 H.P. each	80
4 Feed rolls		
4 Cornish rolls		
4 Trommels...	30 H.P.	120
8 Parallel trommels		
4 Coarse jigs	2 H.P. each	8
4 Fine jigs	1½ H.P. ,,	6
4 Wilfley tables	¾ H.P. ,,	3
4 Ball mills	10 H.P. ,,	40
2 Krupp tables and		15
24 Warren vanners		
4 Elevators	5 H.P. each	20
5 Slime pumps	2 H.P. ,,	10
1 Circulating water pump		25
1 ,, ,, ,, ...		10
1 ,, ,, ,, ...		10
		338

The whole of the machinery will be driven by motors, each section of the plant having its own group of motors. In this way the network of shafting and belting will be avoided.

Concentration by Gravity Processes.

Ordinary concentration is a difficult operation at Broken Hill, the minerals occurring in the lode being galena—sp. gr., 7.5; sphalerite, or zinc blende, sp. gr., 4; rhodonite, sp. gr., 3.63; garnet (almandite), sp. gr., 4; quartz, sp. gr., 2.65. Assuming all the minerals were in separate grains of appreciable size, and all grains were of the same diameters, then the first, or galena, and the last, quartz, could be wholly separated by efficient concentrators. It would not be possible to separate the blende from the garnet, or even perfectly from the rhodonite. The problem gains in complexity when the intimate mixture of the minerals is considered; it is not possible to wholly separate the admixed minerals by crushing down to 3-32 of an inch, although a large proportion of the galena is separated in comparatively coarse grains, and recovered in the first hutch of the jigs in an almost pure state. The rest of the material is either crushed much more finely or is still a coarse admixed product, while with every crushing operation some of the material is reduced to an impalpable powder, which remains suspended for a long time in water.

The hardness, brittleness, and cleavages of the minerals are all of importance. Galena is the softest, H. 2.5, and is easily pulverised by attrition, or broken into minute grains owing to its perfect cubical cleavage. Blende has a hardness of 3.5 to 4, and is also cleavable, although hardly so easily triturated as galena. Rhodonite has a hardness of 6.6, and is readily cleavable, but it triturates much less readily than the softer sulphides. The hardness of garnet is over 7, and it does not usually break along cleavage planes; it is harder than the hardest steel used in rolls, and is broken with difficulty. Quartz has a hardness of 7, and possesses no cleavage planes, so that the fracturing of it is due to great pressure or percussive action.

After the material has gone through the coarse rolls, and the finer grains have been tromelled out, then the rational treatment is to gradually reduce these in size by treatment in separate rolls, as at the Proprietary mine. The coarse product gives a good concentrate. The middle products from the jigs consist of admixed grains of sulphides, or of silicates and sulphides, and smaller grains of galena. When these are all brought down to the size desired they pass through jigs or concentrators capable of giving a clean product, fairly clean tails, and a middle product. The sudden steps in reduction of minerals at the Hill to practically three or four sizes cannot possibly give good extractions; yet, on the other hand, since the practice is so commonly adopted, the number adopted must be considered to be the economic limit. Lead concentrates are sought; the zinc and balance of the silver and lead have remained for others to deal with. After the reduction of the middlings from the fine jigs the coarse sands are treated on the Wilfleys; the fine sands on Luhrigs. The heads, middles and tails are made with the first lot of these, but in many cases the middles are returned to the same machine. Now, if the machine is doing its work properly these middles should be caught the second time, as they are the first, and in fact they should accumulate until they overload the machine. This actually takes place to a greater or less extent in some mills, but never appears to be recognised.

By following out a system of regrinding the middlings every time a much better recovery could be made, but towards the finish the slime separators would not pay for the outlay on them and their upkeep. It is to be noted that buddles have been discarded on the field. There is not the same scope for hand picking as in most other fields; at the same time there is too little of it done on the Hill. Much of the material put through the mills only serves to dilute the better class of ore, and give unnecessary work to the various machines. The work done on the whole has served to show that ores above a certain grade in values can be treated at a small profit, but is not at all creditable to Australian metallurgy. The recognised losses have been, and are now, enormous, and very little if any effort has been made to materially reduce them. Mines which have produced millions of pounds worth of metals have gone on with the old wasteful methods while means of doing much better work have been within their reach. Instead of leading they have been following. It is to be hoped that healthy rivalry on new lines of work will be started before long, and the policy of never doing to-day what you can put off till to-morrow, will soon be a reproach of the past.

Taking the published figures of the various companies, the concentration losses will be seen to indicate very imperfect metallurgical work:—

Name of Mine.	Ore Value Per Cent.			Concentrates Value Per Cent.			Recovery Per Cent.		
	Silver. Oz.	Lead.	Zinc.	Silver. Oz.	Lead.	Zinc.	Silver. Oz.	Lead.	Zinc.
South ..	9.17	16.56	16.93	25.12	67.89	7.91	50.37	66.87	7.41
Block 10..	13.8	15.2	17.4	33.92	64.24	8.43	36.83	65.08	
British ..	10.7	17.1	16.8	27.0	62.0		40.0	63.0	
Seconds				24 0	46.0				
Junction ..	8.0	12.14	8.3	24.0	59.2	4 9	33.84	54 8	
Seconds				19.7	44.8	5.9			
Central ..	12.5	17.9	19.0	29.5	61.2	10.5	48.6	68.7	
Tailings	9.2	8.3	27.2	13.4	12.2	38.9	73.7	74.1	72

The figures for the Proprietary mine do not appear in the half-yearly reports, but the recoveries cannot be better than those given, as practically the same methods, with the exception of slime treatment, are employed in all. The South uses more vanners for fine material. The figures are somewhat erratic for the various mines, yet the losses are certainly not over-stated. In no case do the wet concentration methods give a recovery of 50 per cent. of the silver; probably one-third of the value is nearer the mark. The lead recovery may amount to 66 per cent., or two-thirds of the total, whereas the zinc recovered in the concentrates is not wanted at all. Taking the commercial value of the products—that is, clean lead concentrates, clean zinc concentrates, and the silver values distributed in each—it may be said that more than half their value is lost. One notable classification of their dump heaps get over a lot of odium. These dumps, containing more than half the value of the ore, are called "middlings," and these have accumulated to the extent of several millions of tons. The directors of the mines possessing these are always congratulating themselves in their reports as to the asset these are, apparently neglecting the fact that the sooner they are realised on the better will be the profits. They should not be allowed to accumulate, for double handling means more expense, and when all the companies get to work they will glut the market with the class of product they can turn out.

The foregoing remarks are not meant to disparage the work done with present appliances. The managers of the mines and works do their utmost to get the last economic particle of ore, and in this they are most ably assisted by the employes. As an instance of this, Mr. Delprat has notices posted up throughout the concentrating mill—"Look around and see if you can save money; your suggestions and name, if put into this box, will be taken direct to the general manager." On the other mines, likewise, the same enthusiasm prevails. The position, however, is not that the best work is not done with the present machinery, but that the machinery employed is wofully incomplete, while the means of improving methods have been within the reach of those companies, which have done nothing more than concentrate by wet methods for years.

Reports and Balance-sheets.

Another matter which needs calling attention to is the meagre information supplied to shareholders in the reports and balance-sheets of some of the companies on the Hill. From those companies which are marking time but little can be expected, but from those which are in active work shareholders should be in a position to judge for themselves the state of the mine in which they have invested their capital. The information volunteered generally runs in the following lines—"Heavy rain has fallen, and the threatened water famine is averted." "The development work of the mine has been of a satisfactory character; an ore body of considerable extent has been disclosed, proving larger than was anticipated." "The ore found is of the average lead and silver contents." "The stopes have maintained their good quality," and many more platitudes of this useless and indefinite sort. With the exception of the Proprietary, plans are not published with the reports, and even in this case the plans are useless to any shareholders who desire to investigate the value of the property.

The balance-sheets are also as wonderfully vague, and the facts and figures put down give no more clue to the real position of any property than the general statements in the reports.

The work done on the mines and plant is no doubt as accurately recorded; the values are as well known; the cost sheets are as carefully drawn up as at any other place, so that the method of supplying information to shareholders through the reports is not creditable to some companies. The general managers and directors must know the real position of affairs, but the great body of shareholders must be as lethargic as Chinese to receive such documents uncomplainingly.

Why should not these companies state what their reserves are, and their values if these are known? Why should not the costs per ton for mining and development, the cost for concentration, realisation, or smelting be made known? Shareholders would then see what ore could be worked at a profit, and be in the same position as those who have access to all the figures of the company. As it is, the unfortunate outsider who puts his money into Broken Hill may fare as badly as others have fared before, through trusting to nebulous reports as to quantities, reserves, and costs. The cautiously guarded statements which are worthless are in marked contrast to the clear, incisive remarks in the reports of mines managed by competent mining engineers who have control of other properties in Australia. One can only hope that these objectless padded reports will shortly be replaced by those which are of some use. The only way to rapidly improve matters at the Hill is to generate a healthy rivalry in mining and milling—not a milk and water rivalry, giving rise to a mutual admiration society—but such a rivalry as pushed Kalgoorlie to the front, even at the expense of many reputations.

The Sintering of Slimes.

It has previously been stated that no attempt is made at Broken Hill Proprietary mine to concentrate the slimes. The so-called slime treatment only deals with sand; the fine grains suspended in water flow away, and are allowed to settle in large vats. These slimes contain all the minerals which are to be found in the ore, but in the state of very fine division and most intimately mixed. A general analysis would be as follows:—

Galena, Pb S	24.00	
Blende, Zn S	29.40	
Pyrite, $Fe S_2$	3.38	
Ferric oxide, $Fe_2 O_3$	4.17	
Ferrous oxide, FeO	1.03	Contained in rhodonite and garnet.
Oxide of manganese, MnO	6.66	
Alumina, Al_2O_3	5.40	In garnet and kaolin.
Lime, CaO	3.40	In garnet and rhodonite.
Silica	22.90	
Silver	.06	

A later general assay of the lead, zinc, silver and sulphur contents was kindly supplied by Mr. Bradford, the chemist of the company:—

	Before Sintering.	After Sintering.	
Lead	18.83	16.5	
Zinc	16.50	12 0	A loss in weight of about 5 per cent. takes place in the operation.
Sulphur	13.00	7.3	
Sulphur as sulphide		2.9	
Sulphur as sulphate		5.4	
Silver, per ton	18.83	17.4	

The galena, as indicated in a previous article, on account of its softness is the most finely divided of all the products; in fact, the slimes are usually higher in that metal than is the original ore, and it is the last mineral to settle, even in still water.

The slimes which float away in the manner suggested amount to about 1200 tons weekly, or over 10 per cent. of the weight of the ore dealt with, and containing over 11 per cent. of the lead contents. On the other mines similar losses must take place. It is not necessary to indicate the difficulties of dealing with such material by ordinary roasting operations, followed by briquetting and smelting; such methods mean costly and not always efficient work.

The officials at the Proprietary mine have simplified and improved on old methods by adopting a form of heap roasting, which is one of the most interesting and instructive of the metallurgical processes at the Hill. The slimes are now run into trucks as a thick liquid, and are then run along a branch line and run out on to the ground. In the course of a very short time they become partly dried, and are split up with shovels into rough mud bricks. The bricks so made are about 9 inches square, with a maximum thickness of 5 inches. Each man splits from 20 to 30 tons per day. In a few days the bricks become baked by the sun, and probably cemented to a certain extent by the evaporation of water from some of the contained soluble salts. These are then reloaded as sun-dried bricks into the trucks, and are run out to the sintering works. The white cloud arising indicates the position of these works. Through the courtesy of Mr. Delprat, I was permitted to go through these works, and obtained the following details from the superintendent, Mr. L. F. Hayward.

The site selected is on the main tram-line, about 6 miles from the mine. The ground is almost level, and is flanked by a low hill on each side. The heaps—for the operation carried out is a form of heap roasting—are ranged in line. Between the first and second there are two working roads for the trucks carrying in the raw and taking out the finished products; similarly, between the third and fourth are two tram-lines, while outside this is another road for bringing material to and from the pug mill. The length of these roads is about 1500 feet, with a grade of about 1 in 100. These sidings between the heaps are practically on ground level, the trucks have side doors, and all the bricks are discharged by hand.

The heaps are rectangular in shape, 220 feet in length, 20 feet wide, and 7 feet 6 inches high; they may be longer or shorter, but the other dimensions are adhered to. The batter of the sides and ends between ½ to 1 and 1 to 1. The ground is first marked off with a number of strips of free-burning wood, on which the lumps of slime are ranged so as to form small flues. The main flue runs along the centre of the heap, terminating five feet from the end; a diagonal flue starting from near one corner of the heap joins the end of the main flue. The next flue on this side is 10 feet from the end, and runs at right angles to the central flue; other flues along the side are similarly spaced. On the opposite side the first flue starts at five feet from the end, and junctions with the extremity of the central flue; other flues along this side are spaced 10 feet apart, and thus alternate with those of the other side. The accompanying illustration (Fig. 1) gives the position of these flues. The heaps are built up as shown in Fig. 2. First, the wall furthest from the railway track is built up. Building proceeds along the length of the heap, until this side is 7 feet 6 inches high; the rest of the heap is then built towards the side nearest to the truck. Owing to the fragile nature of the slime lumps they have to be handled with much care; they are thrown by hand to an intermediate man, who catches and throws them on to another, who hands them to the builder. No great care is taken in laying them, so long as a fairly solid heap is built up with large blocks; subsequent operations are not interfered with. There are always enough open spaces to act as channels between the blocks. In transferring the material from the trucks a certain amount of fines are produced—these amount from 15cwt. to a ton for a seven ton truck, or from 10 to 15 per cent. The rough material from this is forked up and thrown over the top, the dust is made into a paste with water put through a pug mill, and the sides and ends of the heap are plastered over with this, every crack being filled, and the surface trowelled off. When the heap is burnt the outside crust for about a foot is not sintered, but converted into a yellow coherent material, which is full of sulphates. This would probably amount to about 10 per cent. of the whole heap. Such half-burnt material goes into the next heap. The dried lumps of slime do not contain much more than one per cent. of water. After a heap is built, it is ready for burning. This is effected by lighting the fuel in the side flues. The heaps run slightly to the west of north, and as the prevailing winds blow across this direction, the products of combustion are carried across the heaps. The first flues set going are those on the leeward side, which may be started in the morning: at 4 p.m. the windward side flues may be lit, and the heap allowed to burn until the following morning. Firing is kept up by inserting firewood into those small flues until the material round the fire holes clinker. Loose bricks are then inserted into the flues on the windward side, in order to exclude as much air as possible. The air is also partly excluded from the leeward side by loose bricks. The

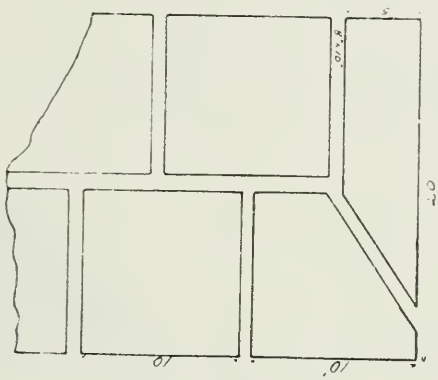


Fig. 1.

first stage of the operation is sweating; in other words, the heat from the burning fuel and pyritic material causes the moisture, which is contained in the slimes, to be driven out; afterwards sulphur is distilled out, and as the temperature rises a white crust of various salts may be seen coating the outside of the heap. The time taken for the charges varies, but in a good kiln the fire reaches the top at the sides in 16 hours, and at the centre in two days. After the action has become well started no further attention, beyond that regulating the air supply through the flues, is necessary. The fuel consumed at first amounted to as much as five per cent., but has now been reduced to less than $1\frac{1}{4}$ per cent. In eight days the heap has burnt itself out. In burning the heap contracts and sinks irregularly, the settlement being from 2 feet to 2 feet 6 inches in a 7 feet high kiln. The products appear to sink in places, leaving irregular cavities on the surface; from these issue sulphur dioxide, a small amount of sulphur trioxide, and certain volatile salts, formed during the reactions. The temperature, even at the surface in these cavities, is from 500deg. to 700deg. C., while in the interior of the heap it is much higher. A curious sublimate is formed in all the surface cavities, and encrusts the plaster of slime on the sides of the heap. Owing to its yellow color, and the fact that sulphur dioxide was escaping, the workmen considered it to be sulphur, but as a matter of fact, it is a basic sulphate of zinc, which, like nearly all zinc compounds, is yellow while hot, and white when cold.

As soon as the heap has cooled down sufficiently, the sides are stripped. These are a brownish yellow color for about a foot in depth. This is known as the sulphatised material. Solid lumps may be selected from this, but if kept for a few days will split and crumble to powder like quicklime. This is due to the absorption of water by dehydrated salts. The interior of the heap consists of lumps usually weighing many hundredweights, but often tons, of a black fused or sintered product. Much of this has the appearance of a fused basic slag, separated from other pieces by a sintered product. There was practically no dust or very fine material left. Trucks were brought up alongside, the siding sprayed to harden the surface, this doing away with the necessity for flat sheets, the sides of the solid and sintered heap broken down into sizes fit for handling in the trucks, into which it was at once thrown, and sent down to the smelters at Port Pirie. Each heap holds about 1000 tons, and three are always being dealt with. More than a dozen are built; some waiting for firing, some being fired, others fired and cooling On account of the judicious arrangement

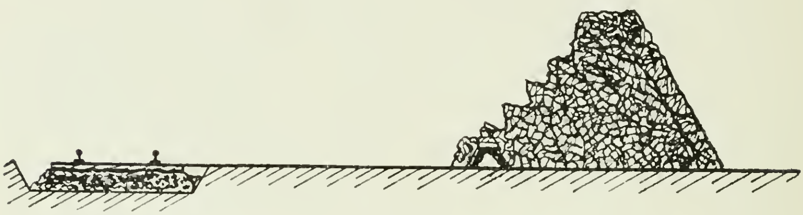


Fig. 2.

of these heaps, work has only been blocked for two hours in the eighteen months of work. About 275 tons are sent away daily, or 1500 tons per week. The number of men employed was 90, and since the only cost, beyond that of cutting up and carriage of the slimes, is in building and pulling down the heaps, the cost per ton for labor alone cannot exceed one-third of a man's wages per day. The advantages over roasting these slimes mechanically and then briquetting them is that there is no loss of dust, the sulphur is reduced to the same extent, a splendid furnace product is made, and the cost is 3s. per ton less. Against this there is a considerable volatilisation of lead, zinc, and silver in heap roasting. The work could probably be cheapened somewhat by mechanically handling the sintered material, and adopting any safe device for lessening the handling of the slime blocks. As before stated, about 5 per cent. loss in weight takes place in sintering. Part of this is due to the exchange of sulphur for oxygen, but on the other hand there is a gain when sulphides become sulphates. From 50 to 60 per cent. of the sulphur is volatilised, part as the element itself, but mainly as sulphur dioxide, with traces of the trioxide. According to the analysis given, raw slimes, carrying 16.5 per cent. of zinc, are reduced 5 per cent. in weight, and then carry only 12 per cent. of zinc. The loss of this metal in that grade of ore is therefore over 5 per cent., or nearly one-third of the total zinc present. The loss of lead is three per cent. on 18.5 per cent. slimes, or about 17 per cent. of the total lead. The loss in silver is 2.3 per cent. on 18.8oz. ore, or about 12 per cent. of the metal originally present.

The chemistry of heap roasting of slimes is not fully enough understood to write positively about it. In view of the late interesting discoveries in the Huntington-Heberlein process, and the corol-

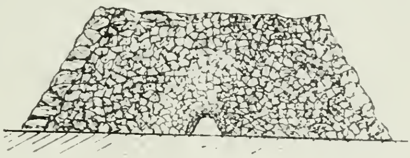


Fig. 3.

lary to it, the Carmichael-Bradford process, Mr. Carmichael, of the Proprietary mine, proposes the following reactions as most probable:—

At a dull red heat, rhodonite is converted into manganous oxide and silica. Probably Mr. Carmichael means that at a dull red heat, in presence of oxygen, rhodonite loses some of its manganese, which becomes oxidised, a higher silicate forming. Certainly heat alone would not decompose silicate of manganese into the oxides mentioned. Many of the blendes of Broken Hill contain manganese sulphide, so that if manganese were required to explain the reactions given, this source of the metal would suffice. The man-

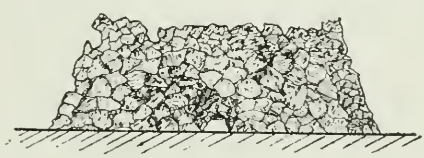


Fig. 4.

ganous oxide, if at first formed, becomes oxidised to the more stable oxide—

$$6MnO + O_2 = 2Mn_3O_4$$

Any calcite present in this ore, or even some lime compounds in easily decomposed silicates, become converted into sulphates by the joint action of sulphur dioxide and oxygen. Mr. Carmichael holds that given these compounds, the following reactions will occur:—

$$PbS + 4Mn_3O_4 = PbSO_4 + 12MnO$$

$$PbS + CaSO_4 = PbSO_4 + CaS$$

The MnO and CaS are then regenerated into the higher oxide and sulphate respectively, and thus act as oxygen carriers to the sulphides of the heavy metals. Further, given sulphate and sulphide of lead, metallic lead will be produced—

$$PbS + PbSO_4 = 2Pb + 2SO_2$$

The lead at once oxidises and the oxide of lead reacts with more galena, giving rise to the same products as in the last equation—

$$2PbO + PbS = 3Pb + SO_2$$

With excess of lead sulphate, the following reactions are given:—

$$PbS + 2PbSO_4 = 2PbO + Pb + 3SO_2$$

$$PbS + 3PbSO_4 = 4PbO + 4SO_2$$

Whether these last reactions take place is doubtful. It is more probable that the molecule of sulphide would react with the molecule of sulphate in contact with it, giving rise to metallic lead and sulphur dioxide, and that excess of sulphate would not change. Of course, in an atmosphere of SO_2 the reverse actions would go on.

While the explanations given would account for the desulphurisation of the lead, they do not account for that of the zinc. The zinc, on slow oxidation which goes on in the heap, becomes sulphate of zinc, which as the temperature rises becomes converted into basic sulphates, sulphur trioxide escaping to attack some unoxidised material. The sulphate and sulphide of zinc react on each other readily enough to produce zinc oxide, with the evolution of sulphur dioxide—

$$ZnS + ZnSO_4 = 2ZnO + 2SO_2$$

The silver present is probably converted into sulphate after the manner of kernel roasting. The partial desulphurisation may be accounted for on general lines as due to the oxidation of the sulphides; probably the lime and manganese oxides play a small part in the process, but it is just as likely that this material would behave in the same way if they were not there. The very easy fusibility of garnet and rhodonite, which constitute the bulk of the gangue, coupled with the fact that oxide of lead is readily dissolved by these fused silicates, is sufficient to account for the sintering at a comparatively low temperature; and, finally, sulphate of lead, and even sulphate of zinc, in presence of these fused materials, will be split up into their oxides, sulphur dioxide and oxygen escaping.

The comparatively large losses of the three metals has not been satisfactorily explained. Metallic zinc and metallic lead are easily volatile, but no satisfactory proof of the formation of the former has been adduced, although the latter may be readily enough found in roasted products. The explanation based on the volatility of some of the compounds in gases produced is not wholly satisfactory. There is always in this case excess of inert nitrogen; the volatility of silver also needs explanation. If the water used in milling contains appreciable quantities of chlorine, then it is easy to understand a considerable loss, for by evaporation of the slime product the salt would be concentrated and most evenly mixed throughout the material.

Broken Hill Ore Treatment.

The Potter Process.

In 1901 Mr. C. V. Potter took out a patent for the recovery of sulphides from their ores by means of a separation effected by dilute acids. Mr. Potter found that certain sulphides, when treated with highly dilute acids and warmed to near boiling point, rose from the rest of the ore, and could be either skimmed off or floated away. Almost every assayer, in dealing with acids, especially nitric, must have seen the separation of sulphur and a portion of sulphide, which obstinately floats on top; yet when this very process was seriously put before some of our most brilliant metallurgists, the scheme was looked upon as too chimerical for ordinary practice. The explanation of the process is that, if there are mixed sulphides in an ore, on warming with dilute acid gaseous bubbles will be generated and attach themselves to certain particles, and in course of time so increase in size as to carry or balloon them to the surface. In some cases these bubbles are generated from the minerals raised; in other instances, they are generated from admixed products, but the result is the same, the bubbles become attached, and if the particles are small enough they are carried to the surface. Some work has been accomplished, but much remains to be done in working out the details of this most simple and yet original method. After the discovery, Block 10 appears to have experimented with it, but were evidently unsuccessful, since they dropped it. It was not until Mr. G. A. Goyder, formerly analyst to the South Australian Mines Department, observing when heating some Broken Hill sulphides in a test tube filled with dilute sulphuric acid, that the bubbles, with their sulphide load travelled to the surface of the liquor, broke, and then fell back again, by inclining the tube as in Fig. 2, the sulphides rose, but travelled up the smooth side of the tube until they met the surface, when, as before, the granule of sulphide fell off, but this time it fell vertically until it met the side of the tube, down which it glided. The inference was that if there was a pocket in the under-side of the sloping test tube that the sulphides should gather in the pocket. Mr. Goyder had a tube made of this description, and it answered admirably. The next step was to devise a machine based on this principle. It was easy enough to effect a separation in any vessel, but it was not as easy to make the process continuous. After numerous experiments with Mr. E. Laughton, the managing director of Block 14, a machine was devised. It was first made of copper, but the acid, in conjunction with the oxygen of the air, acted on it so rapidly that it was destroyed within a fortnight. Lead was tried, but it buckled. Other metals were more or less unsuitable, until at last a well-known acid-resisting alloy of antimony and lead was found to be eminently successful. The construction of the machine may be readily followed from the diagrams. Fig. 1 is a longitudinal elevation of the vessel used. It is a rectangular, punt-shaped, shallow box, with a bottom curved upwards at the feed end, and sloping at the discharge end. This vessel holds the dilute sulphuric acid. Three sets of bearings, one at each end, and the third where the upward slope of the vessel commences, carry shafting. On this is fixed polygonal-faced wheels, three large on each bearing. Over each of those wheels, in the same place, passes an endless link chain. The three chains are connected horizontally by means of circular rods. This system of wheels and chains is really to carry the horizontal rods which pass along under the first wheel, thence under the second, scraping ore and gangue along the bottom of the vessel. The rise in the bottom of the tail end of the box enables the scraper to remove the sand from the liquid and discharge it over the edge. The scraper returns around the end wheel over the vessel, and enters it as before. The arrangement for catching the sulphides is based on the test tube experiment. The cross section of the box is shown on

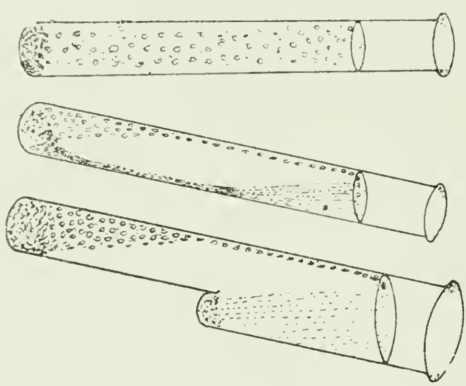
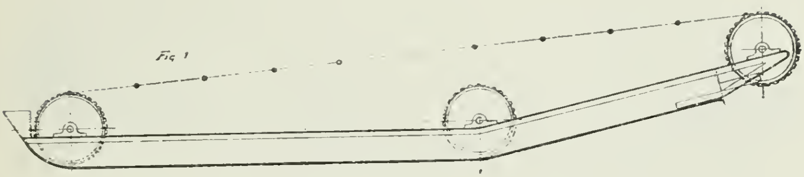


Fig. 2. The thick line on the bottom and sides denotes the outline of the box. The inclined divisions are so placed that any particle of sulphide rising from the bottom will glide upwards along the underside of these inclined boards, which are all immersed in the liquid, pass through a small opening left, and drop down into the V-shaped launder. They are removed from these by V-shaped scrapers, which are attached to a smaller wheel on the same shafting as the larger. The chain which passes over the three smaller wheels in this case is not connected to the chain passing over the other three. Each has its own set of scrapers, shown by the upward lines on the lower chain, Figs. 1 and 3. These V launders follow the slope of the bottom of the vessel, but they project beyond it, consequently the sulphides are carried further along and dropped into a separate receptable. The V launders and the deflecting plates are made of wood, and are all in one piece, which fits the box from end to end, but allows sufficient space below for the scrapers. The drawings explain themselves. In Fig. 1, the upper dotted line indicates the endless chain carrying the scrapers for gangue; these are indicated by the black circular dots. Fig. 3 shows a section of the same. The upper dotted line, with the circular dots, is as before; the lower dotted line, with the upright pieces shown above,

is the chain carrying the scrapers for the sulphides separated. The trough in which the sulphides collect is indicated by the second thick line, the lowest thick line being the bottom of the machine. The two sets of wheels are also shown in this, as well as in Fig. 2, where the second and fourth wheels are shown, and the V-shaped scrapers on the chains above and below.

The machine is worked as follows:—It is filled within an inch or two of the sides, with dilute sulphuric acid, generally from 2 to 3 per cent. The solution is heated by a fire underneath, and as soon as the temperature approaches 190deg. F., an even stream of pul-



verised ore is fed in. Certain sulphides at once are superficially attacked, and rise and get into the V-shaped launder. They are scraped out of these, while the ore is also carried forward, so that by the time it has reached a few feet the sulphides are all separated. The gangue is carried by the schaper up the inclined end of the table; a fine spray of water washes any adherent acid off. Similarly the sulphides are washed, the wash water running back to the bath to make up for that lost by evaporation. At the same time, an amount of sulphuric acid, equivalent to that used up by the ore, is allowed to fall in at the head of the vessel, so that in this way the strength of the solution is kept constant. The strength of the sulphuric acid used is from 3 to 3.5 per cent. The amount con-

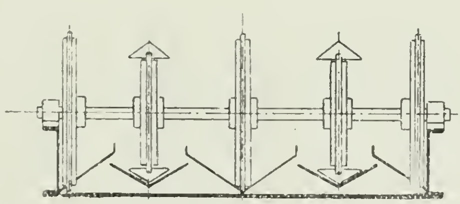
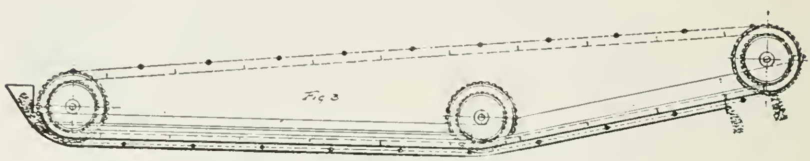


Fig 2

sumed varies with the ore and gangue, but it would appear that from 40lb. to 50lb. of sulphuric acid is used per ton of ore treated. Mr. Potter discovered that all the acids he tried were effective in causing a separation, and claims them in his specification, but of course chose sulphuric as being the cheapest available. From a commercial point of view, the composition of the gases evolved is a matter of little importance. The action will take place if only sulphuretted hydrogen is evolved, but even when carbon dioxide is evolved it appears to be of service in attaching itself to certain sulphides, and raising these to the surface. The same volume of carbon dioxide or sulphuretted hydrogen, or any similar gases, will require exactly the same weight of sulphuric acid to produce it, so that a small quantity of calcite, such as exists in most tailings heaps, may be of positive advantage. On theoretical grounds, we can find out the amount of acid required for the work. Assuming that zinc blende is the sulphide to be recovered, and that H_2S is the gas which is generated and lifts the sulphide,

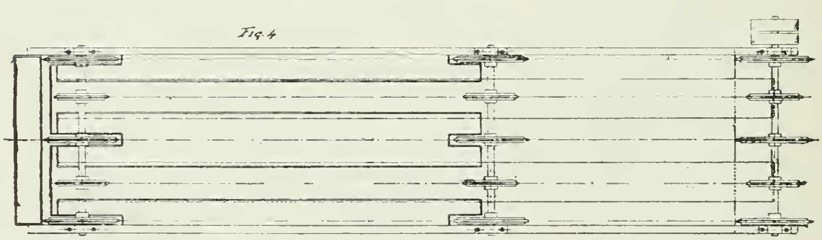
$$H_2SO_4 + FeS = FeSO_4 + H_2S,$$

Ninety-eight grammes of sulphuric acid will produce 22.3 litres of sulphuretted hydrogen at 0deg. C. and 760 m.m. press. Taking the



sp. gr. of blende as 4, 4grms. of blende displace 1c.c of water, so that to float this 3c.c of gas are necessary—neglecting the weight of the gas—so that 22.3 litres will float 29,760 grammes of blende, or, practically speaking, 300 times the weight of the acid decomposed; or about 7.5lb. of sulphuric acid should generate enough gas to float a ton of blende. By heating the solution to near boiling point, on account of the expansion of the gaseous bubbles, a smaller quantity of gas is required, and therefore a lesser amount of acid. In fact, about 5½lb. should do the work. It will be seen that if gas is produced by the decomposition of the acid, that future working will show a greatly diminished consumption.

That this process has passed the experimental stage may be seen from the work carried on at Block 14. I was courteously given all



details by Messrs. Winter and Laver, who were working this most original and interesting process. Experiment after experiment was carried out, advances being made each time, so that the successful working of the process is in a large measure due to them. Over 2000 tons of tailings had been treated at the date of my visit, and a clean zinc concentrate obtained, while the small amount remaining in the tails was due to coarseness of materials. It would appear, with the plant described, that the most suitable size is from a 35-mesh downwards. The finest slimes will rise most readily, but there is trouble owing to the muddiness of such mixtures, so that a certain proportion of fine gangue is also brought over. At Broken Hill, most of the true slimes have been eliminated in the previous ore-dressing treatment, so that the heaps contain a large proportion of residues from the jigs and sands from the buddles. At present, the latter are being screened through a 35-mesh, and the fines are put through the machines. Some simple crusher will have to be devoted to the coarse material to reduce it to the size required. Probably some disintegrator, which would pulverise the easily cleavable blende, and leave the garnets and silica coarse, would be the most suitable appliance.

The weak parts of the Goyder-Laughton are the chains, which are immersed and have to travel through hot acid solutions. These gave considerable trouble at first; bronze and other alloys were tried, then wood, but the eyelet holes in these elongated into ovals in a short time, so the lead-antimony alloy was made, which appears to answer well. The capacity of a machine, such as the one described, is about one ton per hour. In other words, each unit of this plant is of greater capacity, is less complicated, and requires as little attention as a vanner, so that from this point of view, even small machines are a commercial success if they can be run without stoppages. Mr. Berry, of Melbourne, invented a machine, which does away with all chains and pulleys immersed in liquids. This consists of a shallow cylindrical vessel, having a trough around its circumference. The inner edge of the trough is formed by the outside of the cylindrical vessel, and the outside edge of the trough is at a higher level than the inside. An inverted cone is placed in the liquid, thus serving as a deflecting surface. Attached to this cone are scrapers, which move the sand around in the inner vessel also, which move material forward in the trough. By feeding in ore at one part of the circumference of the inner vessel, and heating the liquid contained in it, the sulphides rise up the deflecting cone, and overflow into the circular launder or trough, the level of the liquid being over the rim of the inner vessel. As the sulphides pass over, they are swept round by the scraper, and are dropped through a slot into a receptacle, whence they are discharged through a spigot. Similarly the gangue moves round until near the point of feed on the circumference, whence it also is discharged. One result brought about by this machine is that a differential separation is brought about. The pure blende comes up at once, and is carried over; further on it contains more lead, and at the discharge more still. From assays by myself, made on the products of this machine, the blende from the first compartment runs 51 per cent. zinc, down to 47 per cent. from the last; the lead contents of the first go 2 per cent., and from the last 7 per cent. Other machines have also been devised, and while in all cases as much clean zinc concentrates may be obtained as it is possible to get by any process, yet the machine of the future will have very few parts which are liable to get out of order, and which will make use of flowing solutions to effect a separation.

The Delprat Process.—Sulphuric Acid Manufacture.—The Carmichael-Bradford Process.

The year after Mr. C. V. Potter took out his patent for the recovery of sulphides from their ores, by means of dilute acidulated solutions, Mr. G. D. Delprat applied for a patent for a process to effect the same purpose. Mr. Delprat claimed the use of salt cake, or a by-product in the manufacture of hydrochloric acid. Salt cake is the commercial name for this crude sulphate of sodium. It is made by treating common salt with sulphuric acid. If less sulphuric acid is added than is required to decompose the whole of the salt, then sodium chloride and sulphate of sodium will be the main products; but if excess of sulphuric acid is used, then bisulphate of sodium or acid sulphate of sodium and the normal sulphate will be the main products. In practice the salt cake obtained contains from 95 to 96 per cent. of sodium sulphate, and the remaining 4 to 5 per cent. is mainly sodium hydrogen sulphate; traces of undecomposed salt and other materials remain. In some specifications Mr. Delprat adds sulphuric acid to this solution; in others he claims sodium bi-sulphate as the active ingredient. The crude sodium sulphate is added to the solution until the sp. gr. is stated to be approximately 1.4; in other words, the solution must be practically saturated. Mr. Delprat's solution, therefore, is made up with two objects in view, the first being to raise the particles of sulphides by the attachment of gaseous bubbles, and the second to densify the solution so as to enable each bubble to support a greater load.

A large experimental plant, said to be capable of dealing with 1000 tons per week, has been erected, and work is going on with what are said to be highly satisfactory results. Owing to litigation now going on betwen Potter and the Proprietary Company, Potter stating that the so-called salt cake process is an infringement, I cannot say anything about the working of the latter process on the Proprietary mine. From specifications, also from a small plant used on another mine, enough may be said to indicate the process. The solution of the substances before mentioned is heated to near boiling point, as in the Potter process, and the pulverised sulphides are fed into an inclined launder. The gangue at once slides down to the bottom, and is removed through a spigot. The zinc sulphide is superficially attacked, and floats. The flowing stream of liquid carries the sulphides over inclined plates in the launder, whence they are separated from the gangue and may be removed.

One strange feature about the "salt cake" process as carried on at the Proprietary mine, is that a large sulphuric acid plant has been erected to supply the necessary acid for working it. The sulphur dioxide is produced by the Carmichael-Bradford process; and from this sulphuric acid is made in the ordinary way. Gypsum and sulphides of lead and zinc, with a small amount of siliceous material, is made into small balls, placed in a converter and heated; afterwards a current of air is blown through, when sulphur dioxide is given off, and the action continues as long as excess of sulphur is present. The gypsum required for the process is obtained within five miles from the Hill. It consists, as is well known, of hydrated sulphate of lime, having the formula $CaSO_4 2H_2O$. On being heated to a temperature from 120deg. to 130deg. C. it is converted into Plaster of Paris.

$$2CaSO_4\,2H_2O = 2CaSO_4\,H_2O + 3H_2O.$$

On raising the temperature to 200deg. C. the anhydrous sulphate is formed. This product, from which nearly 21 per cent. of water is driven, when moistened with water sets slowly and returns to the original state. After dehydrating, the powdered sulphate of lime is mixed in the following proportions with slimes and concentrates: Slimes, 3 parts by weight, concentrates 1, and gypsum 1, or 60 per cent slimes, 20 per cent. concentrates, and 20 per cent. gypsum, the last becoming 16 or 17 parts when heated, owing to loss of water.

The average composition of these would be—

	Slimes.	Concentrates.	Gypsum.	Average.
Galena	24	64	—	27
Blende	30	14	—	20
Pyrites	3	—	—	2
Ferric-oxide	4	—	—	2.5
Ferrous-oxide	1	—	—	1
Manganese-oxide	6.5	—	—	4
Alumina	5.5	—	—	3
Lime	3.5	—	32	8.5
Silica	23	—	—	14
Sulphur-trioxide	—	—	33	7

The sulphur contents of the mixture vary from 13 to 15 per cent. The whole of these materials are mixed and moistened with water, then passed through a pug mill. The small amount of water used serves to partly set the product, owing to the sulphate of lime becoming partly hydrated. This also serves to bind the mixture together. The whole lot is spread out on a drying floor and broken into pieces not exceeding 2 inches in diameter. It is allowed to stop there until all surface moisture has been evaporated by the joint action of wind and sun. When dry it is shovelled into the converter. This is a cast-iron pot of similar pattern to those used in the Huntington-Heberlein process. A cover is luted on to the converter. This cover is connected on to a horizontal flue by means of a telescopic pipe. Three converters in line are attached to the same flue. A couple of nitre pots are placed further along the flue; these connect with the leaden chamber, having a capacity of 40,000 cubic feet.

It is necessary to start the action in the converter by heating it to about 400deg. to 500deg. C. This is done by first feeding in a small amount of coal and igniting it. On turning on a small blast of about 2oz. pressure the mass warms up and the coal is rapidly burnt out. Water is given off abundantly, at first. As soon as the lower porton becomes dull red the sulphides start to oxidise, the mass rapidly rises in temperature, and sulphur dioxide is evolved in abundance. The mixed gases escaping, if the action were allowed to go on freely, would contain 14 per cent. of sulphur dioxide. For successful work they must not exceed 10 per cent. In other words, the reaction $2SO_2\ O_2 = 2SO_3$ shows that two volumes of sulphur dioxide require one volume of oxygen in order to produce the trioxide. Air contains about 21 per cent. of oxygen, so that if two-thirds of this were consumed in burning sulphur to the dioxide only one-third would remain for the conversion into trioxide. In other words, the reaction would be a quantitative one, and no oxygen would be left over. This would mean that towards the finish of the conversion the reactions would proceed so slowly that gases would be discharged from the chamber uncombined; further—although this would not apply to the Broken Hill plant—that there would be no oxygen for the nitric oxide to combine with, a condition necessary for its absorption by strong sulphuric acid. When sulphur alone is burnt the amount of sulphuric dioxide generally allowed is about 11 per cent., but in the case of pyrites or other sulphides a portion of the oxygen admitted is used up in oxidising the metal. For example, assuming galena to be oxidised to lead oxide, and sulphur dioxide to be given off, then for every volume of sulphur dioxide produced, or for every volume of oxygen used up in producing it, one-half that volume of oxygen is fixed by the lead, so that with even 10 per cent. of sulphur dioxide escaping there would be about two per cent. of oxygen left over at the finish; assuming, as is the case, that all the air required is passed through the converter. By having three converters the action is fairly uniform, and may be made continuous. Owing to the absence of Gay, Lussac, and Glover towers, there will be a large consumption of nitric acid in the manufacture, but these desirable adjuncts are to be added in the near future. The acid produced is chamber acid, or of about 68 per cent. in strength. It is of no use concentrating it, since it is only used in a highly diluted condition. The highest credit should be given to Messrs. Carmichael and Bradford, both employes of the Proprietary Company, for having worked out this most interesting and novel process, and having applied it so successfully to the manufacture of sulphuric acid. As I have already stated, in first dealing with the Huntington-Herberlein process, the reactions which take place in roasting sulphides with limestone is that sulphites, then sulphates, of lime forms. This material, when placed in a converter with semi-desulphurised ore and air blown through, gives rise to plumbate of calcium, with the evolution of sulphur dioxide. What Messrs. Carmichael and Bradford have shown is that, by taking sulphate of lime and mixing it with the same class of material as is used in the Huntington-Herberlein process, the preliminary roast may be dispensed with. The reactions said to take place are those previously given in connection with the Huntington-Herberlein process as used in Tasmania, Cockle Creek, and Port Pirie. The discovery is a valuable one, even for the application of producing sulphur dioxide in a concentrated form. There is little doubt but that the intimate mixture of materials in this case leads to good results. There is said to be no dust produced, while, what is more interesting still, no volatilisation of metals takes place. The sulphur which remains in the residual product is slightly less than originally introduced with the gypsum. The converter is emptied by hoisting the cover, inverting the converter on its trunnions, when the sintered or semi-fused product falls out. This, like the Huntington-Heberlein material, goes to the smelters.

The success of the "salt cake" process, and also the manufacture of sulphuric acid, has opened up great possibilities for the Broken Hill mines. Mr. Delprat recently stated that the mines of Broken Hill turned out ore annually which contained 200,000 tons of zinc, or more than one-third of the total zinc annually produced. Six hundred tons of zinc concentrates obtained by the salt cake process had already been shipped away, and regular shipments would continue. The success of this process largely depended upon the cheap production of sulphuric acid. This could only be carried on locally with the material at hand. The present sulphuric acid plant, which is capable of turning out 35 tons of chamber acid per week, would be added to so as to give a production of 70 tons per week. In addition to this, sulphuric acid works will be erected at Port Pirie, which will give a weekly production of over 200 tons of chamber acid. This will be used in the manufacture of superphosphates from the phosphate beds of Yorke Peninsula (S.A.).

It is to be hoped that this sanguine forecast will be realised. In Mr. Delprat this company is fortunate in having a scientific man at the head of affairs, if science means organised common sense. The apparently simple discovery by Mr. Potter in 1901 has had far-reaching effects; the wonder is that it was not pushed forward long ago, as it should have been by some of those companies so vitally interested.

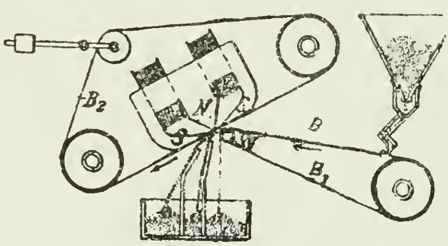
Magnetic Concentration.

The modern science student is apt to seek for commercial results. To him all scientific work should be capable of being applied. While teaching in technical schools and schools of mines should be so directed that the student will not waste his time over subjects which have only a remote bearing on the work he will have to do subsequently, yet it is essential that science, for science sake, should be investigated in all its branches. This function, as matters now stand, belongs to a university. The technical student must rely on the specialist in pure science. The pure science of to-day becomes the practice of to-morrow. No matter how capable the practical man is in doubt or difficulty, he appeals to the literature of the specialist. The work done in pure science is the only safe bed-rock for the applied science to be built upon. That a particular mineral had magnetic properties was known as far back as literature goes, but the lodestone was only a term, and magnetism was only this name—a curious property possessed by certain substances—until the genius of Faraday laid the foundation for the modern science. The properties of an ordinary magnet are so well known that it is only necessary to say that, if a bar of hard steel be magnetised, it will maintain this property for a long time. If dipped in iron filings it will be found they cluster at the ends and do not adhere at the middle. If a sheet of paper be placed over the magnet, and some soft iron filings be sprinkled on it, and then tapped, the filings will arrange themselves in curved lines. The amount of filings and their direction give an indication of the direction and the intensity of the force produced by the magnet at the various points around it, so far as this plane is concerned. By having a square or rectangular ended bar, the filings will cluster round the edges; by having a pointed end, the filings follow the external shape of the magnet. Next, if a piece of soft iron, which cannot be magnetised permanently like steel, is brought into contact with a magnet, or even close to it, the soft iron becomes a magnet while in this position. If the iron is removed, it is no longer a magnet. If, while the iron is still near the magnet, or is still a magnet by induction, some iron filings be dropped close to the end of the iron, they will strongly adhere, but fall off the instant the soft iron is removed; this principle is made use of in magnetic concentrators.

Owing to familiarity with magnets and with iron, a one-sided view of what magnetism really is is often gained. Iron, nickel and cobalt are said to be magnetic; other substances non-magnetic. Faraday long ago showed that nearly all substances were capable of being acted upon by a powerful electro-magnet. Such substances as iron, which are attracted by either pole of a magnet, were called para-magnetic, whereas a metal like bismuth is repelled by either pole of a magnet, and hence is called dia-magnetic. This does not only hold for solids, but also for liquids and gases. For instance, oxygen is para-magnetic like iron, while nitrogen is almost neutral; the air, therefore, is para-magnetic. Water, on the other hand, is dia-magnetic. Not only are iron, nickel and cobalt para-magnetic, but also manganese, titanium, platinum, and other metals, and many of their compounds, while bismuth, antimony, lead, gold, silver and copper, quartz, and many other substances, are dia-magnetic, or repelled by either pole of a magnet. Not only metals, but other substances, are acted upon one way or another, and in the case of crystalline minerals, Tyndall discovered that they are susceptible of magnetic induction to different degrees in different directions. While it is true that there is an attraction by a powerful electro-magnet for all para-magnetic substances, yet the amount varies enormously. If steel be taken as 100,000, then magnetite would be 65,000, siderite or carbonate of iron 120, hematite 65, and limonite about 50. In other words, an electro-magnet would have to be 1000 times as powerful to attract hematite as one which would attract the same weight of magnetite. Not only are these minerals attracted, but also garnet, the variety at Broken Hill containing a large quantity of ferrous oxide, with some ferric oxide; also rhodonite, which is mainly a silicate of manganese, and blende, which contains a small amount of sulphide of manganese, as well as sulphide of iron. By having a electro-magnet whose intensity can be varied at will, it is possible not only to separate para-magnetic substances from dia-magnetic, but to separate the various para-magnetic substances from each other. This is actually done at Broken Hill, and it opens up a new field in metallurgical concentration which will enable ores not separable by gravity processes to be dealt with. The first types of machines were only used to separate strongly para-magnetic substances, such as magnetite, from gangue. By giving a rapid roast to many sulphides, or sulph-arsenides, of iron, they are converted into magnetic sulphides or magnetic oxides. By heating carbonate of iron, or giving a reducing roast to hematite, these substances become magnetic oxide, and are much more powerfully attracted than the original minerals. One simple machine for dealing with such material consisted of a brass drum, with horizontal projections on it. Inside the drum, but in a fixed position, were placed a number of electro-magnets, having their poles under the surface of the brass shell of the drum. The ore was fed on to the drum, as water is on to a breast water wheel. The magnetic minerals at once were brought under the influence of the magnets, and these adhered to the surface of the drum and were carried upwards as the drum rotated, while the gangue dropped into a bin below. The magnetic minerals were carried round beyond the range of the magnets, which only extended over about 1-6th of the circumference of the drum, near the feed, and carried on the horizontal projections over to the other side, and dropped into a bin. Another appliance, by Edison, was to place powerful electro-magnets so that the magnetic particles in a falling stream of fine ore would be deflected towards the magnets. A partition may be fixed below, so that the deflected material lies on one side, and the gangue, which has dropped straight down, lies on the other.

None of these machines were of the intense type, necessary for the separation of faintly magnetic substances. In order to secure a strong magnetic field, it is necessary to concentrate the lines of force. This is done by making the magnet wedge-shaped, and placing the thinned edges face to face, with a small air space between. Faraday, in testing the magnetic properties of various substances, did practically the same thing. He tapered off his electro-magnets into conical points, and brought these poles close up to each other. Wetherill was the first to turn this to practical account. He invented three classes of machines, known as the parallel, horizontal, and inclined separator respectively. The first machine consisted of two powerful electro-magnets of horseshoe pattern, with their poles downwards. An endless rubber belt runs horizontally, and the poles of the magnet are almost in contact with this. Below this runs a corresponding belt in the same direction, but a little slower: the belt is narrower than the upper one. Dry pulverised ore is fed on to the lower belt, and is carried along until it reaches the magnets; the magnetic particles are carried on to the upper belt, and travel diagonally along the pole, which is set at an angle of 40deg. with the belt, until they reach the edge, when they pass out of the influence of the magnet and drop off past the edge of the lower belt into a bin below.

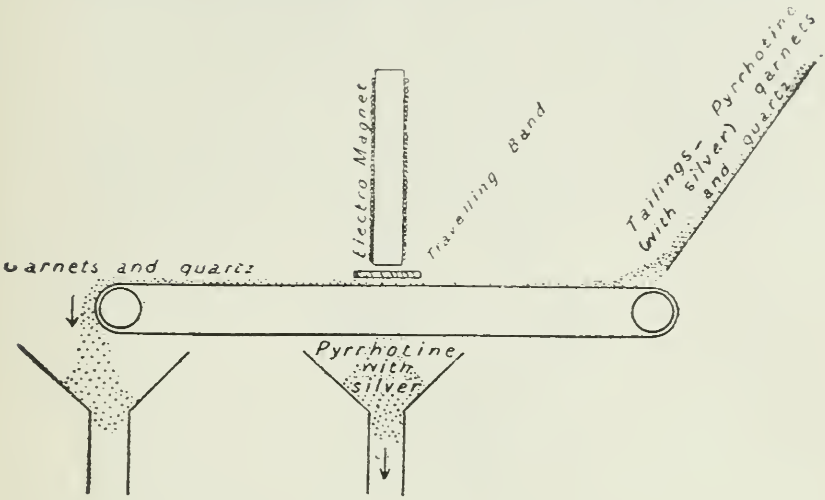
The second machine had two broad wedge-shaped magnets, placed horizontally, with their ends less than an inch apart, thus giving a very intense field. A horizontal belt, driven by a pulley, travelled round the polished wedge-shaped end of the magnet. Ore was evenly fed on to this belt; as soon as the weakly magnetic particles reached the edge of the magnet they clung to the belt, while



the gangue dropped straight through. The magnetic particles were carried round for a fractional space, and fell on one side of an adjustable shutter, thus being divided from the tailings.

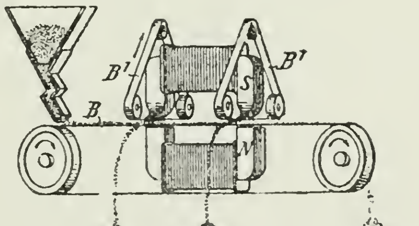
The final type of machine of this description was made in Germany by the owners of the Wetherill patents. A diagram of the machine is given. An intense magnetic field is given by the central pole (N), and two lateral poles (S). The lines of force are concentrated across the plane of the belt (B_2). Material is fed in from a hopper on to an endless belt (B), which carries the material up to a point where the belt passes over a small wooden pulley. The magnetic material here enters an intense field, and leaves the belt (B) and clings on to the lower surface of the belt (B_2), which is travelling in the direction indicated by the arrow. The more permeable material is carried over into a field of lesser intensity, and drops into a bin; an intermediate product drops into a middlings compartment, while the tails simply fall over the end of the small wooden roller.

Another belt separator, designed by Mr. Odling, for the treatment of ore at the Pinnacles mine, near Broken Hill, consisted of an endless belt, on to which ore was fed. Another belt, moving slightly above this, and at right angles to it, had powerful electro-magnets over its upper surface. As the material on belt No. 1 came under belt No. 2, the magnetic portion of the ore, consisting mainly of argentiferous pyrrhotine, sprang up towards the magnet,



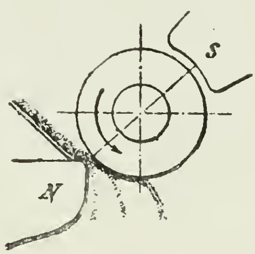
and adhered to the underside of the cross belt until carried over the edge, the garnets and quartz, as gangue, passing over the end.

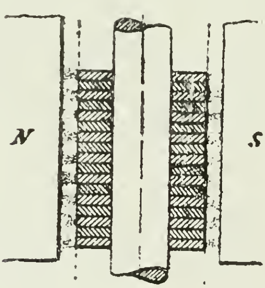
A further example of this type of machine in America, greatly improved, known as the Rowand machine, which has displaced the Wetherill, may be given. A horizontal endless belt is fed evenly; the material passes into an intense magnetic field across the belt, by the action of blunt poles (N), under the belt, and pointed poles arranged across it above, as shown in the figure. The wedge-shaped end of these poles is turned round slightly towards the discharge, so as to cause the intensity of the field to diminish gradually: the magnetic product this falls off easily when it passes the projecting end. The magnetic minerals are carried laterally across, after the



manner with Odling's machine, and dropped into bins at the side. With this system, it is possible to arrange a number of poles one after the other, each being more powerfully excited than the other. By this means, minerals of different permeability may be removed one after the other. This machine is used for separating titanic iron-sand from gold, wolfram from tinstone, monazite from quartz. They are also used at the Kimberley diamond fields for separating magnetic substances from diamonds. All these belt machines have at least one drawback—the material has to be lifted as well as shifted laterally. Any faintly magnetic material might easily be deflected when falling, yet it might be impossible to lift it by the same magnetic force. Mainly for this reason the later types of machines have done away with belts, and depend upon the deflection given to the minerals affected.

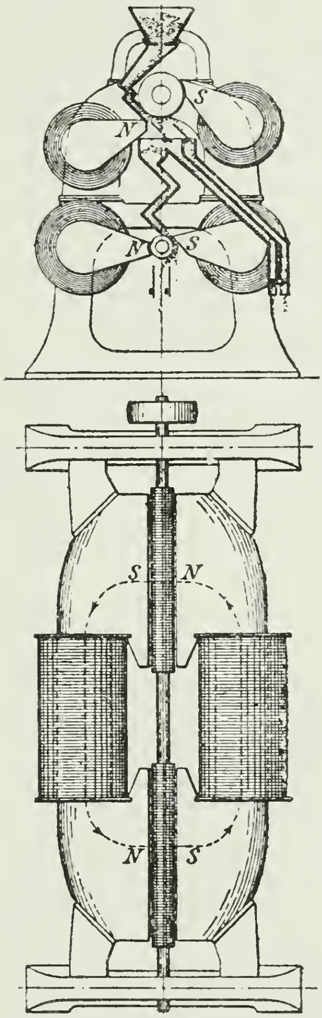
The latest forms of magnetic concentrators of the intense type are constructed without belts. The principle of these may be readily understood from the following figures. A roller rotates between the poles of two electro-magnets (N and S). One of these poles is





made wedge-shaped, so that the lines of force are focussed on to the roller, passing through it to the recessed end of the other pole (S). To still further condense the lines of force, the roller is made by placing alternate rings of iron and zinc on a cylinder. The lines of force are thus collected on the iron discs, especially near the edges. The poles induced in the roller should be normal to the roller, and at right angles to this there should be a neutral plane, having no magnetic attraction or repulsion. Each disc of iron has become a magnet, with its poles in line between the magnets. These discs are rotating so that the poles are continuously changing, and the magnetic field is lengthened out by hysteresis, or extends from the pole to near the neutral zone. Any magnetic material will therefore adhere to the surface of the iron disc until the field is too weak to keep it attached to the disc, when it flies off in the direction in which it was travelling at that instant, and falls into a separate shoot.

One of the first milling companies to keep pace with the times was the Sulphide Corporation, and it is a credit to the management that it adopted what appeared to be the best of the methods then offering for the treatment of the residual products left after concentration. The milling plant itself does not differ greatly from the type used on the field. The ore is raised in cages carrying two

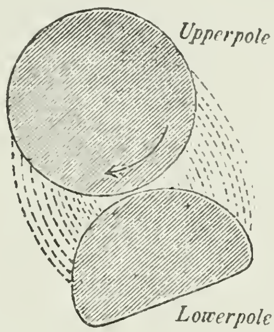


trucks, each holding about a ton. Every truck is weighed at the brace, and samples taken. The trucks are tipped from the brace on to a 2½-inch grizzly. The coarse goes to two Gates' breakers, No. 4, made by Jaques Bros., Richmond (V.), where it is broken down to from one inch to ½-inch gauge. It then passes on to the hopper with the fines from the grizzly, and from this drops on to a Robins' belt conveyor, which delivers it into bins, from which railway trucks are automatically filled. Each truck when filled passes over a weighbridge; thus the weight is checked. From the trucks it goes to the mill bins. The steam for motive power is generated in Babcock and Wilcox boilers; these are supplied with a mechanical feed. This consists of a firegrate composed of a number of links side by side. The whole set of links forms an endless band, which is driven forward slowly. A feed hopper, filled with small coal, discharges this slowly on to the front end of the bars, and by the time the bars which received this feed have reached the other end of the boiler, the coal is burnt out, and the empty bars return underneath, where they are cooled by the draught passing through. This form of mechanical feed acts well, and enables small coal to be burnt. They are being introduced on to nearly all the mines.

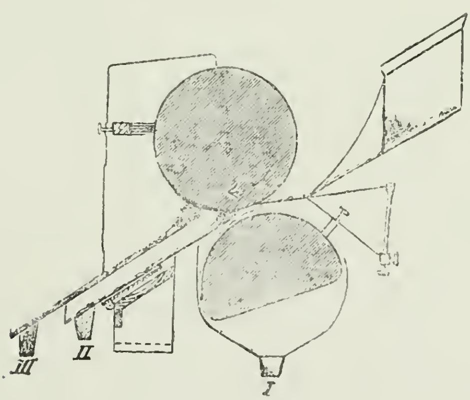
The system of ore dressing comprises breaking the ore from the mill bins down to ½-inch gauge. There are four jaw breakers employed for this in each of the two sections of the plant. The crushed material goes on to a shaking launder, where it is wetted. It then passes into Cornish rolls, and what does not pass through a trommel is returned to the rolls by a raff wheel. The coarse material is treated with four coarse jigs, the product from the third and fourth hutches being reground and treated with four fine jigs. The coarse material treated passes through holes ⅛-inch in diameter. For treating coarse sands and fine material, Wilfley tables are used. The slimes are got rid of by running them into pits, water from these being returned to the mill. Taking this part of the plant, it is no better than the concentrating mills on other mines; no serious attempt is made to cut the work to a fine point. Big losses of silver, lead and zinc occur, but the cream of the ore—the lead concentrates—are looked upon as sufficient to pay working expenses.

One of the most pleasing features on the Hill lies in the treatment of the tailings and middlings which have accumulated at this mine by the Mechernich magnetic process. The management of this mine did not wait until processes were perfected before starting to work. Many of the mines which have adopted this lame policy put one in mind of the man who was supposed to be learning French for some years, and who gave as an excuse for his silence that he was never going to speak one word before he was master of the language. Already a great inroad has been made on the heap, and the management is so satisfied with results that the present plant is to be duplicated. The tailings are loaded on trucks, and hauled up an inclined tram. They are then delivered on to a Robins' belt conveyor, and fed into a drier. This consists of a hollow conical drum rotating on a horizontal axis. The ore passes from the large end through a tube placed inside the drum, which is fed from the open end of this cylinder at its apex. As the drier revolves, the sand travels slowly towards the larger end, whence it passes out and in conveyed to the trommels or shaking sieves. The drier is kept hot by means of the gases from a firegrate traversing the same course as the ore. The drying is only sufficint to dissipate surface moisture, the sand being just about hot enough to be inconvenient to handle. The sieves for screening the dried sand start at about 3m.m. (millimetres), the coarse going to rolls and being again returned. That which passes through this sieve goes on to a 1m.m., and the fines on to a ½-m.m.

There are thus three products—from 3m.m. to 1m.m., from 1m.m. to ½-m.m., and from ½-m.m. to the finest size. Dust is withdrawn from every machine and appliance by means of a strong draught, induced by fans The sized products go to 5 double machines, which put through 1000 tons per week. The principle of the Mechernich machines may be understood from the accompanying illustrations. The first shows the two bar electro-magnets mounted



parallel, the upper one cylindrical, and capable of rotation, the lower one semi-cylindrical, and fixed. The dotted lines indicate the magnetic field, which has its greatest intensity where the poles are close, and gradually decreasing as the distance increases. The crushed and dried ore is fed in between these poles by means of an inclined shoot, slightly pressed by a spring against the upper



cylinder, as shown in the illustration. The quantity of ore fed in depends on the pitch of the shoot and the velocity of rotation. All para-magnetic minerals cling to the under surface of the upper pole, while the dia-magnetic substances fall over the lower into a receptacle (I) placed below. The faintly para-magnetic minerals soon drop from the cylinder, and are then carried down a lower shoot (II). The more highly para-magnetic particles cling to the surface, until the conjoint action of gravity and centrifugal force is greater than the magnetic attraction; they then drop off into an upper shoot, marked (III).

The minerals which are most lightly magnetic at Broken Hill are garnet and rhodonite. Their specific gravity are such, as pointed out before, that wet gravity methods of concentration are not possible. In this case, however, they are easily removed in a clean state, and are devoid of blende and galena. They are sent below for filling. By having a system of double magnets, the strength of current may be adjusted on the first magnets to remove the rhodonite and garnets, and on the second to remove the blende, which is faintly magnetic. Or the blende and the garnets may be removed in the one operation by the means previously indicated. At the Central, the machines turn out three products, namely, clean rhodonite and garnet, clean blende, and the remaining product consists of galena, blende and quartz. This material is then concentrated in the wet way by means of jigs, the escaping product from the jigs passing on to three Wilfley tables. The remainder of the lead is obtained here, also portion of the blende, containing some quartz, the remainder of the quartz being eliminated.

The work done may be gauged by the fact that from middlings running about 9oz. silver, 8 per cent. lead, and 27 per cent. zinc, concentrates carrying 39 per cent. zinc, 12 per cent. lead, and 13.4 oz. silver were obtained. These amounts are for 21,000 tons, the cost of production being about 6s. 9d. per ton of middlings. The trouble with regard to this work is dust. The management have done all that is possible in the way of protecting their workmen and minimising the evil, but yet the objectionable material is ever present. The workmen employed wear respirators, and Mr. Hebbard changes them about, so as to reduce the danger to any man of being leaded. While dry crushing and treatment processes are invariably objectionable, if it can be proved that there is no danger of lead poisoning—assuming ordinary precautions be taken by the workmen—then the operation is certainly no worse than the work done on a large scale at Mt. Morgan (Q.) and Kalgoorlie (W.A.).

Through the courtesy of Mr. George Ullrich, manager of the Australian Metal Company's works at Broken Hill, I was enabled to see the magnetic separators invented by Mr. Ullrich at work. This company did a vast amount of useful work in introducing magnetic separators, and in showing that the waste heaps of Broken Hill could be profitably dealt with. They purchased 66,000 tons of tailings from the Central, and 100,000 from Block 10, paying 5s. per ton to the latter company. Already they have dealt with 130,000 tons, and could have treated the whole of the material in a single year if necessary. The first machines made use of were the Wetherley, but these have been discarded, and Mr. Ullrich's machines installed. The tailings are trucked into a hopper, which delivers an even feed on to a Robins' belt conveyor. They are elevated, and pass into a revolving drier running on friction rolls. This is after the same type as used on the Central mine. The heat is just sufficient to dissipate the moisture, of which there is from 2 to 3 per cent.

From the driers it passes on to a shaking screen, a hood being placed above the screen to draw off any dust. The material which does not pass through the coarse screen, which is only about 2 per cent. of the total, passes to rolls, where it is crushed. The screened material goes to the boot of an elevator, and is carried to the top of the building. It is then distributed between two double trommels, with screens giving three products—2, 3, and 4—No. 1 being reduced.

No. 1.—Retained on a sieve having 9 holes per linear inch.

No. 2.—Passes through No. 1, but retained on a 20 mesh.

No. 3.—Passes through No. 2, but retained on a 40 mesh.

No. 4.—Passes through the 40 mesh.

The last product includes everything below 40, but dust, which is drawn away through steeply sloping pipes, connected with a fan, and sent to a dust chamber. The fines are treated by four Ullrich machines; the coarse by four of the same type. These machines consist essentially of powerful electro-magnets of wedge-shaped section. A pair of these are set up with the edge of the wedge horizontal, and a roller between. The roller is a hollow brass cylinder, capable of rotation on horizontal bearings, and driven at the rate of about 50 revolutions per minute, the rate of speed depending on the material dealt with. On this roller are placed alternately rings of iron, each ring having a rectangular section, and brass or zinc, or some non-magnetic material. These are clamped up together by means of a brass nut, and so form an external cylinder about 3 feet long, composed of alternate washers of zinc and iron, each being about one-fourth of an inch wide. This arrangement, as already pointed out, gives a very intense field near the edges of the rings.

Each machine contains two sets of rolls above and below; that is, each frame contains four rollers, two above on the same axis, but divided by a central bearing, and two below similarly separated. Steel bushes are used for the bearings.

The ore is fed in a steady stream through a hopper having a distributing arrangement caused by having an inclined plate, with zig-zag channels cast in it. The mixed material falls on the roll. The current is so adjusted on the upper roll that only garnets and rhodonites are removed and fall over a deflecting board. The balance of the ore drops on to the next roll, and a stronger current is passed through; the zinc blende is then romoved in a practically pure state.

The final products which come out are garnet and rhodonite, and garnet on the upper and blende on the lower; while the non-magnetic material escaping mainly consists of galena and quartz carrying a small amount of blende. There are eight machines on the top floor, four for dealing with the coarse, and four for the fine, grains. The blende still remaining in the galena is removed by passing the escaping tail product from the upper machines over two half machines on a lower floor. The tails from this machine consist almost entirely of galena and quartz. This goes to an ordinary wet dressing plant, where the coarse grains are separated with jigs, the fine on Wilfley tables.

At present, 200 tons are treated per day, and about 100 tons of zinc concentrates are recovered, while from 50 to 60 tons of lead concentrates are recovered per fortnight.

From 75 to 80 horse-power is required to work the plant. Each machine requires only 1½ M. and E.H.P. The dynamo at present used is much beyond actual requirements; it is also used for lighting the works. The current indicators showed 120 amperes, at a pressure of 110 volts. Steam is generated in two multitubular boilers, at 120lb. per square inch, and supplies a compound engine; the power is about double that necessary.

The zinc concentrates from the fines go 46 per cent.

The zinc concentrates from the coarse go 41 per cent.

The zinc concentrates from the mixed go 43 per cent.

The recovery of zinc from the first machines is from 82 to 86 per cent.

The lead concentrates run 45 to 47 per cent. for the fine.

The lead concentrates run 42.5 to 44.8 per cent. for the medium.

The lead concentrates run 40 to 42 per cent. for the very coarse.

The recovery of lead is 80 per cent. The lead concentrates are rich in silver; the zinc concentrates run from 15 to 18oz. per ton.

As showing the relative quantities of minerals in some of the mines on the Hill, the following is the result of the treatment of some hundreds of tons of material:—

	Block 10.	Junction North.
Garnet and rhodonite	30	66
Quartz	20	—
Lead	.5	20
Zinc	20	10

Mr. Ullrich is to be congratulated on showing the Broken Hill people how to deal economically with the vast heaps of tailings which have accumulated. Only 10 men per shift are required, and the work they have to do is of the simplest character, for the various machines are practically automatic. The dust problem is a serious one, but every effort has been made to minimise the evil, by providing powerful fans with exhaust pipes to withdraw the dust from every appliance. This is finally collected in a chamber, the amount being about 2½ per cent. of the ore treated, wetted, and sold as slimes. Mr. Ullrich considers that his machines could deal with raw ores, and give a better return than that given in the wet dressing plant. It is also evident that much better extractions could be obtained by separating out a good lead product from the wet concentrating plants, and a poor tail, consisting mainly of quartz, leaving an intermediate product rich in lead, zinc, and containing rhodonite and garnet, to be dealt with by magnetic concentrators. It is gratifying to note that the Junction North Company is about to adopt this process for treating raw ore. The work will be intensely interesting, but only what should have been done years ago by the wealthier companies.

Mr. Ullrich estimates the cost of a plant capable of treating 1000 tons per week to be only £6000, so that the experiment would have been less expensive, and a step in advance of what those companies are now attempting to do.

Broken Hill Proprietary Company, Limited.

Smelting and Refining Works at Port Pirie.

The Broken Hill Proprietary Company selected Port Pirie as the site for its smelting operations. The choice was an excellent one in many respects. Vessels of 5000 tons, drawing up to 23 feet, may be brought almost up to the works. Limestone is delivered by boat from the adjacent shores, and iron flux from the Ironstone Knob. Coke comes from the company's works at Bellambi, where 100 ovens are employed producing it, as much as 18,000 tons being delivered in six months. Timber for the mine, oregon from America, stores of all kinds from oversea, are despatched to the mine from this centre, while market lead, silver and other products are shipped direct from the works.

The smelting and refining works, probably the largest of their kind in the world, are now wholly employed in dealing with the products from the Broken Hill Proprietary mine. These consist of argentiferous lead concentrates and slimes. The former are conveyed to the roasters; the latter, which arrive as a sintered product, go direct to the smelters. The roasted concentrates go to the Huntington-Heberlein pots, where they are partly desulphurised and semi-fused. This material then goes to the smelting floor, where it is mixed with the slime product and the necessary quantity of ironstone and limestone.

The concentrates, as a rule, run from 50 to 55 per cent. lead, 10 per cent. zinc, and 26oz. silver per ton. These are mixed with a certain amount of ironstone finely ground, the ordinary hematite being used, first being crushed with 2 Gates' breakers, and then passed through rolls; limestone in a pulverulent condition, and also sand, are added, the proportion of the ironstone, limestone, and sand being such as to give an easily melted slag, low in silica. These are all fed into a Ropp furnace automatically, by passing over a fluted roller feed, being driven proportionately to the speed of the rabbles. Challenge feeders were formerly in use, but have been discarded in favor of the simple fluted roll placed at the discharge end of a hopper.

In previous articles, a description has been given of the Ropp furnaces, and it is only necessary to state that they are long hearth reverberatories, closed by double sets of hanging flap doors at either end. A slot runs along the centre of the floor from end to end, and widens out to a tunnel below, after the same manner in the cable tram system. A cable runs in this slot; on to this is attached a carriage, having a vertical arm, which passes through the slot into the furnace; on to this a horizontal arm having a number of blades arranged, which enter the ore on the floor and turn it over in a series of parallel ridges. The next row of blades cuts these ridges in the centre, and throws the ore over in the opposite way. The rabbles, after passing through the furnace, pass into the air, and return on a track and re-enter the furnace again. The ore is dragged slightly forward at the same time as it is turned over by each rabble, and thus passes from the cooler to the hottest part of the furnace before being discharged. Heating is done by having four fireplaces on one side of the furnace; the flame is distributed by making the arch high at these places. Five of these Ropp roasters deal with the whole of the concentrates produced at the mine. About 100 tons of the mixture of concentrates, sand, limestone and ironstone are put through each roaster per 24 hours, and about one-half of the sulphur is eliminated. The calcium carbonate is decomposed into lime, but practically all of this is converted into sulphate of calcium by the joint action of the sulphur dioxide evolved on roasting, and the oxygen of the air. Probably, also, calcium sulphate forms from the inter-reaction of the oxide with lead sulphate. The material discharged from these roasters is in a mealy condition; this is essential for successful work in the subsequent operation. While still hot, it is fed into the Huntington-Heberlein pots. There are very much larger than these used in other places. There the 17 pots in one horizontal row, each holding an 8-ton charge. If filled with pure concentrates, they would hold 10 tons. Each pot is the segment of a sphere, the diameter being about 8 feet and the depth 6 feet; an annular punched plate lies on the bottom. On the open circle left by this annular plate lies a short frustrum of a cone, also punched with holes. The top of the conical piece is closed by a circular plate, also punched with a few holes. The top plate is about 2 feet above the bottom of the pot. Air is admitted below the chamber so formed, which is practically a wind-box, allowing of a fairly even distribution of air current through the material placed above it. The pots themselves are made of cast iron, and hung on trunnions. A conical cover, or hood, which may be raised or lowered, has a flanged lower edge resting on the corresponding upper edge of the converter, thereby giving a fairly close joint. The hood is provided with a number of small doors, through which the charge may be inspected or bars introduced for turning over portions of the charge while the operation is proceeding. A pipe attached to the top of the conical cover carries away the gaseous products, due to the reactions of the intermixed materials, to a flue common to all, leading to the main roaster stack. This pipe is telescopic, so as to admit of raising or lowering the hood.

After the charge is fed in, a steady blast of cold air is turned on, the mass starts to warm up, as ignited coke before a bellows, and in course of time the whole mass becomes red hot, and the products resemble a glowing coke fire. When the reaction is complete, the whole mass becomes fritted or sintered into honey-combed lumps. These are tipped out of the converter by inverting it. The blast is increased up to 24oz. pressure, and the pressure in the pots will depend on the interstices between the ore particles. The physical state of the ore mixture is all important; it must be granular or in an evenly porous condition, otherwise a patchy product will be turned out of the converter. The time taken is no longer than with smaller pots used in other places; in general, it amounts to about four hours. The reactions which probably take place in this process have already been discussed. It is found that a certain proportion of sulphates, as well as sulphides, are necessary for the reactions. These sulphates are formed in the preliminary roast.

The zinc appears to be desulphurised through the inter-reaction of the sulphide and sulphate—

$$ZnS + 3ZnSO_4 = 4ZnO + 4SO_2.$$

The amount of sulphur still retained amounts to about 3 per cent. Practically no lead is volatilised in the process, and very little zinc. As might be expected, there always remains a small amount of unburnt charge on the top of the converter; this goes into the next. A small amount of dust is blown out; this is caught in pyramidal boxes in the common flue from all the pots. The red hot fused and semi-fused material, on being tipped out of the converter, has a jet of cold water turned on it; a portion of the sulphur still remaining is eliminated as sulphuretted hydrogen. It is then broken up with hammers, loaded into buckets, hoisted and carried by means of a Flying Fox to the smelting floor. Very few breakages of pots occur, only two breaking in eighteen months. Some crack, but are patched up with a plate of iron and a few rivets, and are as good as ever. A great saving has been effected by the adoption of this system. Better work is now done with eight and a-half furnaces than was previously done with thirteen.

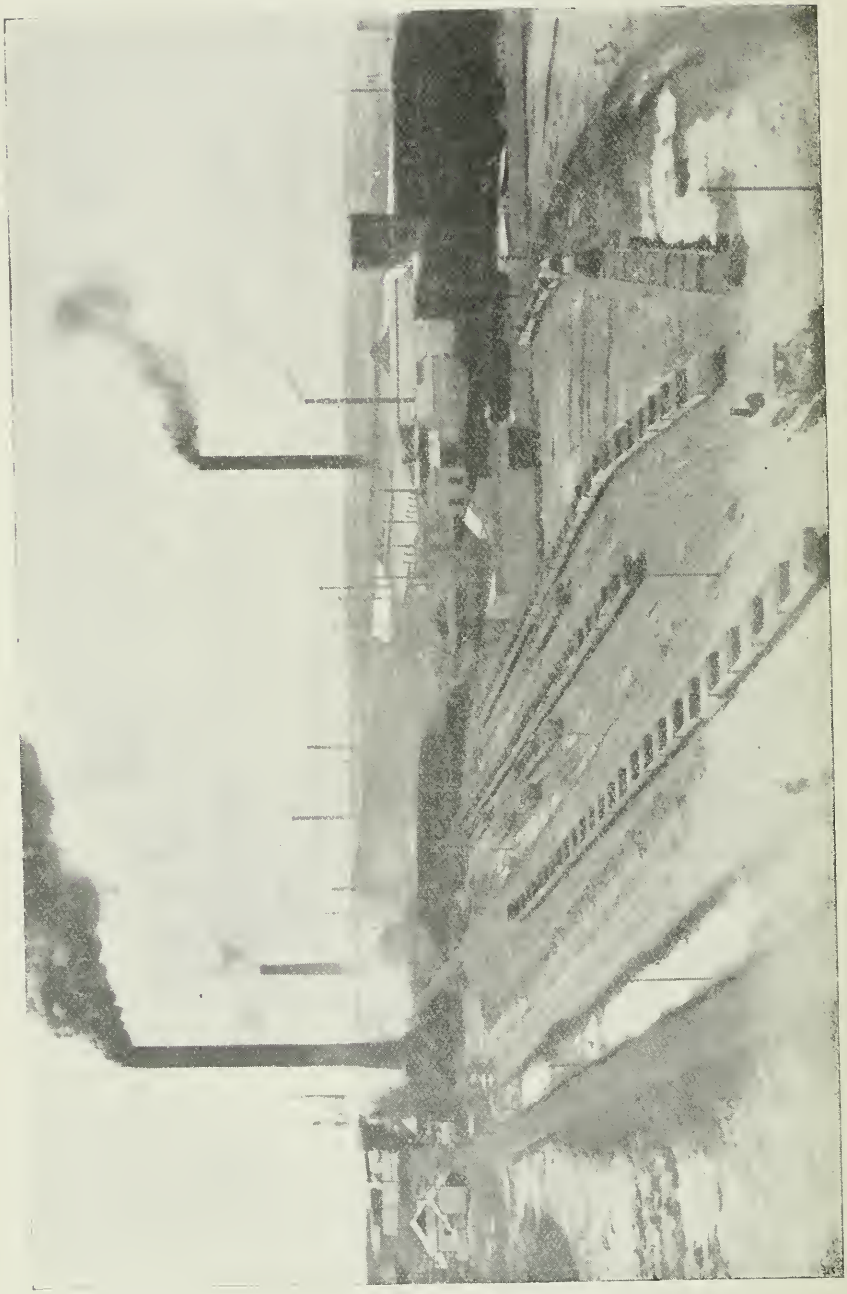
The sintered slimes come in trucks from Broken Hill. These are run up alongside the smelter floors, and unloaded direct into the charge barrows, any excess going to storage bins. There are no bins for bed ores, nearly everything being delivered with as little handling as possible on to the smelter floors, the excess bins only being used as a stand-by. Limestone, ironstone and coke are delivered from the vessels, slag from the dump by Flying Foxes into bins on the feed floors and charge barrows loaded direct from these bins. Wherever ore has to be barrowed, the floors are laid with iron plate. The barrows are so made as to be balanced over the wheels, as well as provided with roller bearings. Each weighs 700 lb., and a man can wheel as much as a ton over the floors. The discharge of material from them is effected by means of a lever connected with the handle opening a door forming the front part of the barrow. The feed floor of the furnaces is quite clear of any hoods, the ore being delivered into the partly-closed top, all gases and fumes being drawn off at the down-take flue. The furnaces are arranged in one continuous row, with their longer axis in line.

The products fed into the furnaces are all weighed, as is also the output, with the exception of slag, which is closely estimated. A daily tally is kept, ending at 8 a.m.

The material fed in consists of sintered slimes. These assay about 16.5 per cent. lead, 12 per cent. zinc, 7 per cent. sulphur. and 17oz. silver per ton. The balance is made up of silicates of manganese and iron and alumina derived from the quartz, rhodonite and garnet of the ore.

The Huntington-Heberlein (H.H.) product consists of silicates of lime and iron and oxides of lead and zinc. The sintered slimes contain the most of the silica; that in the H.H. material was only added to make a low silica slag with iron oxide, lime and zinc.

The limestone used for the H.H. plant is a soft yellow tertiary variety, full of fossil casts. It is very pure, containing only from 2 to 3 per cent of insoluble matter. A dense limestone, equally pure, is used for the smelters. Old slags from the centre of the slag dump are also added. The ironstone is a dense pure hematite. The sintered slimes and H.H. product are fed in in blocks up to a



General View.

foot in diameter; the limestone and ironstone in blocks up to 9 inches through. The whole charge weighs 5600lb. From 1700 to 2000 tons of ore, fuel and fluxes are fed into the furnaces per day, or the average capacity of each furnace is 240 tons of ore, fuel and fluxes. The output of the furnaces of slag is about 1000 tons per day. There are ten (10) large furnaces and two (2) smaller ones, as well as a Piltz. The eight large ones and one small were in blast at the date of my visit, the Peltz not being used at all. The small furnaces are 60 inches by 112 inches at the tuyere level; the large measure 62 by 212 inches. The height from this level to the bottom of the down-take is 17 feet 6 inches, the down-take being 2 feet 6 inches high. The large smelters are provided with 20 three and a-half inch tuyeres, 10 on either side. The noses of these project some distance into the furnace. The smaller smelters have an end tuyere also.

The lead well is built up solidly. A single tier of cast iron jackets, 4 feet in height, rests on this. There are eleven on each side, and three at the ends, or 28 in all. The jackets are cooled by passing sea water through them. The total quantity of sea water used in all departments of the works amounts to 5 million gallons, or 25,000 short tons, per 24 hours. The blast is supplied at a pressure of from 28 to 36oz., from 11 to 12 million cubic feet being used per furnace per 24 hours, or about 8000 cubic feet per minute. The slag is drawn off at each end of the furnace; it flows direct into slag pots, of which there are two on each carriage, each holding 18cwt These are drawn by horses on a track laid down round the edge of the dump. The slag is tipped out, and the pots are returned to the smelters, the whole distance travelled being about ¾ mile. It would seem as if this work could be done more cheaply by granulating the slag, and delivering it to the dump through centrifugal pumps, as at Mt. Lyell, but perhaps the comparatively low rates paid at Port Pirie for horses and drays is a sufficient answer to this.

The lead is tapped out from two openings at one side of the furnace, the first lying between the first and second tuyeres, and the second between the ninth and tenth. It is run direct into a series of moulds, each holding 80lb., placed on a carriage set on rails. As one fills up, the carriage is moved forward and another fills.

Some of these furnaces have run for a long time without stopping; the record is of one which was in continuous blast for four years and ten months, or 1760 days.

The slags as a general rule run as follows:—

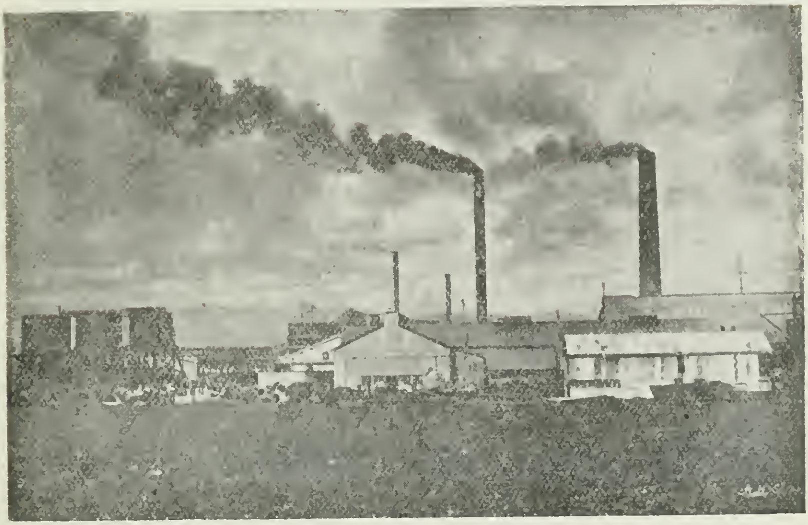
Silica (SiO_2), 25—26
Ferrous oxide (FeO), 31
Manganese oxide (MnO), 5—5.5
Lime (CaO), 15.5—17
Zinc oxide (ZnO), 13.0
Alumina (Al_2O_3), 6.5
Sulphur (S), 3—4 (5 per cent. abnormal)
Lead as determined by wet assay, 1.2—1.5
Silver, .7oz. per ton.

In spite of the amount of sulphur present, no matte is formed: the sulphur is dissolved in the slag. There is practically no copper in the ore. The blast for the smelters is supplied by seven (7) Green blowers, No. 8, each of which delivers 112 cubic feet of air per revolution. Blowers run 100 revolutions per minute. Eight hundred horse-power is expended in driving the blowers. Boilers have Green's economiser for feed water and steam superheaters attached. Seven Lancashire boilers, working at a pressure of 120 lb. per square inch, supply the steam for this section of the plant. Centrifugal pumps are used for the circulation of water in the jacket, these being favored before the steam-driven pumps. Each pump can raise 8 million gallons per 24 hours. This part of the plant is in duplicate. Dynamos for supplying the light for the plant are also in this section of the buildings. High tension currents of 2000 volts are transformed into those suitable for the arc light.

The bullion produced at the smelters is transported to the refinery, and goes into four softening furnaces, each holding 45 tons. These are of the usual type, jacketed near the top. The lead is heated to just above its melting point, when any dross and also traces of copper are removed by skimming. The liquid lead still containing antimony is run into similar furnaces, of which there are five (5), called the antimony softeners. A higher temperature is employed, and the antimony oxidises and combines with the lead oxide simultaneously produced, forming antimoniate of lead. In order to extract the silver still contained in this, it is mixed with just sufficient coke to reduce part of the lead; the silver is thus transferred to the lead, and the balance, consisting of oxide of lead with antimoniate of lead dissolved in it, is run down periodically in a blast furnace, an alloy of from 20 to 25 per cent. antimony being produced. The softened lead, containing the gold and silver, is then run into the zincing kettles, of which there are ten, each holding 42 tons when filled. The bullion contains under 1dwt. of gold per ton. This is removed by fractional separation with zinc. It is noteworthy that a lesser amount of zinc is required to remove this gold along with some silver than is ordinarily sufficient to saturate the lead. A second zincing removes the bulk of the silver, which runs from 80 to 100oz. per ton, and a third reduces the contents to from 2 to 3dwt. per ton. The skimmings from the last zincing, containing, as they do, only a very small portion of the silver, are used as the second zincing for the next pot.

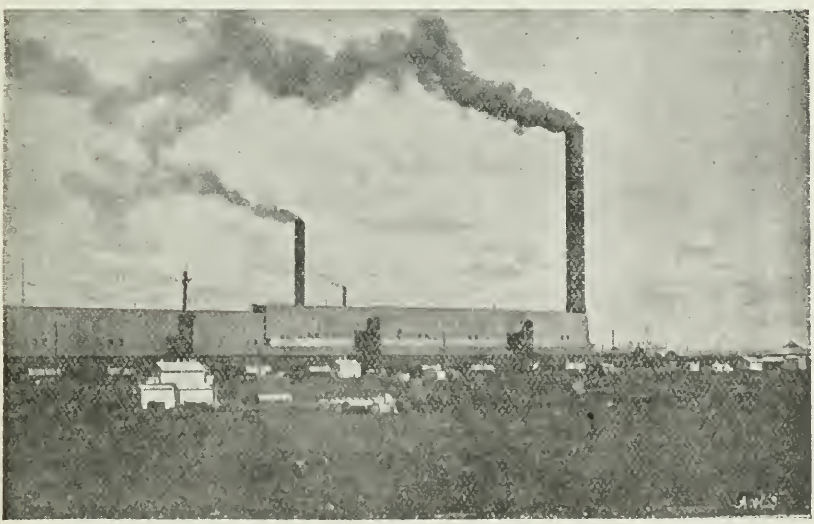
The mixing of the lead and zinc is done by a Howard stirring machine, instead of by steam or hand paddles. The stirrer consists essentially of a horizontal paddle, having four inclined blades. This is attached to a vertical shaft, which is driven by a small engine. A conical cover, with an inner flange, is so arranged that when lowered on to the kettle the edge rests on the rim of the kettle, and the flange dips into the lead, thereby forming a seal, and preventing contact with the air, except for the small portion enclosed. The whole apparatus is hung up on a traveller, and may be run over each pot in succession. The motive power for stirrer is supplied by a small engine, which drives a line of shafting running parallel with the zincing pots. The power is transmitted from the shafting to the stirrer by means of a rope. The paddle is driven at the rate of about 100 revolutions per minute, and the mixing is complete in about 15 minutes.

The zinc crusts are removed and liquated in one operation by means of the Howard press. This consists of a cylinder about 2 feet in diameter, with a perforated hinged bottom. A piston is driven by air pressure into the cylinder. The whole is suspended



Cooling Water Tanks, Saltwater. — Fitting and Blacksmith Shops. — Repair Shops.

Broken Hill Prop. Co.'s Smelting Works, Port Pirie.



[illegible] — Smelting Sheds. — Stack 205ft. high, 41ft inside. diameter. — Assay Office. — General Office.

on a traveller, and may be moved over any kettle. It may be raised or lowered by means of a pneumatic lift or cylinder. Compressed air for working the press and lift is supplied from a main by means of a flexible hose. The zinc is removed by lowering the press into the molten metal. As soon as it is as hot as the metal, it is slightly raised, so that the upper open end of the cylinder is just below the surface. The piston is raised out of the cylinder to admit of this. The zinc crust is then worked into the cylinder. This is then raised, and any lead allowed to drain out. The air is then turned on, and the piston descends, driving the crust before it, and expressing the liquid lead, which runs back into the kettle. The press is now pulled to one side, and the solid cylinder of pressed zinc, silver, lead alloy, allowed to drop out, and is broken into small pieces. The molten lead, still containing about 0.75 per cent. of zinc dissolved, is siphoned into the refining furnaces, which are ordinary reverberatories. A current of hot air oxidises the zinc, which is removed with some litharge. A small sample of the lead is removed from time to time, and tested by pouring into a mould. If large, bright, platy crystals appear on the horizontal surface of the bar, the lead is refined, but if patchy crystals, with a white streak along the centre, occurs, the lead is impure, and needs further oxidation. The refined metal is cast into pigs weighing either one or two hundredweight. The large bars go to China, where the tariff is said to be per pig of lead. It is also said that, since it takes two coolies to carry each pig, the larger size is a suitable load. It is worthy of note that the lead, from the time it reaches the refinery, is transferred in a molten state from one appliance to another, until it is solidified as refined lead; that the expensive zincing kettles are not used for the somewhat destructive work of refining the lead; that the softening furnaces and antimony softeners, requiring different temperatures, are separate furnaces; that the zinc, silver, lead alloy is turned out of the lead in solid blocks (broken while hot into small pieces for charging into the retorts), carrying the gold and silver bullion, and that the litharge carrying the zinc oxide from the lead refining furnace is a solid fused product, and not the powdery, friable material obtained by oxidation by means of steam. The zinc crusts carry from 2500 to 3000oz. silver per ton, and from 12 to 14 per cent. of zinc, the remainder being lead. This alloy is placed in pots, each holding 12cwt., in a regenerative furnace, and the zinc distilled out. This alloy is placed in pots in a furnace regeneratively heated, and the zinc distilled out. The furnace has a small gas producer at one end. The heating chamber is divided longitudinally by a wall, thus making two furnaces of it. Two retort pots are placed in each; these project through the walls, and on to the mouths of these a receiver is fitted to hold the condensed zinc. The high temperature requisite is attained by allowing the gas from the producer to mix with the air in the combustion chamber. The products of combustion are then led away through checkered brickwork, which becomes heated up to the temperature of the escaping gases. When this occurs, the valves are turned, and the products of combustion are cut off from the first, at the same time entering another chamber with bricks loosely packed, which becomes heated up. At the same time air is admitted through the first chamber, and by doing so it absorbs the heat from the bricks, becoming red hot itself. This hot air then enters and burns with the producer gas. The latter is not regeneratively heated, since the producer is close to the combustion chamber, and the gas is already hot. As soon as the entering air cools down the brickwork, the valves are turned, and the temperature maintained. It is found that this is best done every half hour. The adoption of this type of furnace has effected a saving of 75 per cent. of the fuel used in those types where solid fuel is burnt. About 10 hours are required to work off a charge. Of the zinc used, about 66 per cent. is recovered and used over again. Most of the metal lost is dissolved in the lead, and is converted into oxide. On refining the latter, as already pointed out, a certain proportion is lost as powder, owing to the chilling of the zinc vapor into a finely divided solid. The pots used stand about 70 charges, some as many as 100.

The silver-lead alloy is ladled into moulds, and transferred to the concentrating furnaces, of which there are five. Each of these has an oval test, about 5 feet by 4 feet by 6 inches deep. This is cooled by a water coil at the rim, and at the litharge channel. The cupel test bottom is made from a mixture of limestone, fireclay and cement. The test on a carriage forms the hearth of a short reverberatory furnace. This is heated up, and as soon as hot enough an air jet plays over the surface of the molten lead, which becomes oxidised and melts, draining away through a channel at one side. Additions of silver-lead are made to this, until a convenient quantity has accumulated. The whole is worked down until it contains from 50 to 60 per cent. silver. This rich alloy is then run into finishing cupels, of which there are three. The temperature for finishing is higher than that used in the others. By this means, the last of the lead is oxidised and got rid of. The silver so obtained is nearly pure, but every trace of oxidisable metal is got rid of by re-melting it at a high temperature on a fresh hearth. It is then tapped out and run into octagonal moulds, the blocks being of a suitable size for introduction into crucibles. Practically pure silver is produced by this treatment, but since the silver is sold as 996 to 998 fine, a small amount of copper is added to give the additional weight, and the standard brought down to 996 or 998. The metal is then melted in crucibles. Some pieces of charcoal are added to prevent absorption of oxygen and subsequent spitting on pouring, due to the evolution of that gas. The metal is poured into moulds holding about 1050oz. The bars of pure silver are trimmed up, and weighed exactly, any fractional part of half an ounce being cut off. On each bar is stamped the company's mark, fineness, and weight.

Parting Dore Bullion.—The amount of gold in the Broken Hill ores is small, yet the aggregate amounts to about £10,000 per annum. The first zinc crust from the kettles containing the gold is treated in the same way as those containing the silver. The amount of gold to silver present in the dore bullion is about 1 is to 25, whereas if both gold and silver were separated together the amount would be in the proportion of 1 of gold to 1600 of silver. Bars of dore bullion are allowed to accumulate until there is sufficient material for a run, this happening twice a year.

There are three cast-iron kettles in line, each heated below. The two outside ones are for the solution of silver, the central one for dealing with the gold residue left in the other two. Bars of silver are introduced into pots 1 and 3, and strong sulphuric acid, or mother liquor, from one of the subsequent processes introduced. A hood is lowered to carry away any noxious gases liberated. As is well known, silver dissolves in strong sulphuric acid—

$$2Ag + 2H_2SO_4 = Ag_2SO_4 + SO_2 + 2H_2O.$$

The silver sulphate thus formed dissolves in excess of the acid, one part of the hot acid dissolving four parts by weight of the silver sulphate; whereas, only one part of the salt will dissolve in 200 parts of cold water. Further, if silver sulphate is dissolved in less than three parts of hot, strong sulphuric acid, the bisulphate will crystallise out on cooling. It will thus be seen that by keeping the acid in excess, hot and concentrated, that a large quantity of silver may be taken into solution. After the silver has dissolved, the gold is allowed to settle, and the solution of sulphate of silver in sulphuric acid decanted by a siphon into cast iron vats. Here it is diluted by means of steam, and allowed to cool, when hard yellow crystals of silver sulphate separate out. The silver sulphate is thrown on to filters, and washed with cold water to wash out the free acid. These washings are run through boxes holding copper plates; the silver is precipitated, and the copper goes into solution as sulphate. This, in its turn, is precipitated on scrap iron. The precipitated copper is sold as such.

The mother liquor from which the silver sulphate was recovered is evaporated down to strong acid, and used over again, so that the only acid lost is that used in the solution of the silver and the small amount adhering to the sulphate crystals.

The crystals of sulphate of silver are dried in a furnace, and are then introduced into a cupelling furnace, with about 4 or 5 per cent. of crushed coke. Sulphate of silver is split up into silver, sulphur dioxide and oxygen, the last of which combines with the carbon added.

$$Ag_2SO_4 + C = 2Ag + SO_2 + CO_2.$$

The metallic silver melts down, and is poured into bars.

The gold left after siphoning the bulk of the silver sulphate from the pots No. 1 and 3 is ladled into No 2. It is here boiled with more sulphuric acid, and then ladled out into a lead-lined vessel, and washed with boiling water, to remove the silver salts. It is further boiled with hydrochloric acid, to remove the iron and lead salts; again filtered and washed, and then smelted into bars, which are almost chemically pure.

The Sampling Mill.—This is a separate building. The upper floor, which is on the same level as the smelter floors, contains two small breakers, one for the samples which require to be crushed finely, the other for those that do not. After passing through the breakers, the crushed material goes through rolls, and after this, in the second case, the pulverised stuff is coned and quartered on a sampling floor, and the final sample selected. In the other case, where the sample has to be reduced to a fine state of division, it is fed into a pipe sampler, so arranged that when the ore drops down the cylinder that a certain portion is deflected and thrown out by inclined plates let into the pipe. These plates are arranged at different heights, and the openings for allowing the ore to escape are at right angles to each other. Above each deflecting plate the interior of the pipe is narrowed, so that the descending stream passes through a narrow funnel-shaped opening. The ore passing out goes to the ore bin, and that which passes down the pipe, being about 1-64th of the total sample, is taken as the assay sample. This portion is then further triturated by grinding in a mill of the coffee mill type, which will crush the material fine enough to go through a 100-mesh sieve. The final sample is selected from this. There is also a third sampling provided for slags and material devoid of silver. This is provided with a breaker, rolls, and an iron covered floor for coning and quartering. As much as from four to five tons are taken as a starting sample from materials of variable composition, such as H.H. products and sintered slimes.

The assay offices are well equipped, and the work is continuous over three shifts. Silver bullion is assayed by the sulpho-cyanide method. It is found that a small percentage of copper does not interfere with the accuracy of the method. The special feature about this work is that a carboy, holding several litres of the KCyS solution, are permanently connected with a pipette by means of a T-piece fused into the latter below its gradiated stem. The pipette can be filled by opening a pinch clip on the rubber connecting these. The pipette holds 99c.c. between a mark on the upper part of its stem, and on the lower the last c.c. is divided on the narrow stem into 100 divisions. The amount of silver weighed out is such that the finishing point will be somewhere on this last division. In order to run the solution out in minute quantities, a piece of glass bulb is inserted in the rubber connecting the end of the pipette with the delivery pipe. By pinching the rubber laterally, a small channel is made, which allows the fraction of a drop to run out. Absolutely clear glass flasks, having a capacity of about a litre, are used. The final volume of liquor is about 500c.c., and iron alum is used as an indicator. The strength of the KCyS solution is such that 1c.c. = 0.010grm. silver. The work is always checked with pure silver, or silver of known purity. The results are as accurate as those by Gay, Lussac's method.

The Sulphide Corporation, Cockle Creek, Newcastle.

After the mad inflation of Broken Hill stocks by those who rushed the rich and easily treated carbonates and oxides through the furnaces, came the day of reckoning, when silver-lead-zinc sulphides had to be treated. Smelting under conditions then existing was not possible. The intimate admixture of the zinc-lead ores indicated trouble with concentration, and altogether the outlook was almost as gloomy as if the mines were exhausted. At this time, a process was put forward by Mr. E. A. Ashcroft, which promised to revolutionise the treatment of such complex compounds as existed, not only at Broken Hill, but in many other places in vast quantities. Briefly, it was proposed to roast the ore. The zinc was then to be dissolved out, and deposited from its solutions electrolytically, while the lead and silver remaining would be smelted in the ordinary way. After roasting the ore, the zinc was oxidised to oxide and sulphate. The sulphate could be washed out with water, while the oxide was soluble in a solution of ferric chloride, ferric hydroxide being simultaneously precipitated.

$$2FeCl_3 + 3ZnO = 3ZnCl_2 + 2Fe(OH)_3$$

In practice, it was found that not only did the zinc dissolve when treated in this manner, but also manganese, silver and lead. Further, the ferric chloride was not wholly precipitated, as indicated by the above reaction, but partly reduced to ferrous chloride, probably by the action of unaltered sulphides.

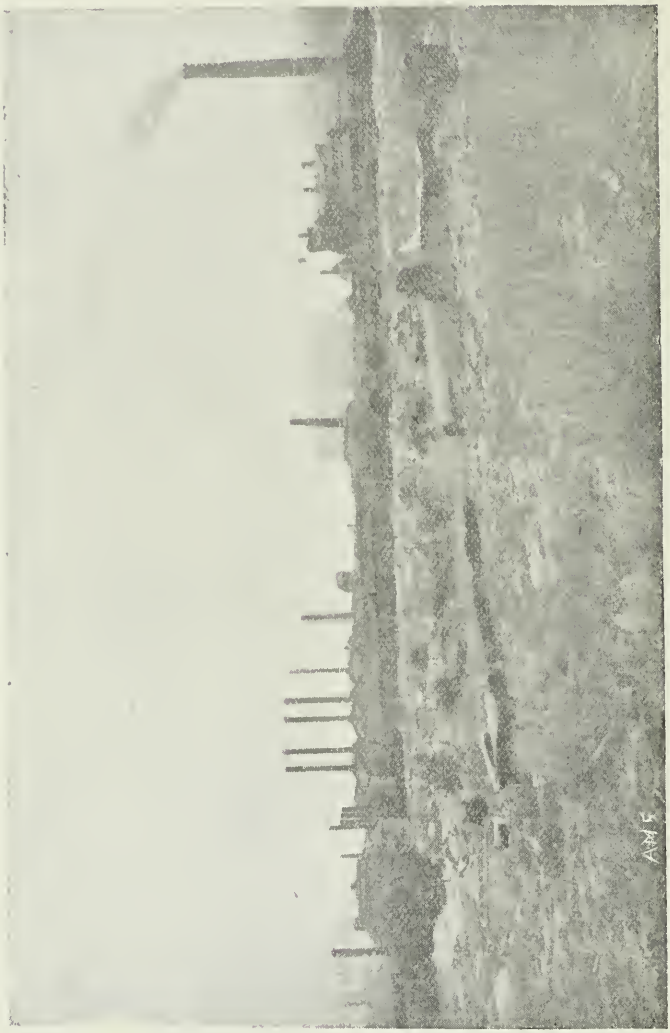
$$2FeCl_3 + ZnS = ZnCl_2 + 2FeCl_2 + S$$

Sodium sulphate had to be added to precipitate the lead, while excess of ferric chloride was precipitated with zinc oxide; no method was hit upon for removing the manganese. The precipitates from the solutions were obtained by filter-pressing, and the cakes sent to the smelters. The clarified solutions were then treated with zinc dust to throw down the silver, lead, antimony or arsenic still in solution. After these had separated, the solutions carrying zinc, manganese and ferrous salts were circulated through electrolysing vats, and the zinc deposited on thin sheets of rolled zinc. Carbon was first used for anodes, but these disintegrated so rapidly that sheet iron was used, except in certain vats, in which carbon was still retained. Zinc was deposited, and an equivalent weight of iron passed into solution as ferrous chloride. This was converted into ferric chloride in the carbon anode vats. The ferric chloride thus produced was used on a fresh lot of ore.

After an extensive trial of the process in England, and after favorable report from the well-known metallurgist, Dr. Carl Schnabel, who had previously reported exhaustively on the ores, but failed to find a practicable method for treating them, very extensive works were erected at Cockle Creek, near Newcastle, New South Wales. The site was no doubt chosen on account of the proximity to an excellent harbor, and in the very midst of the northern coal mines. When work was started on a large scale, it was soon found that ferric chloride solutions were most difficult to work with. They were therefore abandoned, and dilute sulphuric acid used as a solvent, lead plates being substituted for the carbon ones. The process, however, could not be successfully carried on, the roasting was imperfect, trouble was caused in filtering owing to the separation of silicic acid, manganese accumulated in the solutions to such an extent that they had to be run to waste periodically, while great trouble was occasioned by maintaining the electrolyte in such a condition as to give a solid coherent deposit.

The Sulphide Corporation now owns what is left of these once extensive works, but has gradually eliminated most of the old plant, and substituted a fine smelting plant, and also the only zinc-producing works in the Southern Hemisphere. Constant and large supplies of ore are drawn from its own mine, the Central, at Broken Hill, as well as from other fields in Australasia. Ores are purchased according to tariff. It has been a difficult and somewhat unsatisfactory task for the management to remodel a plant for ore to be worked on entirely different lines. At the present time, the work of demolition and reconstruction is going on, so that in a very short time great improvements will be effected.

There are two distinct operations carried on at the works, the first being the smelting of lead concentrates and other auriferous and argentiferous ores, the second being the production of zinc. The bulk of the ores for both processes come from the Central mine. This mine turns out about 16,000 tons per month, averaging 12.5oz. silver, 18 per cent. lead, and 19 per cent. zinc. This material is crushed and dressed, the concentrates, amounting to about 20 per cent. of the ore treated, running 60 per cent. lead, 10 per cent. zinc, and 30oz. silver per ton. The recovery of silver thus being only 48 per cent., and of the lead 66 per cent., while the zinc present is not wanted at all. The slimes, amounting to about 8.5 per cent., contain silver, lead and zinc in almost the same proportions as the original ore. The great bulk of the material discarded is known as middlings and tailings; these amount to 71 per cent. of the total ore, and carry values in silver equal to 7oz. per ton, lead 5.7 per cent., and zinc 21 per cent. This material thus carries nearly 40 per cent. of the total silver, 22 per cent. of the lead, and 80 per cent. of the zinc. There is little wonder that the metallurgists at Broken Hill and elsewhere looked upon this loss as appalling, and searched after every method which promised to give a better recovery. The only two processes which appear to have any chance of being successful are magnetic separation processes, and the process discovered by Mr. C. V. Potter, and named after him. The Mechernich system of magnetic concentration has been adopted at the Central Mine. By this system, the ore is delivered in an even stream between two magnetic poles, the upper one of which rotates. As the ore reaches the magnetic field, the particles capable of being attracted attach themselves to the rotating pole, and are carried round into a diminishing magnetic field. The rotating pole here throws them off the surface on to an inclined chute. The non-magnetic particles fall into a different receptacle. Since blende or zinc sulphide invariably carries a trace of sulphide of iron, by using a machine of an intense type, it is possible to make use of its magnetic properties to separate it from non-magnetic materials. Usually the material is given a slight preliminary roast in order to convert the sulphide of iron present into magnetic sulphide or oxide, which is much more powerfully attracted. As much as 3000



General View Sulphide Corporation Ltd. Works, Cockle Creek, N.S.W.

tons per month of this material are treated, assaying 9oz. silver, 8.3 per cent. lead, and 27.2 per cent. zinc. From this is obtained a concentrate running 13.4oz. silver, 12.2 per cent. of lead, and 38.9 per cent. of zinc, the recoveries being 73.7 per cent. of silver, 74.1 of lead, and 72 of zinc. These concentrates also carry about 20 per cent. of silica. The gangue consists of rhodonite, garnet and silica. Potter's process depends on the fact that when aciduiated solutions are in contact with certain sulphides, that certain gases will be evolved and attach themselves to the sulphides and remove them from the gangue. Taking a cheap commercial acid, such as sulphuric, he recommends in this case to take from a 1 to a 10 per cent. solution, and to heat it to near boiling point, when it will be found that if pulverised Broken Hill sulphides are under treatment, that bubbles of gas are evolved, which attach themselves mainly to the zinc sulphide, and carry it to the surface of the liquor. On a small scale, the recovery of zinc is perfect, and even on a working scale as much as 90 per cent. of the zinc sulphide may be recovered. For instance, taking an ore running 7oz. silver, 7.5 per cent. lead, and 18.5 per cent. zinc, the concentrates obtained in Potter's machine was — silver 11oz., zinc 49.07 per cent., lead 7.3 per cent., which is 10 per cent. higher in zinc, and much lower in lead, than that obtained by magnetic concentration. The tailings also are almost free from zinc. A modification of this process, known as Delprat's, or the Salt Cake process, has been experimentally used at the Proprietary mine. In this, Potter's solution of sulphuric acid is densified with salt cake, which is the commercial name for crude sulphate of sodium. It is claimed that less waste gas escapes with this, and that the solution being denser requires less gas to lift the sulphide particle.

The concentrates containing the zinc obtained by the Mechernich magnetic process are roasted in hand-rabbled reverberatory furnaces These roasters, of which there are seven, were erected for dealing with ores by the Ashcroft process, and have hearths 62½ feet long by 12 feet wide. They have nothing to recommend them, and the wonder is that cheap work can be done at all. The price of coal is low, but this is not justification for such cumbrous and ineffective types of roasters. The present management is not in favor of such furnaces, but they are some of the last relics of the Ashcroft plant. It need only be said if the rest of the appliances for an intricate chemical process were no better than these, there is little wonder it failed. In roasting zinc-lead sulphides, a great deal of trouble is experienced, even with the best furnaces. In the subsequent process of reduction and distillation, it is essential that sulphur be removed from the zinc. If finely powdered zinc sulphide is rapidly heated in a current of air, it is mainly converted into oxide.

$$2ZnS + 3O_2 = 2ZnO + 2SO_2$$

With a limited supply of air, and a low temperature, a large amount of zinc sulphate forms.

$$ZnS + 2O_2 = ZnSO_4$$

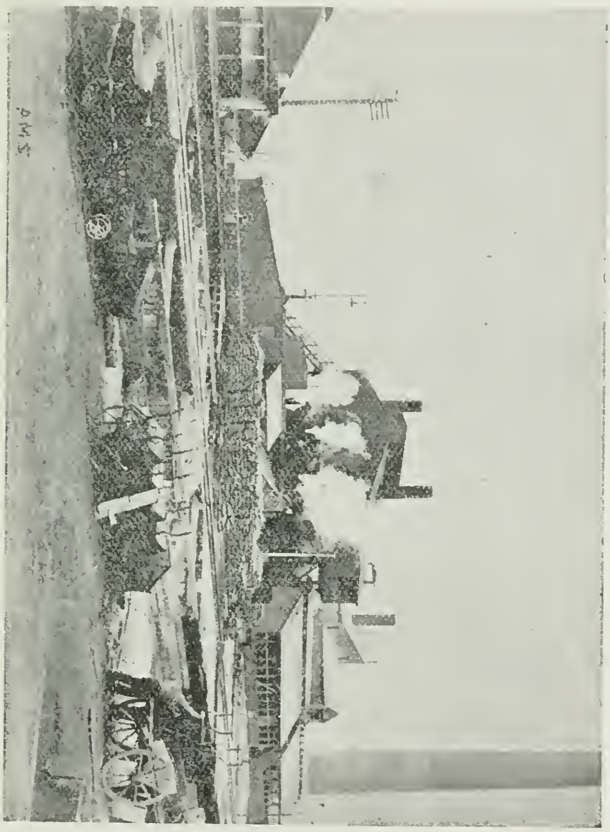
On heating zinc sulphate to bright redness, it is partly decomposed, basic zinc sulphate forming, and sulphur dioxide and trioxide being given off. It requires a white heat to decompose the whole of the sulphate into oxide. If zinc sulphate is not decomposed, then, if it is placed in a closed vessel with carbon, it will be reduced to zinc sulphide again, so that there is no gain in roasting it to sulphate unless the sulphate is split up and the sulphur eliminated. As a rule, zinc sulphide is associated with lead sulphide, or galena, so that roasting of the associated minerals must be considered. In the presence of zinc sulphide, the tendency of the lead is to become sulphate. The sulphate of lead on the outside of a particle of galena may react with the sulphide in the interior, and produce metallic lead, if the temperature is high. Any silver adjoining the lead will be absorbed, and with a high temperature, oxidation of the lead may be accompanied by a certain amount of volatilisation of both lead and silver. The lead ultimately oxidises, and if silica is present will unite with it and form a silicate. The sulphate of lead, though not decomposed at a white heat into lead oxide, and sulphur trioxide, will at a bright red heat react with silica, giving silicate of lead and sulphur trioxide. Both silicate and oxide of lead are apt to form a glaze over the surface of other materials, thus preventing direct contact with the oxygen of the air.

If an oxidising flame alone plays on the surface of the ore, a slight amount of lead and silver may be volatilised, but no zinc should escape, since the sulphide, the sulphate and the oxide are non-volatile. If a reducing flame acts near the fireplace, the oxide of zinc may be reduced to metallic zinc, and as this is highly volatile there may be loss.

In the ordinary process of zinc distillation, the oxide is heated with carbon in a closed fire-clay vessel, and the reduced zinc distils over and is collected in a receiver. When oxide of lead is present, a considerable portion of this also is carried over, while some of the oxide of lead combines with the silica of the vessel and rapidly corrodes it. The problem of obtaining zinc is further complicated by the fact that at a red heat the metal decomposes water vapor, liberating hydrogen, and forming zinc oxide. Under the same conditions, it will decompose carbon dioxide, liberating carbon monoxide.

$$Zn + H_2O = ZnO + H_2$$
$$Zn + CO_2 = ZnO + CO$$

These reactions only take place when the gases mentioned are in excess. If hydrogen or carbon monoxide is in excess, the reverse reactions take place. The temperature required for the reduction is always so high that the metal is volatilised at the instant it is produced. It boils about 900deg. C., and melts at 432deg. C., or, if a mass of zinc is heated in a vessel with only a narrow outlet, it will first melt at 432deg. C.; on continuing to heat, the metal will boil about 900deg. C., and if heating is continued, the whole of the metal will evaporate away or volatilise. If the vapor is suddenly chilled to below its melting point, or, say, 400deg. C., the vapor will instantly turn to a finely divided solid, as water vapor may be condensed to fine particles of ice by being chilled below the melting point of ice. Now, when zinc comes down in this state, it is almost useless. The finely-divided zinc dust is not wanted. Again, if the zinc vapor is mixed even with inert gases, the partial pressure it exercises will only be due to the amount of its own vapor present. If the other gases are in considerable excess, then it is not possible for the zinc to condense as a liquid, even if the temperature is high enough. For this reason, it is not possible to obtain liquid zinc by condensation from reverberatory or blast furnaces. It is essential, then, for a good recovery of zinc, that the reduced and vaporised metal should be cooled to a temperature between 420deg. and 550deg. C., so that the vapor may be transformed to the liquid state, that no gases capable of reacting with the oxide be produced save in small quantities, and that the zinc vapor should exercise the greatest partial pressure possible. In other words, it must be diluted as little as possible with other gases. As previously



Ore Receiving Trucks and Floors Over Ore Bins.

pointed out, if the ore contains sulphur, then an equivalent of zinc is locked up. If it contains lead, iron or manganese as oxides, then these eat into and corrode the vessels, and also form a fusible slag, and seal up considerable quantities of zinc oxide. For these reasons, the metallurgy of zinc must be looked upon as offering more difficulties than any common metal, and the high price of the metal, in spite of the vast supplies of ore, is mainly due to metallurgical difficulties. It would seem as if the metal would have to be produced by new and simpler methods before its price comes down.

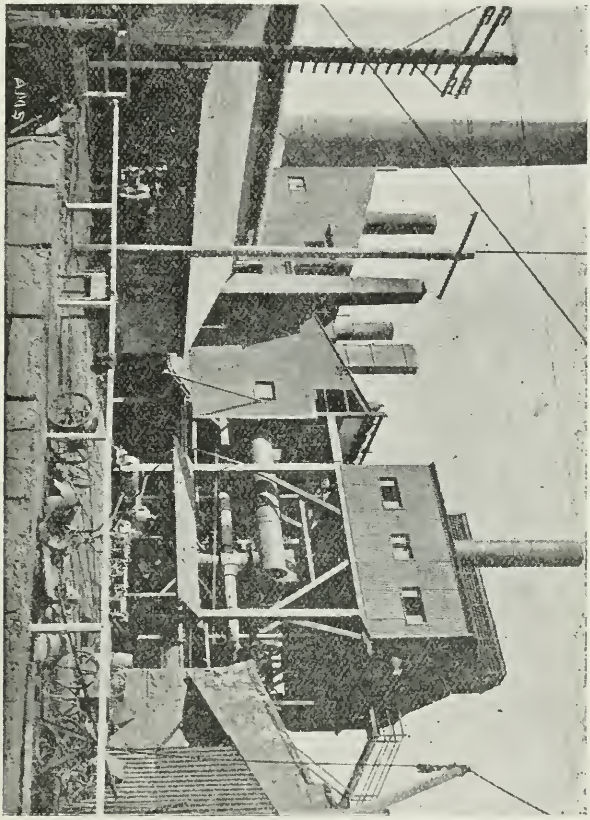
The zinc ore treated at Cockle Creek comes from the Central mine at Broken Hill, and is the product from the middlings by magnetic concentration. It runs about 40 per cent. of zinc, 12 per cent. of lead, and 13oz. silver per ton. The balance is made up mainly of sulphur, iron and silica. The material, the bulk of which would go through a 20, but stop on a 60, mesh sieve, is roasted in long reverberatory furnaces, originally erected for the Ashcroft process. A considerable volatilisation of zinc is said to take place in the operation. The roasted product runs about 36 per cent. of zinc and 20 per cent. of silica. It is not exactly clear why zinc should in some circumstances be volatilised during an oxidising roast. Neither zinc sulphide, nor oxide, nor sulphate, are volatile. It is possible that with such large roasters, a great deal of coal must be used to maintain the requisite temperature, and that especially near the hearth the flame may be occasionally reducing, and thus reduce some oxide to metallic zinc, which is highly volatile. A remedy for this would be a different type of furnace, or the use of gaseous instead of solid fuel. It is possible that the sulphide and oxide react at high temperatures, as suggested by Percy, and produce metallic zinc, or that some intermediate product is formed which is volatile. Apparently the escaping product is held in solution by some escaping gas. It is certainly impossible to wholly desulphurise a refractory mixture of sulphides in such a furnace as this unless at a very high cost.

The product from the roasters is mixed evenly with from 30 to 40 per cent. of a mixture of crushed coking coal and coke, and is fed into fire-clay vessels and strongly heated. The coal present cakes and cements the whole mass together. Volatile hydrocarbons are given off, and subsequently at a higher temperature the zinc distils over, leaving the lead and silver disseminated evenly through the remaining coke.

The retorts or pots used are made at the works from local clay. The clay is worked up to a uniform plastic state by the action of a pug mill. It is then allowed to mellow. What happens in this operation is not yet definitely known, but certain changes take place which are essential for the construction of the finished article. The action has lately been stated to be due to the presence of certain bacteria. The mellowed clay is then raised and mechanically compressed into blocks, each one being sufficient to make a pot; this is placed in a press, and subjected to a pressure of 5 tons per square inch, the clay being forced round a maudril, the projecting end is cut off with a piece of wire, and the raw pot withdrawn. The pots so made are stacked in an upper drying room. After the outside moisture has evaporated, they are shifted to the next room, where they are heated to about 90deg. F., and after some days are transferred to the lowest drying room, where they lose the remainder of the water mechanically held. They are then baked in an oven.

The receivers, into which the zinc condenses, are moulded on a cone by hand, dried and burned. They are much softer and more porous, and much more fragile than the pots. Each pot lasts from 30 to 77 days, each charge taking 24 hours. The size of the pots are about five feet long, 12 inches high, and 9 inches wide externally, the clay being $1\frac{1}{2}$ inches thick at the top and bottom, and 1 inch at the sides. Each pot holds a 1cwt. charge.

The furnace for heating the retorts is of the regenerative type. The fuel is placed in a gas producer, and the gas is led away into the furnace through a mass of checkered brickwork; air is similarly led through parallel chambers, and meets the gas in the furnace. The product of combustion, instead of being sent to the stack, is passed through chambers filled with checkered brickwork, which becomes heated up to the temperature of the waste products. As soon as this takes place, valves are turned, and the gas and air are led in through the heated brickwork, and consequently when they burn in the furnace raise the temperature proportional to their initial temperature, as well as to that due to their combustion. The



New Blast Furnace and Building—Old Furnace on Left.

products of combustion escape through the first-mentioned masses of brickwork, heating it up. After a time, the valves are reversed, and the incoming gases are heated to a higher temperature than in the previous case. Each time the valves are turned, the temperature of the incoming gases are raised, and the temperature of the furnace is raised, and also the temperature of the heating chambers. It would be possible to have such a cumulative heating action go on until the brickwork would soften, or until the temperature of dissociation had been reached. In practice, it is found that while the temperature, owing to cumulative action, rises rapidly at first, a limit is approached, when the temperature rises much more slowly.

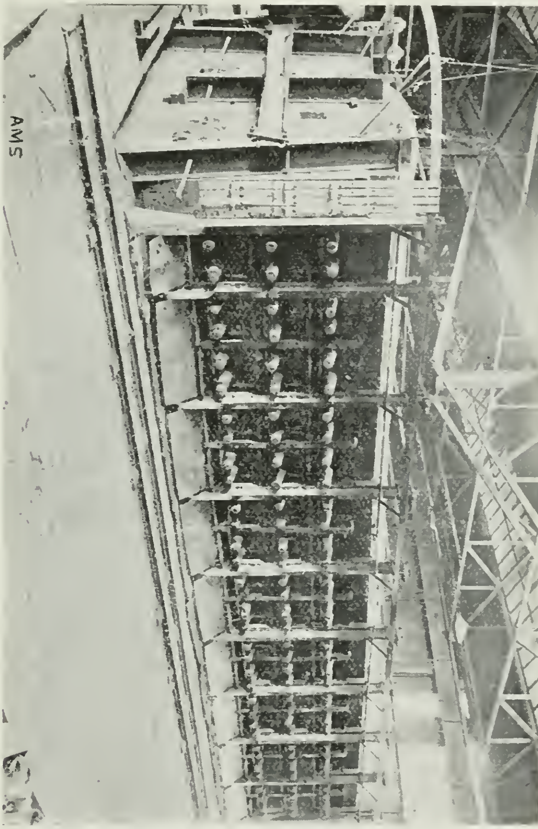
Such furnaces are now commonly used in metallurgical work, where high temperatures are desired. They are open to one objection in heating zinc retorts. If any zinc vapor should escape from cracks, or due to the breakage of retorts, the zinc would escape, and oxidise, ultimately choking up the flues. In spite of this drawback, they are invariably used in Belgium, the home of the zinc distilling industry. The furnace, or heating compartment, is a long, rectangular chamber, having an arched top. The walls on each side have three rows of openings, each row having thirty-three, or 198 in all. Provision is made for the support of the pots, which are inserted nearly horizontally through the openings, the forward, or open end, inclining slightly downwards. The adapters of fire-clay are fitted into the mouths of the retorts, and project beyond the walls of the furnace. Each pot is supported at back and front, but is surrounded by flame as in the same way as a muffle.

The process used cannot be said to be new. Crushed coke and pitch had been used in other places with the same objects in view, namely, the distillation of the zinc, and the recovery of the lead, in a form suitable for feeding into the blast furnaces. At Cockle Creek, the local coals are very gassy, often giving no more than 50 per cent. coke, so that these have the same action as pitch. The ore at present being treated runs about 36 per cent. of zinc, 12 per cent. of lead, and 20 per cent. of silica, and from 12 to 14oz. silver per ton. After being roasted, the finely-divided material is uniformly mixed with from 30 to 40 per cent. of its weight of crushed coal and coke. About one cwt. of the mixture is fed into the retort. The temperature is high enough to decompose the coal at once. At a dull red heat, the hydrocarbons are driven out. As the temperature rises, the zinc oxide is reduced by carbon, and the zinc vapor escapes and liquefies in the condensers. A portion of the zinc passes from the state of vapor to that of blue powder, either through being chilled too rapidly, or through being diluted with too much gas. This material, which forms at the beginning and end of the operation, consists of zinc, containing zinc oxide. The liquid metallic zinc, known as spelter, is run into moulds, and sent to the market. There is practically no cadmium in the ore now being treated. Curiously enough, more of it may be found in the slimes than coarse concentrates. Arsenic, which is present in the ore, does not pass into the zinc at all; while the amount of lead present does not exceed from one-half to one per cent. The silver, gold and balance of the lead remain with the coke.

I was not able to learn what the recovery of zinc or other metals were, but from operations with similar processes elsewhere, the recovery is stated to be about 70 per cent.

The material left in the retorts consists of a coked mass, containing practically all the lead. Part of this is in the form of metal, very finely divided, part in the form of sulphide. The remainder has united with silica, and occasions much trouble by corroding and eating through the retorts. Part of the zinc remains in the retorts, mainly in the form of sulphide; some penetrates the clay of the retort, and some unites with the clay itself. About 20 per cent. of the original zinc values remain behind. These residues, consisting of coke lead, silver and a small portion of zinc, are suitable for treatment in blast furnaces.

The metallurgical plant for the recovery of zinc, although highly interesting, and the only one in Australasia, is but a small section of the Sulphide Corporation's plant. Lead concentrates from the Central mine, which assays lead 60 per cent., zinc 10 per cent., and silver 30oz. per ton, as well as purchased ores, containing these metals and gold, are reduced on an extensive scale. The ores to the works are conveyed by rail from Newcastle to the company's siding, where extensive bins are erected. The sampling mill has a capacity



Spelter Furnaces for Treatment of Broken Hill Ores.

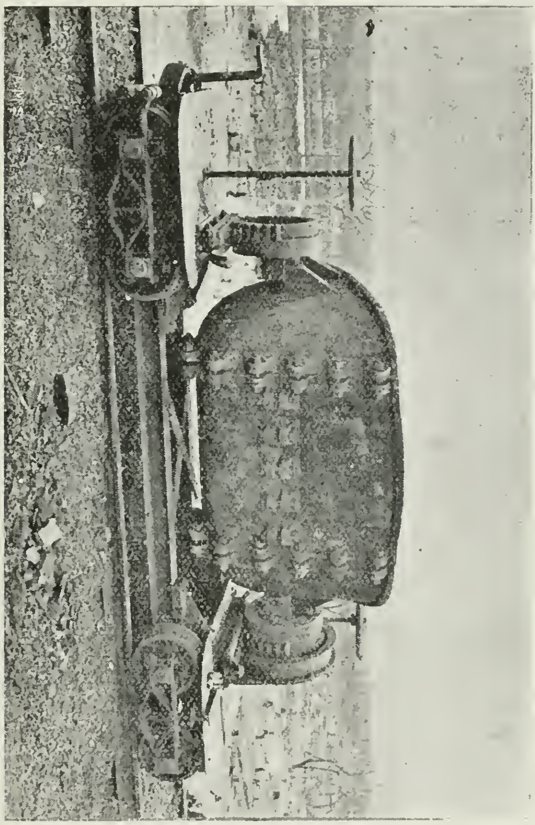
of 200 tons per day. The final sampling is done by coning and quartering.

The sulphide ores are crushed in a No. 5 ball mill, and roasted. There are seven hand-roasters of the type described. These are mainly used on the zinc ores. Several more of these have been dismantled to make room for mechanical roasters. Provision is made for eight of these. The type is somewhat the same as used by the Tasmania Smelting Co., Zeehan, but substantial improvements have been made here. Each roaster consists essentially of a low dome-shaped roof, on a circular wall. At a short distance below the roof is a hearth, capable of revolving on a vertical axis. This pan, or hearth, is built up of sectors of cast iron, bolted together;

is 21 feet in diameter; it is filled with about six inches of brickwork and tamping. It is supported on rollers, and driven from below.

The ore is fed through the centre of the dome of the furnace. A fixed rod passes from the circumference to the centre of the furnace. On this are arranged a number of blades equally spaced, capable of adjustment at an angle to the axis of the rod. As the hearth rotates, the ore, which is fed in at the centre by means of Challenge feeders, is deflected by one of these blades towards the circumference. After passing the fixed rod, the particle of ore is not disturbed until it again reaches the arm. The second blade throws it outward, where it lies until it is carried round again; and so on until it has undergone as many revolutions as there are blades, when it is delivered at the circumference. By placing these blades at right angles to the arm, the ore would not move forward at all; by giving them a slight inclination towards the approaching ore, a small portion would be deflected; with a greater angle, more would be thrown over, so that the time of discharge will depend on the angle of the blades and the speed of the hearth. The fireplace is built outside the furnace, and near the discharging rabble. The dome-shaped roof has projecting pieces, which causes the heated gases to pass over the ore which has been in the furnace some time, thence over the raw ore before they escape. There is no doubt that this type of furnace as improved at these works has many advantages; but perhaps the main one is the simplicity of the rabbling apparatus, and the small amount of wear and tear as compared with most other mechanical furnaces. The ore after roasting contains from 6 to 8 per cent. of sulphur. In order to eliminate this, it is treated by the Huntington-Heberlein process. The semi-desulphurised ore from the roasters is in a fine state of division, the original ore having passed through a 0.5 m.m. screen. This is mixed with limestone, which has passed through a 10 to 15 mesh sieve. In each case, at least 60 per cent. of the material is finer than the particles, which just pass through the screen. The limestone may be added before or after roasting, but the proportions are so arranged that it carries from 6 to 9 per cent. of lime and from 25 to 50 per cent. of lead. If silica is not present, it is also added. The amount of silica, as slag making material, may be as low as 20 per cent., excluding the lead contents, but such slags are not economical, since they require a high temperature, and chill at once without going through a viscous state. Manganous and ferrous oxides are also present in proportions to form a good slag. If too much lime is present, the resultant material is useless, for it will crumble. It is found that a uniform and intimate blending is necessary for good work, otherwise blotches of unacted-on ingredients will remain in the final product, and the pots will volcano, or the heated gases will spurt through orifices instead of bubbling through the mass in the converters.

The next stage of the operations is carrying out the essential part of the Huntington-Heberlein process. It is remarkable that at Port Pirie, at Fremantle, and at Zeehan the process may be seen in active operation, yet here the owners of the patent rights have stipulated that no outsider shall be admitted. Since the process was familiar to me, having witnessed all the operations elsewhere, the restriction was a somewhat narrow one. A description of the converting vessels used has already been given in a series of articles on Tasmania. Briefly, a small inverted cone of sheet iron is hung on trunnions: the diameter and depth are about 4 feet 6 inches. Larger vessels nave been installed elsewhere with advantage. A perforated plate, or colander, is placed horizontally about 9 inches from the apex of the cone. An air pipe is connected with the space thus formed. The ore from the roasters, mixed in the proportions indicated, or mixed so that excluding the lead, the silica and bases present will form a good slag, is fed in hot to the converter until it is filled; a gentle stream of air is admitted, the



Large Slag Car Lately Adopted.

mixture warms up, the blast is increased, until towards the finish of the operation it reaches 20oz. The temperature increases, the material softens, frits and finally melts down into a compact stony-looking mass. The time taken is about 5 hours. The fused or sintered slag-like material is tipped out of the converter by inverting it. The material is broken up, and is ready for the blast furnace. It is found that if a little hot ore is placed on the bottom of the converter, and a cold mixture on top, that the whole mass will warm up when the air blast is turned on.

The chemistry of the process has not been definitely worked out. The inventors believed that calcium dioxide played an important part in the actions, but many other reactions go on; for instance, if hot lead sulphide and lime are mixed, calcium sulphide and lead oxide will form.

$$CaO + PbS = PbO + CaS$$

Other changes take place as well when heated air is present, calcium sulphite and ultimately calcium sulphate forming. Lead sulphate also forms by the action of oxygen on lead sulphide; litharge or lead oxide at a comparatively low temperature becomes red lead, $3PbO + O = Pb_3O_4$. The peroxide of lead in this is instantly acted upon by any escaping sulphur dioxide, lead sulphate forming with a great rise of temperature.

$$Pb_3O_4 + SO_2 = PbSO_4 + 2PbO$$

Calcium sulphate galena and oxygen give calcium plumbate, with the evolution of sulphur dioxide.

$$CaSO_4 + PbS + 3O = Ca\ PbO_3 + SO_2$$

And lead sulphate and lime give the same products.

$$CaO + PbSO_4 = Ca\ PbO_3 + SO_2$$

Other bases as well as lime will give much the same action. The main reactions, which go on, must be strongly exothermic, for the temperature rises to at least 1000deg. C. Metallic lead is also found in the fused mass, no doubt from the well-known inter-actions between the sulphide and oxide, and between the sulphate and sulphide.

$$2PbO + PbS = 3Pb + 2SO_2$$

$$PbSO_4 + PbS = 2Pb + 2SO_2$$

In all these cases, sulphur is eliminated. The main object of the patentees seems to have been the elimination of sulphur from galena, but the most valuable part of the process seems to be the desulphurisation of refractory minerals, like blende, and therefore transform one of the most troublesome metallurgical compounds into a comparatively harmless one. This action is probably brought about by the oxidising current passed through at such a high temperature; zinc sulphide is oxidised to oxide or sulphate, while the latter is split up into oxide, sulphur dioxide and oxygen being eliminated. In the final product, there is only one per cent., or even less, of sulphur; the material is in an excellent condition for smelting, while all trouble as to briquetting the fines has been got over, and the final output is increased by more than 100 per cent.

The product from the converters is elevated to the smelter-floor; here it is mixed with other ores, fuel and fluxes, and smelted. Limestone comes from Portland, the ironstone from the Blythe River, Tasmania; the coke is made locally in Beehive ovens.

The furnaces, of which there are three, have been, or are being, remodelled on other designs. They are 144 inches by 60 inches at the tuyere level, and are provided with two rows of cast-iron sectional water jackets for a height of 3 feet 10 inches. They are provided with six three-inch tuyeres on each side, there being no end ones. The height of the feed-floor above this level is 21 feet. The throat of the furnace is partly closed by a cast-iron hood, of the shape of an inverted pyramid, without an apex, dipping into it, leaving a space at the top of about 18 inches between the wall of the furnace and the hood. Into this space the charge is tipped; the escaping gases and fume pass out through the central hood. By this arrangement, the finer particles of ore drop down the sides, the coarse running to the centre of the charge. Blast pressures up to 50oz. are used.

Such high pressures, combined with tall furnaces, are in accord with the best modern practice. Upwards of 5000 cubic feet of air are forced through the tuyers of each furnace per minute. The bullion is tapped out into pots, and placed in a drossing furnace, from which the clean bullion is run out into moulds arranged on a circular rotating table. A fluid sample is taken on every 25 bars, and assayed. It is found the distribution of precious metals is too uneven to take chip or drilling samples. The bullion is shipped to England.

The slags are run through a water-cooled slag spout into a forehearth at the end of the furnace, the overflowing slags going into pots. These are emptied over the dump, the shells going back again to the furnace. The slags made are of the usual type, but slags down to 19 per cent. of silica have been made, or silica (SiO_2) 19; ferrous oxide (FeO) 32; lime (CaO) 18; zinc oxide (ZnO) 20. Such slags, however, are not desirable. Excess of zinc oxide tends to make a slag mushy; the addition of more lime makes a slag freer from lead, because there is less enclosed matte. There should not be less than 10 per cent. of lead in the charge.

Air is supplied for the smelters and H.H. convertors by four No. 5 Baker blowers. In order to get the high pressures required for the former, provision is made for coupling the two pipes leading from the blowers together, or forcing the larger volume through a smaller pipe. Owing to slip and leakage, this is not regarded with favor, and a blowing engine will shortly be installed. Steam is generated in 4 Babcock and Wilcox boilers; all the coal fed into these is weighed, and automatically delivered. The works are fitted up with engines, dynamos, motors, and all subsidiary plant, which would take too long to describe in detail.

Smelting & Refining Co. of Australia (1901) Ltd., Dapto.

The smelting works established at Dapto, N.S.W., are, perhaps, the largest Customs works in Australia. Lead ores, as well as auriferous and argentiferous ores of all descriptions, are purchased on their assay values, and refined lead, gold and silver bullion produced. In addition to this, the treatment of the famous New Caledonian nickel ores is about to be undertaken on a large scale. The works are situated on a knoll overlooking Lake Illawarra; a strip of fertile country lies between the coast and the range of precipitous hills in the background. The trip from Sydney, either by road or rail, gives one glimpses of some of the best Australian scenery. Mountain cliffs and rugged gorges lie on one side, and the ocean, sometimes hundreds of feet below, may be overlooked on the other. The native forest is rich in semi-tropical vegetation, while patches of lower lying land are exceedingly prolific. Gold and grass, as a rule, do not go together, but the coal-bearing measures of this area, which originally furnished forests now converted into coal, furnish a soil which supplies all the elements necessary for plant life. The towns on the road must date from the early history of Australia; the quaint two-storied, gable ended buildings, the utilisation of roof space, the churchyards, and other signs of old England, are not known in new Australia. The famous mines about Wollongong, Bulli, Mount Kembla, and the coke works along the line, are signs of the mineral wealth of the district. No doubt the proximity of such an excellent coking coal was the main factor in establishing the site of the works on the south coast; but at the same time the cost of carrying from Sydney is a big tax on the income of the company. Lake Illawarra is closed by a sand bar: the lake itself, even on the removal of this, would only be navigable to vessels of moderate draught. The ores coming to Dapto are shipped to Sydney, and then entrained and delivered at the works. Each parcel is weighed, the moisture estimated, and sampled—every second, fifth, or tenth bag, in accordance with the size of the parcel, its richness and the variability of its mineral contents. If the ore is coarse it is put through the rock breakers, thence it goes to the sampling floor. It is here piled into a cone, which is flattened out by working the sides down until a flat, low, circular heap is left. This is divided by two diameters at right angles to each other, and the opposite quarters are taken. Half the sample thus chosen is crushed finer if necessary, and this is coned and quartered as before. This operation is carried on until only a few pounds are left. This sample goes to the assay office, where it is crushed down, and passed through an 100 sieve. Any metallic particles or scales, called metallics, which are too coarse to pass through the sieve are collected into one button, and the value of the parcel for metallics determined. The fine material is quartered, and the assay quantity selected. This is assayed, and the value determined. This value, added on to that due to the metallics, is the total value of the ore. Samples of fine ore or concentrates are coned and quartered until the final sample sent to the assay office is taken. Mr. Hoyt has also made assurance doubly sure by taking the second half of the parcel, and sampling it down independently of the first, there being thus two samples taken, which, if the sampling is correct, will check each other. When the final assay samples are prepared, parcels of three are made up. One is held by the company, one goes to the owner, and if there is a difference between these values as determined by assay, a third goes to an umpire assayer. If his assay agrees with either the buyer's or seller's, then the loser pays all expenses. The owner of the ore may watch the whole proceeding of sampling through a windowed room. This precaution has been found necessary to prevent the possibility of owners salting worthless ore. With regard to certain ores from Victoria sent to the works I may state, from personal knowledge, that the company's assays invariably turned out a shade better than those done at the works: in this case also there was no representative present. The great majority of the parcels are in small lots, and the sampling, assaying, and smelting of these throws a lot of work on the staff, which is unknown on any big mine having its own smelters.

After sampling, ores containing sulphur and arsenic are sent to the lond hearth hand-rabbled roasters. There are 10 of these, each with a hearth area of 33 feet by 15 feet, and internal height of 2 feet 6 inches, and one with an area of 62 feet by 15 feet. These are not viewed with favor by the management, modern mechanical furnaces being preferred. Samples are taken each shift from each furnace, and are estimated accurately for sulphur. When working on the same charge the samples are generally bulked in pairs for assay. As most of the roasted ore is sent at once to the bed floor to smelt in with the bed ore, the gold, silver, copper, lead, iron, silica, and sulphur are all determined.

The sulphide ores for roasting go to storage bins, arranged on an upper floor. The bed ores are placed in bins on a lower floor. The quantity of the bed ores is always large, and their composition is accurately known: other ores are worked in with these, so as to produce a bullion of desired value in gold, silver, and lead, and to reduce the quantity of barren fluxes required to a minimum. As examples of bed ore analyses the following will serve:—

Lot	SiO_2	Pb.	S.	Fe.	CaO.	Al_2O_3	Zn.	Cu.	Sb
1.	33.4	28.2	6.7	7.0	2.5	—	4.0	tr.	—
2.	33.4	20.0	7.0	14.0	4.4	8.8	5.4	.3	—
3.	39.4	13.4	9.6	14.0	3.0	6.0	3.8	.6	4.1
4.	69.6	7.0	3.8	4.2	1.5	3.8	2.3	tr.	—
5.	26.3	14.0	12.6	22.6	1.4	2.5	4.0	3.8	—
6.	27.0	13.7	8.6	20.4	1.4	3	4.3	3.2	—
7.	49.0	10.8	5.1	17.6	.5	—	3.1	tr.	—

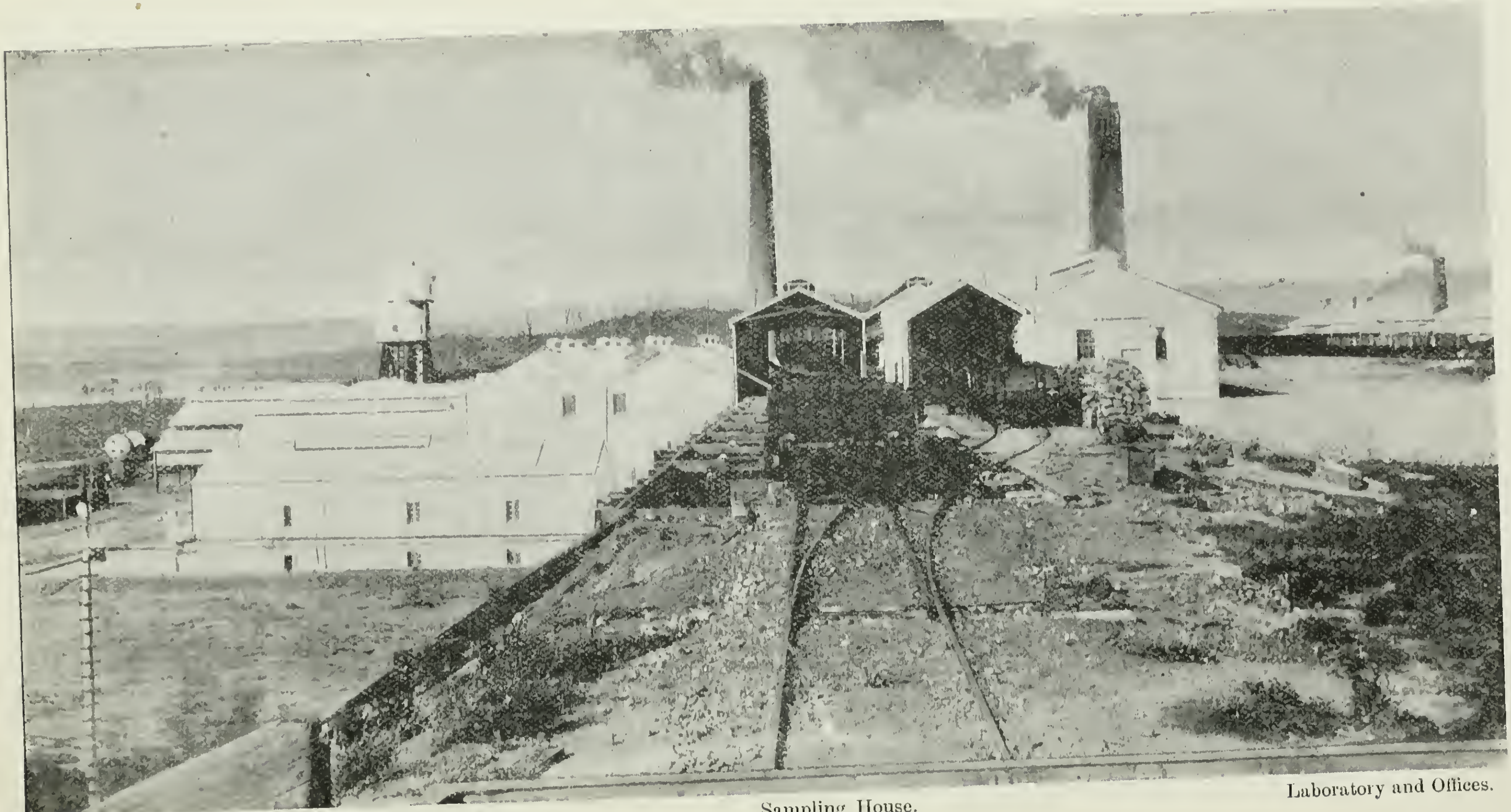
There are three large 120-ton smelters for running on ores, and a small one for running down antimoniate of lead. The ore smelters are 42 inches by 140 inches at the tuyeres, and have a height of 20 feet above the tuyere level. The crucible is a rectangular iron box, supported on a brick foundation. This of necessity is strongly built, and strongly braced and lined with firebrick and the necessary brasque, and provided with the Aarents syphon tap. The crucible holds 20 tons of lead. The lower portion of the smelter is water jacketed: the jackets are made in sections of cast-iron bolted together. The corners are rounded off, thus giving an oval section.

There are eight 3-inch tuyeres on each side. The usual removable mica cover for the peep hole has been replaced by a plug of wood driven in. This may be easily removed if it is necessary to rod the tuyeres. At the same time the wood acts as an indicator, for should slag overflow into the tuyeres the wood will shrink, and at once be blown out. The upper portion of the furnace above the jackets are built up of brickwork in the usual way, and supported independently on cast-iron pillars. One furnace is fed through hoppers at the side; the others through charging doors on each side in the usual way. A blast pressure of 40oz. is used. The slags flow into a fire-hearth, thence into slag pots, which are sent over the dump. Each pot holds 18cwt. of slag. Two are arranged on a carriage, which is drawn by a horse to the edge of the slag dump. The shells are returned to the furnace. The lead flows into the moulds from the smelter. A number of parallel moulds are placed on a carriage, which travels on rails parallel to the length of the furnace. As soon as the end mould is filled the next is moved up, and so on, until they are all filled. The slags are sampled periodically. An iron rod is dipped in the slag, then chilled at once by immersing in water. In this way it is much more easily decomposed. The following analysis give types of slags produced at the works:—

SiO_2	FeO	CaO	Al_2O_3	ZnO.	MnO.
34.4	33.7	11.2	7.3	7.4	2.1
31.4	34.4	13.6	9.5	8.0	1.7
28.6	31.6	17.2	6.4	10.6	2.1

The lead may run from 0.5 to 1 per cent.

Copper ores are also run down in these furnaces for matte without running the lead out of the crucible. After the copper ores are run through, lead ores are charged in again, thus obviating the blowing out of a furnace. It might be thought that the matte so produced would be deprived of its precious metals, but such is not the case, probably owing to the short time of contact with the molten lead. The copper matte produced is concentrated up to 50 per cent., and shipped away. The lead bullion, containing practically all the gold and silver from the ores and fluxes, sent into the furnace, also contains many other metals. Antimony, tin, arsenic, copper, bismuth, sulphur, zinc, and other elements are usually present. The values for silver and gold are accurately determined by sampling every 150 bars of lead bullion; in addition to this, dip samples are taken every morning and evening, and assayed for gold and silver. Before the lead can be desilverised it is necessary to remove the copper, arsenic, antimony, and tin. This is done in a water-jacketed reverberatory furnace. There are two, each capable of holding 15 tons of lead bullion. The lead is just melted; the copper present forms an alloy with some of the lead. This alloy has a higher melting point than lead, consequently it rises as a solid. The surface of the lead bath is covered with ashes. The copper dross is intermingled with these, and scraped off the surface. Practically the whole of the copper is removed in this way. On raising the temperature, tin, when present, becomes oxidised. This unites with part of lead oxide produced, and gives a powdery stannate of lead. This is also scraped off. Next arsenic and antimony are successively removed as arseniate and antimoniate of lead respectively, the former being brown and the latter darker. After the antimony has all oxidised the lead oxidises, and forms litharge, the scum forming



Railway Line—Trucks Discharging into Ore Bins. Roasting Plant.

Sampling House.

Laboratory and Offices.

becoming more fluid as the operation proceeds. The lead now contains only gold and silver, and slight traces of other metals. It is then run into a zincing kettle or pot, there being three of those. These pots are made of cast-iron, and are segments of a sphere. They are about 6 feet in diameter and 3 feet deep, and are heated by a fireplace below. Zinc is fed into the molten lead. The amount of zinc required varies with the grade of the bullion. In all cases the lead must be saturated with zinc. Then the excess of zinc will separate out, taking the gold and silver with it. If equal quantities of lead and zinc are melted together, then at about 700deg C. the two metals will apparently separate, but the zinc will dissolve 1.5 of lead, and the lead 1.3 of zinc, so that, as a general rule, something over 1.3 per cent. of zinc requires to be added even for low-grade bullion. Plattner gives 1.34 for 30oz. bullion, and 1.84 for 120oz. bullion, and 2.45 per cent. zinc for 250oz. bullion; so that while it takes 1.34 per cent. to get the first 30oz. only, .5 per cent. extra will take out three times as much extra. It is usual when gold and silver are both present, and the silver in very large excess, to fractionally remove part of the silver and all the gold with the first addition, and the balance of the silver by a second or even third addition. In this case the bullion is so rich in gold that a separation is not attempted. The lead in the kettle is heated to the kindling point of wood. The zinc is added and worked in by means of paddles or spadelles, which the men, by a levering combined with a twisting action, the lead is set rotating in the kettle, and at the time swirled from outside to inside. This is carried on for from 30 to 40 minutes to work the zinc in. When this is done the bath is allowed to cool down, the excess of zinc carrying with it the gold, silver, and copper, rises to the surface. After a time the zinc crust attaches itself to the sides of the pot. It is broken off and fished out with a perforated skimmer. These skimmings are then liquated, the lead draining out being very poor in silver. After skimming a dip sample is taken and assayed for gold and silver. The necessary amount of zinc is then added until the pan is clean. The zinc skimmed off the second time is kept for adding to the next kettle, so that practically only one lot of zinc has to be treated for bullion. The lead must now be freed from the balance of the zinc dissolved in it. This is done by blowing in steam. A hood is lowered, steam is blown in, and the agitation brings about contact with the air. The zinc becomes oxidised, the lead cools down, and the zinc oxide crust is removed. The process is repeated the second time, after which the lead will be rendered almost absolutely pure. It is cast into bars, and sent to market. A great deal goes to China. The zinc crusts, even after liquation, still containing more than half their weight of lead, are fed in charges of from 1250 to 1500lb. into Faber du Faur distilling furnaces. These are simply large bottle-shaped pots, enclosed diagonally in a cubical furnace suspended on trunnions. Fire-bars are placed across the bottom of the furnace, coke is fed in on top, the mouth of the pot projects from near the top of the front side, and the flue is opposite this at the back. The advantage of this furnace is that it may be tilted by hand-wheel, with worm-gearing working into a pinion on one tunnion or by a lever only. The temperature is raised, and the zinc distills off, and is caught in a condenser. From 60 to 65 per cent. is recovered as molten metal; the balance is in the form of dust.

The lead, containing the gold and silver, is poured out by tilting the retort. This is then cupelled. There are at Dapto two large cupelling plants, said to be the largest in Australia. Each holds about 4500lb. of lead. These are of the English type—a short reverberatory furnace with a movable hearth. The term cupellation is somewhat misleading for this operation, for, although oxide of lead is formed, it is not absorbed as in the case of assay work for this operation, but is drained away as fast as produced. The cupel, or, as it is called, test, is composed of various materials tamped into an iron framework. Formerly bone ash was used, then marl, but the universal practice now is to use cement with clay or other material. Mr. Hoyt uses only five of cement to one of sand, and finds it wholly satisfactory. Since molten litharge has a most corrosive action on almost every material, such a test would rapidly lose its boundary, for the litharge is always thrown off from the centre to the outside of the lead button; the molten stream of litharge flowing away from the test also corrodes it greatly. To overcome this water-cooled rings or jackets have been let into the test, so as to chill the litharge in contact, and thus prevent corrosion. This was partly successful, but the iron was attacked after a time. Mr. Weinberg, the former manager of the works, substituted an oval copper ring 7 feet 6 inches by 5 feet 5 inches, inside measurement, and provided a small water-cooled breast jacket made of brass. This has proved successful. On account of the cooling of the test the lead is only partly oxidised or enriched in silver in this cupel. Lead is fed in, heated up, a blast of air turned on, and the litharge driven out at a notch in the test. By feeding lead in the level is maintained. The lead becomes enriched in proportion to the bullion fed in. The rich lead is removed to a smaller cupel, not provided with water-cooled rings, and finished off at a high temperature. The resulting alloy, consisting only of gold and silver, is known as dore bullion. This is shipped home for refining.

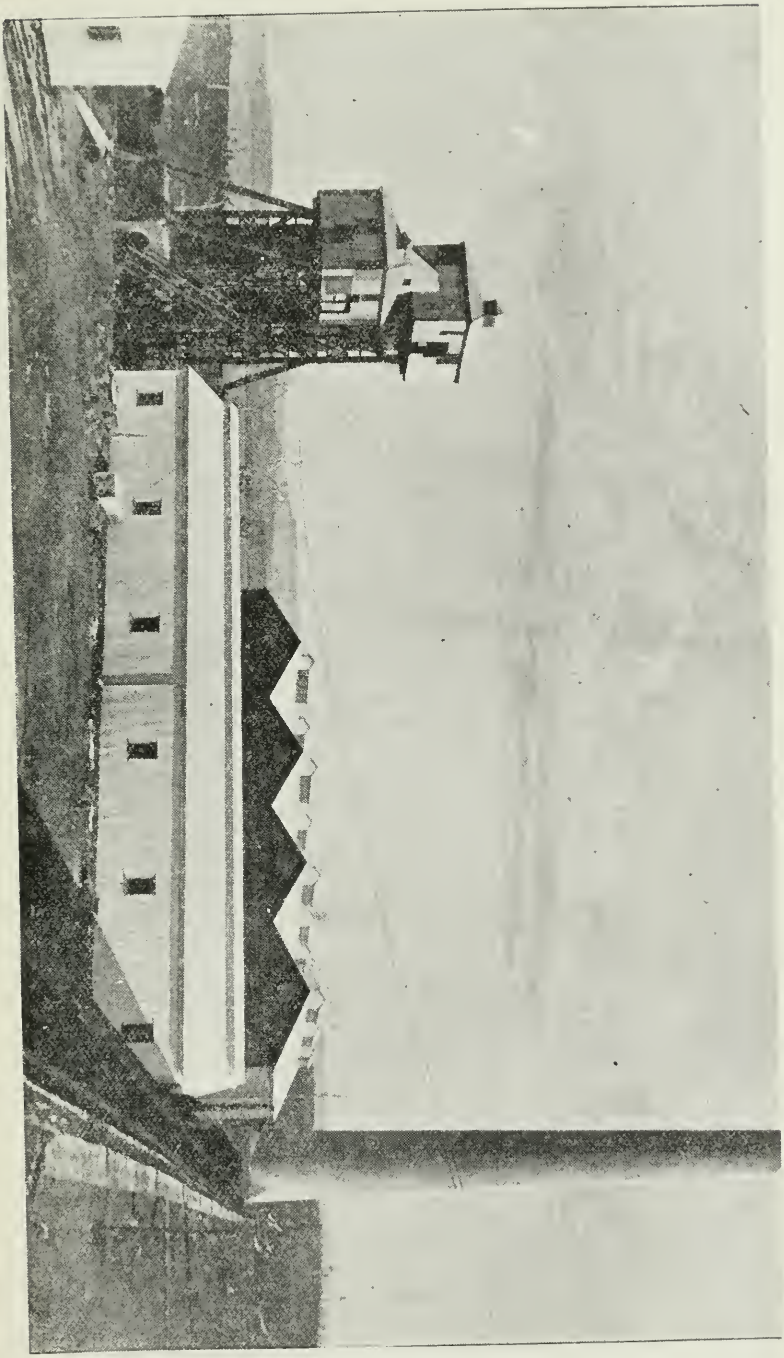
The antimoniate of lead produced in the softening furnace is run down into an antimony lead alloy. The flue dust is fed into one of the roasters, where it is gritted. It is then in a fit state to send to the smelters. It is found that practically all the losses in furnace work, so far as gold and silver are concerned, is due to dusting. The dust contains gold and silver in the same proportion as the ore. Further along the flue, where no dust has reached, there are only traces of gold and silver in the fume, resulting from the volatilisation of the lead. This fume consists mainly of lead sulphate.

The cupel furnaces are made use of occasionally for treating cyanide and other rich slags. These are floated on a bath of lead, the gold and silver present passing into the bullion. The air is supplied to all the furnaces by three blowers, two cycloidal running from 60 to 70 revolutions per minute and delivering 100 cubic feet per revolution, and one Baker No. 5. The plant is equipped with all the necessary motive power, but in this progressive age great alterations are being made to bring the whole of the plant in line with modern requirements. A fine dore bullion refining plant was erected, but is not used, nor is a magnificent sulphuric acid plant. The company has lately erected a large plant for running down the nickel ores from New Caledonia into matte. New Caledonia possesses large supplies of low grade nickel ore, too poor to ship to Europe. The island itself does not contain the requisite fuel or fluxes for dealing with it on the spot, so the Consol Nickel mines, holding large interests, have decided to have the ore dealt with at Dapto. Australia will thus be fortunate in having types of almost every metallurgical process; probably, even iron smelting will shortly be reintroduced. Nickel is a valuable metal, the market price even for large quantities being nearly 2s. per lb., or over £200 per ton. Owing to the extraordinary hardness and toughness it confers on steel, when alloyed with it, even in small proportions, there is no doubt that it will be extensively used in the near future. Its increasing use for plating iron will also cause large quantities to be absorbed. The metal is generally found combined with either arsenic or sulphur, and for a long time attempts to win the metal from its ores ended in failure. In fact, its very name is derived from these failures. The compound of arsenic and nickel has a copper red color, and attempts were made to get the copper out of it that was supposed to be there, but in vain; the German miners, therefore, christened this ore "nickel," which, in South Gorman, is an abusive term applied to an obstinate person, and connected with old "Nick." The allied metal, cobalt, was also christened on account of the refractoriness of its ores from Kobold—an evil spirit or goblin, which was believed to have entered into the ore. It was subsequently found in time that both metals could be won from their ores, but their melting points were very high. The usual method of obtaining nickel is to collect it in a matte if sulphur is present, or a speise if arsenic is present; to roast the sulphide of iron present to an oxide; to slag the iron off until a pure nickel matte remains. This is then roasted to oxide, and reduced to metallic nickel.

The melting point of nickel is from 1400deg. to 1600deg. C., so that it cannot be reduced in an ordinary smelter to the metallic state, and caused to flow like copper or even cast-iron. The ore from New Caledonia is garnierite, or a hydrated nickel magnesium silicate. The usual composition is—

9—17 per cent. NiO.
41—46 per cent. SiO_2.
5—14 per cent. Fe_2O_3.
1—7 per cent. Al_2O_3.
6—9 per cent. MgO.
8—16 per cent. H_2O.

Attempts were made in Noumea to smelt this ore in a blast furnace and produce a nickel iron alloy. This was shipped home, but was a very difficult alloy to purify, so this method had to be abandoned. The usual process is to add some sulphur-bearing flux, and so form a matte containing nickel and iron, the other ingredients being slagged off. It is proposed to follow this method out at Dapto. Gypsum will be brought from the dry lake beds of South Australia, where there are unlimited supplies. This, on being heated with carbon, becomes calcium sulphide, which, in its turn, reacts with iron and nickel oxides, producing sulphides of nickel and iron. The lime liberated at once unites with silica present, forming a slag. The bulk of the ore, which will be sent to Dapto, will contain less nickel and more iron than that indicated in the above analysis,



Sulphuric Acid Plant.

so that on properly adjusting the supply of gypsum a nickeliferous matte of any grade may be produced, the balance of the iron passing into the slag. The metallurgy of the metal is in this respect analogous to that of copper. The matte, having excess of iron, ensures the slag being free from copper or nickel as the case may be.

The matte may be enriched in the same manner as the low grade copper matte, by first roasting and leaving enough sulphur to combine with the nickel; the iron may be slagged off with silica. The usual method nowadays is to run the matte containing iron and nickel into a converter with sufficient sand to slag off the iron oxide. Nickel sulphide differs from copper sulphide in that it is not possible to produce metallic nickel in a converter, because oxide and sulphide of the metal do not react like the oxide and sulphide of copper. If air is blown through after the iron has oxidised the nickel will also oxidise and form a slag. The operation is stopped as soon as the iron has disappeared; the sulphide is poured out and pulverised and roasted to oxide. The oxide is reduced with charcoal in a regenerative furnace.

In the metallurgy of the metal at Dapto it may prove more economical to pass the sulphur dioxide escaping from the roasters over lime and form calcium sulphite rather than import gypsum. The smelters in course of erection at the date of my visit were similar to those used for copper ores. I am indebted to Mr. Hoyt and Mr. Rogers for the many details kindly supplied in connection with the works. The following details on assaying were courteously supplied on a former occasion by Mr. W. E. Thomas, chemist in charge:—

Smelter Slags for Silica.—Weigh out 0.5grm. of the finely powdered slag; transfer to a small evaporating dish; moisten with water; add 15 to 20 drops strong HCl, and stir with a glass rod until gelatinous. Dry for a few minutes; add a few drops of strong nitric acid; evaporate to dryness. Heat to 110deg. C. to render silica insoluble. Cool; take up with dilute HCl. Filter; wash residue free from HCl; dry, ignite, and weigh as SiO_2.

Lime.—Take filtrate from SiO_2 estimation, and heat to boiling. Add NH_4OH until alkaline. Boil and add hot oxalic acid solution until all the lime is precipitated as oxalate. Allow to stand for a few minutes, and decant through a filter. Wash precipitate on filter with hot water to remove all oxalic acid. Dissolve precipitate in dilute H_2SO_4; dilute slightly; heat nearly to boiling, and then titrate with standard $KMnO_4$.

Iron.—Treat .5grm. of slag in same way as for SiO_2. Evaporate off nitric; add HCl. Reduce with $SnCl_2$, and estimate with standard $K_2Cr_2O_7$.

Manganese.—Treat .5grm. of slag as before with HCl and HNO_3. Drive off excess of acid. Add H_2SO_4, and heat until white fumes appear. Dilute; add an emulsion of ZnO to precipitate the iron. Titrate with $KMnO_4$ until the MnO_2 is precipitated, and the solution remains pink.

Alumina.—Treat .5grm. of slag as for SiO_2. After taking up with HCl, take 5grm. NH_4Cl, and boil. Add excess of NH_4OH and boil. Decant, and wash by decantation then on filter. Dissolve precipitate in HCl, and reprecipitate with NH_4OH in excess. Dry, ignite, and weigh the mixed oxides Fe_2O_3 and Al_2O_3. Deduct the Fe_2O_3 found; the difference is Al_2O_3.

Zinc.—Weigh 1grm. slag finely ground in a large evaporating dish.

Stir well until gelatinous. Add a few drops of strong HNO_3 with $KClO_3$ dissolved, and dry, but avoid baking. Cool; add 5grm. NH_4Cl 15c.c. NH_4OH and 20c.c. H_2O. Boil for a few minutes; filter; make filtrate faintly acid with HCl. Place strip of lead foil in solution, and boil for some time to precipitate copper. Estimate Zn with a standard solution of $K_3F_2Cy_6$ using uranium nitrate as indicator.

Lead.—Take 1grm. of sample. Treat as in previous case with sulphuric acid. Cool; dilute with water in the dish. Wash twice with dilute H_2SO_4; then twice with water, keeping $PbSO_4$ in the dish. Add 30 per cent. solution of CH_3COONH_3 to $PbSO_4$; warm; pour through the filter paper used for decanting through. Wash well with hot water. Add a few drops of CH_3COOH. Teat nearly to boiling. Titrate with standard solution of $(NH_4)_2MoO_4$, using tannic acid as indication.

Dore Bullion.—Dip samples are taken of every two bars, and assayed for gold and silver, the gold assay being made as an ordinary parting assay, the silver by Volhard's ammonium sulphocyanide method. Assay of copper precipitate, bullion, matte, etc. The scorification assay is used for silver and gold, and a combined method in the case of mattes.

Scorification Assay.—The sample is scorified down with a large excess of lead in 10 lots of 1/10 A.T., each collecting buttons in pairs, and assaying separately the buttons and slags from each pair. The first scorification is conducted at a high temperature. This method gives the best result for gold only.

Wet Method for Gold and Silver.—Take 1 A.T. (assay ton) of copper precipitate. Dissolve in strong HNO_3 free from Cl. Evaporate and expel all free acid. Dilute to a litre. Add 20c.c. of a saturated solution of lead acitate; then 1grm. of KBr or KI, dissolved in H_2O. Stir; add 5grm. of sodium sulphate, dissolved in water. Mix well; allow to stand for 12 hours. Decant; clear liquid filter. Wash with hot water. Mix with soda, litharge, and reducing agent. Melt in pot with a nail. Scorify and cupel lead button obtained. Run down slags from scorifier, and hard part of cupel, and add these on.

Nickel Assay.—Messrs. Thomas, the chemist at the works, and Derrick, chief assayer, tried many processes for a speedy and reliable nickel assay. Gravimetric processes took too much time; electrolytic methods were unsuitable, so a process based on Moore's volumetric estimation has been adopted. This depends on the fact that if an ammoniacal solution of nickel is treated with a solution of potassium cyanide, then a double cyanide of nickel and potassium will form. As an indicator a minute quantity of silver iodide is suspended in the solution. Silver iodide is insoluble in ammonia, and only dissolves in potassium cyanide solutions when the double nickel potassium cyanide has formed, so that the clearing of the solution indicates the end of the reaction. The amount of potassium cyanide necessary for clearing the solution may be exactly estimated by adding a standard solution of silver nitrate from another burette until the solution is turbid again. The process may be simplified by adding a small quantity of silver cyanide to the potassium cyanide solution and running this into an ammoniacal solution of nickel containing potassium iodide. The solution becomes more and more turbid until the potassium nickel cyanide is all formed, after which it gradually clarifies. By standardising with a pure nickel salt and with pure silver nitrate the value of the cyanide solution, as compared with nickel, may be accurately obtained. The method has proved satisfactory on New Caledonian ores, except when a large amount of iron is present. Of course, with copper, zinc, manganese, or such metals as behave similarly to nickel in ammoniacal solutions the method would be useless.

English and Australian Copper Co.

The English and Australian Copper Company was formed in 1851 to take over the business of the Patent Copper Company. Smelting works were erected at Burra, South Australia, and ores from the Burra Burra mine were treated. In those days railways were not thought of, and all stores and fuel had to be conveyed by teams. As much as £7000 per annum was paid for fodder alone. The cartage at first was nearly all one way; in order to equalise matters, additional smelters were erected at Port Adelaide, and some of the ore was sent down as back loading. In 1888, the supply of copper ores falling low, silver and lead furnaces were erected, and business was continued in these lines until 1894. The values of lead and silver having fallen considerably about this time, and the rapidity with which new smelting methods were introduced, caused the company to again revert to the smelting and refining of copper ores only. Copper smelting works were erected at Waratah, Newcastle, New South Wales, in 1871, and since that time these have been continuously in operation. From 1877 to 1884, large quantities of copper ore from New Caledonia were treated.

These works are interesting in that the ore is dealt with by the old process of smelting in reverberatory furnaces, and that almost absolutely pure copper is turned out. At one time copper ores containing gold and silver were dealt in, but now only the purest ores of copper are purchased. It is found that ores containing antimony and arsenic are not only difficult to refine, but owing to fouling the furnaces they spoil subsequent charges of ores fed in. The ingots turned out assay from 99.8 to 99.9 pure copper. This is purchased by the British Government. The bulk of the ores come from Wallaroo, South Australia, but New South Wales and Queensland furnish good supplies. The ore arrives at the works in lumps, and consists of sulphides, oxides, and carbonates, mixed with the usual siliceous or basic gangue. Parcels are sampled by coning and quartering, and the final assay samples obtained in the usual way. Three samples are taken—one to be retained by the vendor, one by the company, and the third sealed and kept as a reference sample. The company pays for the copper in the ore as indicated by fire assay. Oxidised ores are run down with soda, borax and some reducing agent, such as argol; the slags are cleaned, and any copper they contain added to that first produced. The whole of the copper is then melted into one button, covered with refining flux and poured. Pyritic ores are fluxed, and fused for regulus, or double sulphide of iron and copper. The slag is detached, and the regulus roasted. The roasted material is fluxed, the copper reduced as from an oxidised ore. The copper buttons obtained in either case are flattened, and then beaten out until as thin as an ordinary visiting card. If the slightest crack should show at the edge, the copper is not pure, and the ore is, as a rule, rejected. It is mainly due to the information thus obtained that the dry method of assaying is adopted. There is always a loss of copper in the operation, so that when a wet method is used by the vendor, one per cent. is deducted from a 12 per cent. ore or under, one and a-quarter per cent. from ores running from 12 to 20 per cent., and one and a-half per cent. from ores running from 20 to 35 per cent. In case of dispute, the reference sample is submitted to an independent assayer, and payment is made on the middle assay of the three. The price fixed for copper ore is one shilling per unit less than the latest telegraphed price for standard copper in London. When the ore does not contain arsenic, antimony, bismuth, and other impurities, and the company considers the copper contents of sufficiently good quality to bear their PCC brand, a special allowance of 6d. per unit (£2 10s. per ton of copper) will be made.

The charge made for smelting varies, but does not exceed 40s. per ton of 21cwt. in any case. The ores are blended for the preliminary treatment, which consists of melting the mixed ores in a reverberatory furnace, so as to produce a matte and a siliceous slag. The charge of mixed ores and slag weighs about 3 tons, and carries about 23 per cent. of copper. The furnace has a sectional area of 17 feet by 11 feet, the height being 2 feet 6 inches. The fireplace, in which coal is burnt, is 6 feet square. As soon as all has melted down, the slag is run off or raked off, leaving the coarse metal below. The oxide of iron present combines with silica, while the sulphur unites with the copper, any excess of sulphur uniting with iron. The tap hole for slag is at the end of the furnace opposite the fireplace. The slag, when tapped, runs into gutters made in the sand, falling a few inches each time. Any matte or prills collect in the upper gutters. The heat is maintained by a few shovelfulls of slack coal being thrown in prior to tapping. As soon as the slag is withdrawn, a second charge is fed in, and the matte is not withdrawn until after the second lot of slag has been run off. The slags are all examined, and as a rule the upper lots are returned to the furnace. The matte as tapped out runs 45 per cent. of copper, or has been doubled in value by this system of concentration. The advantage of this method is that slags may be formed having free silica present. These slags contain about 44 per cent. of combined silica, the balance being mainly oxide of iron and a small quantity of alumina. Practically no limestone is added, the total quantity used only amounting to 20 tons per annum. The matte produced is run in hot to the second furnace, which is similar in type to the first. Here the balance of the iron is slagged off with the silica, which is still present. This is skimmed off periodically, and returned to the first furnace. The copper sulphide present then oxidises, and the instant it does so the oxide reacts with some unaltered sulphide, producing metallic copper, with the evolution of sulphur dioxide. The reactions occurring might be represented thus:—

Roasting and Slagging—

$$FeS + SiO_2 + 3O = FeSiO_3 + SO_2$$

Roasting of Copper Sulphide—

$$Cu_2S + 3O = Cu_2O + SO_2$$
$$Cu_2S + 2O_2 = 2CuO + SO_2$$

Reduction of Copper—

$$Cu_2S + 2CuO = 4Cu + SO_2$$
$$Cu_2S + 2Cu_2O = 6Cu + SO_2$$
$$Cu_2S + CuSO_4 = 3Cu + 2SO_2$$

In all cases, the oxide and sulphide react at a high temperature, giving metallic copper.

These actions are allowed to go on for 24 hours, during which time upwards of 5 tons of matte will have been treated. The slags are continuously drawn off, air is admitted, and the heat applied. The product as tapped from this furnace runs 92 per cent. of copper, the balance being mainly unaltered sulphides of copper and a little iron. The sulphur amounts to from $1\frac{1}{2}$ to 2 per cent.

The impure copper produced goes to the refinery, or a furnace of somewhat the same type as the others, but having a capacity of 10 tons. Here the crude metal is melted slowly, and an oxidising flame kept over the surface for from 6 to 7 hours. The whole of the sulphur is got rid of, and the iron and part of the copper oxidise and form a slag. This is raked off and sent back to the second furnace. The copper is tested periodically, and when the sulphur is eliminated it is poled. The base metals appear to be oxidised first, the slag being formed at the expense of the lining. As soon as the slag ceases to form, the sulphide and oxide react on each other, with an evolution of sulphur dioxide, which throws up a shower of copper globules; frothing and foaming also go on, due to the same cause. If the top becomes pasty before the bottom, the sulphur dioxide will cause the mass to swell up. Similarly if poured into a mould, a ridge will form along the bar. The copper still contains sulphur dioxide, as well as cuprous oxide, dissolved in it. In order to eliminate these, the bath of copper is poled, or a green pole is plunged into the molten metal. Gases, consisting of steam, hydrogen, carbon monoxide, and hydrocarbons are given off freely. These sweep out whatever sulphur dioxide still remains dissolved, in the same way as one gas will expel another from a solution. As soon as such gases are expelled, the sample, on being poured out, will have a level, smooth surface after solidifying. Poling is carried on until the cuprous oxide is reduced. In this case, the operation proceeds until a sample of the copper, weighing about a pound, is cast into a shallow cylindrical mould, about 3 inches in diameter, a flat lip being left on one side. This is hammered out into a sheet about a foot in diameter. If it shows the slightest cracks on the edges, the refining is not complete. If poling is kept up too long, the copper is spoilt, for on being poured into a mould a depression will form along the centre of the surface of the bar on cooling. Such copper is brittle, and must be oxidised again to bring it to the right pitch. This action is said to be due to the reduction of certain salts in the copper. The salts themselves, such as antimoniate and arseniate of bismuth or lead, have no injurious effect on the copper, but the elements of which they are composed have.

There are two furnaces used for calcining or roasting. These are of the same type as the others; 8 reducing and roasting furnaces and 2 refineries. Copper, to the amount of 1150 tons per annum, is produced, while 1000 tons of coal per month are burnt, the cost of the coal being 7s. 6d. per ton. It is obvious that this method of smelting copper ores can only be carried on successfully where the ores are fairly rich, where fuel is cheap, and where there is careful management. The first condition is fixed by only purchasing rich and pure ores, the second by being in the midst of a rich coal-producing district, and the third by having at the head of affairs a manager whose connection with the company dates back to 1851. Mr. F. W. Cast, the local manager, has thus been 53 years assistant and manager, so that he has a thorough grasp of all the details of this branch of the work. Mr. F. Cast, jun., his son, is assistant manager, and I am indebted to him for his courtesy in showing me over the works, and supplying the information necessary for this article.

SPECIAL FEATURES.

Choice of Gold Mining Machinery.

When a mine has been prospected sufficiently to warrant the erection of machinery, the question as to what class of machinery is most suitable should be considered. A great deal of experience is needed to determine this in some cases. If the ore is free milling and the percentage of concentrates low, then adopt the simplest methods—in this case wet crushing and amalgamation. Many experiments have been tried with various wet crushers, but so far there is no all-round machine which will equal the ordinary stamper battery. It is a mistake to get light stamps. Let them be 1000lb. or over, and all parts proportionately heavy. The argument in favor of light machinery for carting should not be made too much of. The carting may cost a few pounds extra, but the saving on the first month's run will more than make this up. When away from a town or facilities for obtaining skilled labor, everything should be of the simplest and strongest. For instance, where mine or any hard water has to be used do not get multitubular boilers, for the extra consumption of fuel will cost less than stoppages and repairs for tubes. For similar reasons get belt-driven in preference to spur-gearing.

Assuming there is about 2 or 3 per cent. of concentrates in the ore, it will generally be advisable to have a concentrator, preferably one of the Wilfley type. There are others as good as this, but as a rule they are more delicate in construction, and require much more care. The Wilfley will remove most of the coarse concentrates, any free coarse gold or amalgam that may have escaped. The sands, as a rule, are very poor after passing over the Wilfley, but the slimes, which will amount to from 40 to 50 per cent. of the total material, may carry high values. Very little gold or mineral from these is recovered on such a concentrator as the Wilfley. The sizing of such slimes by spitzkasten and their subsequent concentration is not advisable. Some of their values might be obtained by simply running them over long, slightly inclined canvas strakes. In all cases, it is inadvisable to run either sand or slimes to waste, for although they may be unprofitable to-day, to-morrow they may be a valuable asset.

If the gold is coarse and easily amalgamated, it is better to pay special attention to this branch of work rather than go in for cyaniding. In many cases, if amalgamation methods are properly carried out, the tailings are too poor to cyanide, while if the former operation is carelessly done, then the tailings may be rich enough to cyanide, but nothing like the same profit can be made. When the gold is fine, it is not possible to get the whole of it by amalgamation. Even when in quartz it is not uncommon to have tailings running up to an ounce per ton after passing over concentrators. The gold in this case is encased by the grains of sand. This may be readily proved by first panning off a sample, then grinding finely, and panning off again. When clear quartz runs up to

10oz. per ton, and carries fine gold, the tails should always be kept separate. By putting on finer screens, matters are not improved, for beyond a certain limit as much is transferred to the slimes as is liberated from the sand. Even by pan amalgamation with fine grinding, only a comparatively low extraction can be obtained. By grinding and chlorinating or cyaniding an almost perfect solution of gold may be effected.

One of the most troublesome materials to win gold from by ordinary amalgamation is ironstone. This is specially true when the gold is fine and the gangue is clayey. In most instances from 30 to 40 per cent. of the gold may be amalgamated; very little is caught on the concentrators, but a large portion passes into the sands and slimes. This material is best treated by calcining it; then cyaniding or chlorinating. There is a common belief that by slow running more gold may be obtained by amalgamation. By testing many parcels of ore with a 5-head battery, and making regular assays of the tails, the results always came out very nearly the same, generally slightly in favor of fast running. The ore above water level is generally called free milling, one of those misleading terms based on the assumption that because the ore is oxidised, the gold may be readily extracted by amalgamation, while the ore containing sulphides and other unoxidised minerals is placed in another class. As a matter of fact, a great many sulphide-bearing auriferous ores will yield their gold more readily to amalgamation processes than will some of the so-called free milling. In most cases the gold present is in quartz or in small pieces mixed with metallic minerals. On crushing, the gold, if freed and not too fine, will amalgamate. That which escapes is free gold, and will be caught on concentrators with a large amount of barren pyrites. The pyrites may be altogether free from gold, and yet the mixture may assay 3 or 4oz. of gold per ton. In this case good extractions may be obtained by simply cyaniding the material. Pyrites, arsenical pyrites, blende and galena may sometimes in this way be treated, and excellent returns obtained at a low cost. Experiments should always be made to test this before expensive works are erected. In many cases, the gold in pyritic ore is very fine, and evenly disseminated through certain minerals. For instance, if pyrites are present only, all the gold may be in or on the surface of these, and none in the stone; if arsenical pyrites are present there will generally be more in the arsenical; while if blende or zinc sulphide occurs, the gold clings to it by some process of natural selection, the silver going to the galena. In this case, the simplest method is to grind the materials finely, thereby liberating the enclosed gold; then treat with cyanide. Unless compounds of gold, such as tellurides, are present, extractions may be obtained which are in proportion to the fineness of grinding. The trouble, as a rule, with this method is not to get the gold into solution, but to separate the whole of the solution from the slimes. Even after very fine grinding a large proportion of such concentrates may be treated in vats, for the product, though fine, is still leachable, but the true floating slimes from the battery, and those from the overflow of the grinders used for the concentrates, are impenetrable. It may be indicated at this stage that such slimes may be treated by first roasting them raw with cyanide solutions, and filter pressing, or by agitating them with dilute cyanide solutions, and then decanting them. In all cases if slimes are worth treating provision should be made for so doing when the crushing plant is erected. When a reef carries a high percentage of pyrites rich in gold, it may be more economical to concentrate the pyritic material, roast it, and then treat the roasted stuff with either chlorine or cyanide solutions. It may prove just as cheap to roast some pyrites as to grind them sufficiently fine for a good extraction. The subsequent cost is less on account of the possibility of direct treatment in vats. The objections to roasting certain classes of ore is that if not properly performed a large amount of chlorine or cyanide is used up, and bad extractions obtained, while even in some cases if the roasting has been carried on perfectly, reactions or changes occur whereby some of the gold becomes locked up, and no solvent, such as chlorine or cyanide, will remove it. Curiously enough, even after finely grinding the roasted material and agitating it with cyanide or chlorine solutions, such gold from a particular mine would not dissolve. Yet if the original ore were slimed and cyanided almost the whole of the gold could be removed. No hard and fast rule can be laid down; each case must be decided after careful experimental work.

When the metallic mineral contents of a lode amounts to 30 per cent. or more, and the gold is associated with this mineral, then bulk treatment is advisable. The slimes produced, amounting as a rule to more than 40 per cent. of the material, would have the same or higher values than the original ore. The concentrates would need dilution with sand for an effective roast, while the small quantity of sand discarded would also carry a percentage of gold which would be lost, or additional expense incurred by cyaniding it separately. If ore is to be treated in bulk it should be dry crushed. Wet crushing is slightly cheaper, but a separation of sand, slimes and mineral is always effected. With dry crushing all materials are evenly mixed, which leads to uniform work in roasting, and also in the even percolation of solutions when the roasted material is placed in vats. By such a method of treatment the whole of the fine gold can be dissolved out, and it is questionable whether such material, after all the fine gold has been removed, could not be more effectively treated by amalgamation over copper plates, thence over concentrators to remove any coarse gold, than by removing the coarse gold first, and dealing with the slimes afterwards. The slime trouble would certainly not exist, while any coarse sands could be saved by classifiers if gold was still locked up, and any fused particles or specks of coarse gold would be caught on the concentrators or amalgamated. In case an ore, even when heavily mineralised, contains a notable percentage of galena, it may not be advisable to dry crush and roast the whole of the material. In this case, it will be better to crush through coarse screens in an ordinary stamper battery, and remove the strip of galena carrying gold, which appears on the edge of the Wilfley. A wider strip than necessary may be taken off at first, since the galena band moves laterally as the contents fed on to the table vary. By placing a dividing line so that galena always is caught a good deal of other concentrated minerals come over also. This material should be redressed by feeding it on to a separate Wilfley evenly. In this way, a clean strip of galena may be taken off, which contains a large amount of free gold. This may be ground in Berdan pans, and the overflow, still containing high values in gold, sold to the smelters. The lead present will usually defray all costs, the gold and silver being profit. In any mine where galena is abundant this method of treatment commends itself. The slimes resulting from the wet crushing should be cyanided at the same time as produced. This may easily be done by first thickening the pulp in spitzkasten, agitating and cyaniding. If poor, they should be treated by decantation; if rich, by filter pressing.

In connection with this matter it should be remarked that the Wilfley table, the Luhrig vanner, and such machines which separate minerals into bands or strips in accordance with their specific gravities, are open to objection when such material requires to be roasted. For instance, bands of galena, arsenical pyrites, pyrites and zinc blende form on the table or vanner, and drop into boxes as separate products. These never become properly mixed, and the particles often become lumped together. Galena, as a rule, accompanies the gold, and hence is the richest of the concentrates. When roasted a portion of the lead sulphide becomes lead sulphate, and also lead oxide. These react with the core of unaltered galena, producing some metallic lead.

$$PbS + PbSO_4 = 2Pb + 2SO_2,$$
$$\text{and } PbS + 2PbO = 3Pb + SO_2.$$

The lead so produced gathers up adjoining particles of gold. When this passes through the furnace it may come out as metallic lead, in which case the gold it contains cannot be dissolved either by cyanide solutions or even with chlorine. If the roasting is prolonged the lead oxidises, and the oxide of lead fuses and coats adjoining particles, leaving a minute sphere of gold or silver covered with a glaze. This material also is not attacked appreciably by solutions. The tendency for this effect is greatest when the gold and lead are close. If the small amount of galena present were distributed through the concentrates, and if finely ground, the effect is much less. Again, arsenic compounds lose their arsenic, and antimony compounds their antimony, and these become oxidised much more readily when sulphur is present in excess. The tendency of sulphur is to form volatile compounds with both of the elements mentioned. These compounds become oxidised immediately they reach the hot air, and are either swept away or rendered harmless. When pyrites in excess are thoroughly mixed with arsenical pyrites or arsenical compounds, the latter are amongst the easiest ores to roast. Zinc sulphide also is converted into sulphate much more easily in contact with pyrites than when separate.

An ore running a few pennyweights per ton, with a small percentage of pyrites, the bulk of the gold being in the pyrites is naturally one suited for concentration. If the concentrates produced amount to about 18 to 20 tons per week, it would be as well to treat them on the spot, provided conditions are favorable. If they amount to a much larger quantity, then, unless very near customs works, it should pay to treat them at the mine. If this quantity of pyrites is produced, and it is desired to sell to the smelters, it is better to erect a furnace and rapidly roast the concentrates. The whole of the arsenic may be expelled, and the sulphur brought down to a very low amount for a few shillings per ton. As much as 25 per cent. saving on freight and more than that on the charges per ton for smelting may be effected. There is really no reason, except in a few instances, why gold should not be produced from such material by chlorination or cyaniding at as low a cost as smelting. The main trouble is to secure men who are competent to carry on this branch of the work.

The following furnishes approximate prices for the cost of a mill for any out-of-the-way mine on the lines previously suggested:—

20 N.H.P. Cornish flue boiler for 100lb. working pressure...	£200
30 N.H.P. Cornish flue boiler for 100lb. working pressure ...	275
20 N.H.P. horizontal side valve engine, with governor	275
Compound engine, with governor complete	350
10-head iron-framed battery, 1100lb. stamps, 5-head belt driven from underneath counter shaft, with cross-keyed tappets, etc., complete	600
Copper plate tables, with 20 feet copper for each 5-head	40
30-inch Berdan pan and framing complete	20
Concentrating table, complete	125
Grinding pan, 4 feet 6 inches diameter	50

The cost of a suitable stonebreaker would amount to from £50 to £70.

An automatic ore feeder could be made for £10 complete.

In addition to this, shafting, belting, and pulleys would absorb about £120. The cost of carriage, sheds, and foundations, and erection would cost from half to as much as the plant. Roughly speaking, a 10-head battery, with appliances indicated, would cost from £1000 to £1500.

Ordinary battery practice does not need much discussion. The work is for the most part simple and mechanical. In Victoria, especially at Bendigo, crushing is cheaply done, but costs would be a good deal less than they are if stonebreakers, feeders, and more modern types of batteries were erected. For small mills it is of no use procuring extra machinery where the same number of men are employed with or without it, unless the extra machinery means a reasonable reduction of costs per ton.

No 10-head or larger battery, crushing regularly, should be without a stonebreaker, grizzly and automatic feeders. The stamps, as before indicated, should not be less than 1000lb. in weight. Whether inside amalgamation by means of copper plates is practised in the box or not is largely a matter of taste. In general it is advisable to feed mercury into the box except with heavily mineralised ores. The amount fed in should be $2\frac{1}{2}$ times the weight of gold liberated. That is, if stone is crushing 2oz. per ton, and a ton is crushed in a 5-head battery in an hour, then $2 \times 2\frac{1}{2}$, or 5oz., of mercury should be fed in per hour. If the stone runs one ounce per ton, then 5oz. mercury in two hours will suffice. An even, regular feed along the length of the box should be supplied. The state of the outside plate will generally indicate the proper amount to feed in; if this plate is sloppy, too much is passing in to the box; if dry, too little. Very little care is usually exercised in packing a box; in many cases the rubbish from the last clean up is shovelled back again, making the box foul from the start. Clean, angular pieces should be wedged between the dies, leaving room for any amalgam or gold to lodge. Amalgam will always get into every crevice in a rich crushing, and the provision made by some makers in having liners with ripples cast in them appears to be good, although I never had any experience with them. From 3 to 5 gallons of water per stamper per minute are used in ordinary practice; in this matter it is better to use about two-thirds of the total amount by supplying each stamp evenly and to apply the remainder or one-third after the sand has left the first copper plate. The pulp will not be thinned too much in the box; most of the amalgam passing through the screens will be caught on the top plate, the overflow from the box serving to carry it this far in a series of pulsations. The water to prevent the lodgment of sand on the remaining plates can be evenly supplied across the table. If plates are properly laid, well amalgamated, and attended to, wells are unnecessary.

All that need be said about concentration of gold ores is that concentrators which are well adapted for saving minerals of high specific gravity will not do corresponding work on gold ores. For instance, if the gold is all in or attached to pyrites, then the saving of the pyrites is what should be aimed at; but where the gold is excessively fine, especially when associated with clays, slates, the graphite pug so common in reefs, or even with limonites, then concentration will never give results that should be obtained by other means. The losses in the slimes are particularly heavy, and saving of gold means practically the saving of the whole of the slimes. Fortunately the modern doctrine "slime everything and extract the metals chemically" has transformed the old-time trouble into a positive advantage.

Any well designed plant of any magnitude should have provision made for slime treatment from the outset. A hydraulic separator could be used to wash the slimes from the sand; the slimes should then be thickened up in a short series of spitzkasten; the pulp drawn off at the spitzkasten will contain 45 per cent. of solid matter. This should be run into an agitator, dilute solutions of cyanide added, and stirred or agitated with air until the gold is dissolved. A small amount of lime should be fed in with the ore at the battery; this will facilitate settlement in the spitzkasten. For ordinary low-grade slimes agitation followed by decantation, after the manner practised at the South German mine, Maldon, or preferably after the manner suggested by Mr. Frank Moss, of the Kalgurli mine, would be economical and effective. The overflow from the spitzkasten, containing a very small quantity of slimes, should be returned to the battery. In most cases with our ores, if the slimes are well separated from the sands, the latter are too poor to treat. In certain rare cases, where the coarse material which escapes is as rich as the fines, then as a rule a preliminary grinding must be applied before the sands are cyanided; otherwise the extractions may be very poor. If a small quantity of rich slimes requires treatment, the simplest plan is to first dehydrate them by a rapid roast, put a layer of coarse sand in the bottom of a vat, a layer of about two feet of the slimes above it, and then a foot of coarse sand over this. Solutions may be passed upwards which will come through the upper layer of sand absolutely clear, not clogging it in the slightest. Even sun-baked slimes may be treated very often in this way, whereas they would be absolutely impervious if dumped into vats in the ordinary way. As a rule, the amount of true slimes from pan grinding is small; the bulk of the ground material may be treated by percolation.

In case pyrites require to be roasted and treated by cyanide or chlorination, the form of furnace selected is important. For small mines roasting from one to two tons per shift, the furnace made by Jaques Bros., Richmond (V.) is simple and effective. It may be driven by an oil engine, and requires but little attention. Sometimes the cost of any mechanical furnace is too great; in that case a simple hand-rabbled reverberatory is sufficient; there are many types of these, mostly bad. The furnace described in the article on Bethanga is the best for the class of work in Australia. Through the courtesy of Mr. A. H. Merrin, Queen-street, Melbourne, the following particulars of the reverberatories designed by him for the Cassilis G.M. Co., Gippsland (V.), were obtained:—Outside width, 10 feet; width of hearth, 7 feet 9 inches; spring in arch, 2½ inches; height of arch at centre, 10½ inches; pitch of first hearth, 3 feet in 26 feet; pitch of top hearth, 1 foot 9 inches in 26 feet; width of discharge hopper, 10 inches. These dimensions, together with the notes on the Bethanga furnaces, will give the essential details for the construction of such a furnace. Where bricks can be made on the ground, the total cost, independent of the stack and flues, amounts to only about £120. If the ore contains pyrites running about 20 per cent. of sulphur, about a ton per shift can be thoroughly roasted. If very heavily mineralised, about two-thirds of a ton per shift. More than double that quantity could be put through if a roast free from magnetics were not required, the ore being freed from its sulphur in less than half the time it takes to roast it thoroughly. The furnace is just as large as can be managed by a single workman. The lowness and flatness of the arch is in marked contrast to most furnaces in operation, but experience proves that it is quite high enough, and a great economy of fuel is effected. To work a furnace such as this, it should be fired slowly for a day before roasting begins. The pyrites, preferably dry and free from lumps, should be fed in at the top until about four inches deep. The temperature should be barely sufficient to distil the sulphur off. As they are moved down by rabbling, the sulphur given off will commence to burn, and while this action is going on rabbling should be carried on vigorously. The ore is shifted to the next rabbling door, a fresh lot being run on to the first hearth. After all energetic action is over, it will be found that almost the whole of the material is magnetic, partly magnetic sulphide, but mainly magnetic oxide; this material may be brought down, a door at a time, a fresh charge going in as soon as the top one is moved down. Fuel should only be put on when the material has been discharged from the last hearth. As soon as the smoky flame has ceased the sand may be brought on to the last hearth, and each lot brought down one door, the greatest time being spent near the top in simply stirring the almost raw pyrites. The sparking towards the lower doors when the ore is turned over or allowed to run from the spadelle through hot air will indicate the progress of roasting, but a simple horseshoe magnet, as recommended by me many years ago, and now extensively used, supplies the best guide to the state of the roast. The temperature should always be high enough to decompose any sulphate of iron, but in most cases only traces of this are to be found at any part of the furnace. Sulphate of zinc and sulphate of copper form readily, but they have practically no deleterious effect on chlorine solutions, and may be disregarded. If the ore has been roasted at a low temperature, to start with and a full red, but not high enough to melt gold, at the finish, and if a few odd specks only adhere to a magnet when tested on a sample of the cooled sand, the ore is roasted. Very often salt is used, in most cases uselessly, for just as good an extraction may be got without using it at all. There is also danger of some volatilisation of gold, but in most cases statements made on this score are exaggerated. Probably when lead or copper is present these may be chloridised to a certain extent by the salt crudely fed in. The salt should only be added at the lowest door a few minutes prior to discharging.

After roasting, the ore may be treated either by chlorine or cyanide. If there are no compounds capable of destroying cyanide or unduly using it up, then it should be used in preference to chlorine. Gold will be dissolved almost as speedily by dilute solutions of cyanide as by chlorine solutions of equivalent strength. If much lime or magnesia, oxide of zinc, or even finely divided clayey material of slimes be present, an inordinate amount of chlorine would be used up. In this case either a cheap acid wash, such as with sulphuric, should be given to neutralise these compounds, and then the chlorine solutions or gas may be applied. However, it is simpler to run on cyanide solutions. As to which is the best way of applying chlorine, it may be stated that if the gas is applied as recommended by Plattner, that is, passed through the slightly moistened ore, then if the gas passes perfectly through the ore, the gold will be dissolved in 48 hours. The time generally recommended—24 hours—is too short, and if all the gold does not dissolve in 48 hours, then it is of no use applying chlorine for a longer time. If pressure is used by forcing in an extra amount of gas then a more speedy solution may be obtained, but at the expense of more chlorine. After seeing various forms of barrels at work, including Mears, Newberry-Vautin, and Rothwell's, the author can positively say that no more gold is dissolved by these processes that can be extracted by the Plattner system. By only slightly damping the sand a saturated solution of chlorine is always kept in contact with the gold. The closed vat system is, on the other hand, troublesome. Gaseous chlorine has to generated and kept going. Solutions are highly charged with the gas, and give it off freely; a leak in a generator, pipe or vessel is always a great source of trouble.

It is also difficult to reduce the work to routine. The open vat system or modified Munktell's, is more easily managed, while much larger quantities of ore may be handled. In this, chloride of lime and sulphuric acid are mixed, so as to liberate about 4oz. of chlorine to the cubic foot of solution. The sulphate of lime should be allowed to settle. The plan of running the two so as to mix the two in the stem of a Y pipe, under the belief that chlorine under those conditions is nascent, is absurd. These solutions may take as much as from 10 days to a fortnight to dissolve the gold; in some cases it comes away in from three to four days; further, the top of the ore will often lose nearly all its gold when the bottom is untouched, and finally if such solutions are run through for any length of time channels tend to form, giving unequal extractions. Sulphate of lime also tends to clog up the filter-bed, which should be carefully looked to from time to time. Chlorinated water, as used at Mount Morgan, is preferable to the mixture used in the Munktell process, but it would not pay a small company to erect a plant on the same lines. In the case of the open vat system, a large amount of chlorine invariably escapes, the light destroys a great deal more, and usually a great deal in excess is run through the ore and wasted when the gold is precipitated.

The Black permanganate process has the advantage of being a clean process, absolutely free from escaping chlorine, and shows at once by the fine red color of the solutions when the ore has ceased to destroy it. There may be a claim in this case to nascent action, for the permanganic acid, as long as it is present, shows a gorgeous color; chlorine is liberated from the salt, but as soon as used up in dissolving gold a fresh portion appears to be liberated. In other words, chlorine is not instantly set free from the materials present, but in the same way as from aqua regia. From experiments on a considerable scale, I can say that if it is applied properly, just as much gold will be dissolved in the same time as with any chlorine solution process. It has the advantage also of being much cheaper than any. If an ore contains copper, gold, and silver, it may be roasted, then treated with dilute sulphuric acid, or by the permanganate process. The gold and almost the whole of the copper may be dissolved; the gold can be precipitated either on charcoal, with sulphur dioxide, or by fractional precipitation with sulphuretted hydrogen, and the copper precipitated on scrap iron. All acids should then be washed out, an alkaline solution run on, and the silver and a small quantity of the remaining gold extracted by cyanide solutions. This method was adopted successfully years ago at our school works on ten ton parcels. With other metals present other methods would be adopted; but it cannot be said that many ores are absolutely refractory. Pyrites, arseno-pyrites, and zinc blende, give no trouble worth mentioning; pyrrhotite or magnetic pyrites, copper ores, and galena are more difficult to roast, the last especially causing trouble in some instances. Antimony in small quantities, either by itself as sulphide, or with other metals, such as copper and lead, in fahl ore and bournonite, the last a rather common mineral accompanying gold, and often mistaken for galena, does not make an ore impossible to treat; but if the percentage of stibnite or such minerals is high ordinary methods fail. By mixing a small amount of stibnite with a large amount of pyrites, the gold from both may be extracted; but a simple method for recovering both antimony and gold remains to be discovered. Probably some of the electrolytic methods patented will be yet simplified sufficiently for ordinary practice.

In concluding these somewhat scrappy articles, attention should be drawn to tailings assays. The young metallurgist who starts so often hears that the gold from various classes of ore has been extracted down to a pennyweight, or even a few grains, or often only a trace. He finds that he cannot do such close work, and knows that he dare not confess it; in consequence of this, he is placed very often in a false position. In travelling round the goldfields of Australia I had excellent opportunities of taking samples of tailings from all places, and although not one assay of these has been disclosed, it may be positively stated that the only cases in which traces of gold were left in were those ores which originally carried very few pennyweights per ton. Next it may be said that many believed their tailings to be barren, and did not hesitate to say so; yet they were never tested by any proper assay; others who assayed deluded themselves by taking such small quantities that their balances faithfully recorded a trace, whereas if reasonable quantities had been taken pennyweights per ton snould have been put down. In fact, with regard to this matter, a clean sweep of most sta ements can be made. Up to 90 per cent. may be extracted by amalgamation processes from suitable ores. Concentrates, whether 10oz. to the ton or 5oz. to the ton, should be brought down to less than 4dwt. per ton by chlorination; but it will be found almost impossible to bring them to less than from 2 to 2½dwt. per ton. They may be reduced to the same values with cyanide solutions. If they are finely ground or are slimes, they may come down to slightly below one pennyweight per ton, but this is one of the rare extractions. Any tailings left with 5 pennyweights to the ton or more may be looked upon as the residues of bad metallurgical work. Of course from the best mills bad batches will occasionally be turned out, but there is something radically wrong when high tails are turned out repeatedly without the slighest effort to check the waste

Assaying and Gold Refining.

The following notes are written for a large body of enquirers who have appealed to me for some simple information on the ordinary work connected with assaying and dealing with special auriferous material.

In the first place a furnace is needed. The ordinary forge would do for a good deal of work, but it is objectionable, because it needs constant attention to the bellows, which supplies the draught, the crucibles are heated mainly on one side, and in case of graphite or plumbago crucibles are readily burnt through. Plumbago crucibles, it must be remembered, are made of about equal parts of fireclay and plumbago, and plumbago consists of exactly the same element as coke or charcoal, only it will burn much more slowly. Yet in course of time it may be wholly burnt away. Another objection to the forge fire is that the blast causes pieces of charcoal to blow about, and if these get into the pots where assaying is being done, they are highly objectionable. Gasoline furnaces and others are on the market now, and, although these are clean and useful, there is no reason why a simple assay, melting and muffle furnace should not be built in any place. The requirements of an assay-furnace are that it should have an even draught from below, and that a temperature sufficient to melt cast iron should be attainable. A simple form is shown in the section Fig. 1. This is a small furnace, only suitable for one pot. Since our bricks are 9 inches by 9 inches, it would be advisable to make one like this 9 inches square and 16 inches deep from the top down to the top of the firebars. The bricks in contact with the fire should be good dense firebricks; the others may be ordinary common bricks. Instead of having the frame door (A) down below, a couple of cast-iron bars will bridge the opening to the ash pit. Any piece of sheet iron will do to close this up, and if an adjustable opening is made will answer just as well. The firebars (F) rest on small cast-iron bars about 1 inch square (K) built into the furnace. The firebars may be about 1 inch wide on top and spaced about half an inch apart. Generally speaking, they may be lifted or pulled out and cleaned from clinker or slag. In case there may be delay or trouble in getting such bars, about ¾-inch cast-iron bars may be built in; they should be placed on the angle and spaced about ½ an inch apart. The flue for the escaping gases should be as near the top of the furnace as possible, and should be about 3 inches wide by 4½ inches high. This should lead into a chimney at least 20 feet in height. A damper should be put in this such as is shown in Fig. 2 (G). A cast-iron cover should be provided with an opening the size of the furnace. A slab of firebrick bound round with hoop iron forms an excellent cover.

A furnace for holding at least four H pots may be constructed on the same principle. This should be about 18 inches deep and 15 inches square. An escape flue of 6 inches wide by 4½ inches high will be sufficient for this.

The muffle furnace should take a muffle about 6 inches wide and 12 inches long. A section of this is shown in Fig. 2. A is the muffle, B is the furnace space around it, E is the escape flue into the stack, G is the damper, C is the ash pit, D is the ash pit door. It will be found convenient to have a good space for charcoal below the muffle. The clear space of the furnace, including the muffle, should be about 12 inches by 12 inches, and the muffle should be 4 inches above the bars. It is also a better plan to make the ash pit door so high that the firebars may be stirred, and the spaces between them readily cleared. A well fitting door should be provided.

In all cases all bends should be converted into curves, so that the escaping gas shall not have to pass round sharp angles. The flues should be so arranged to the chimney that the gases do not flow against each other. It is also very important that all the gases which go into the chimney should be sucked through the

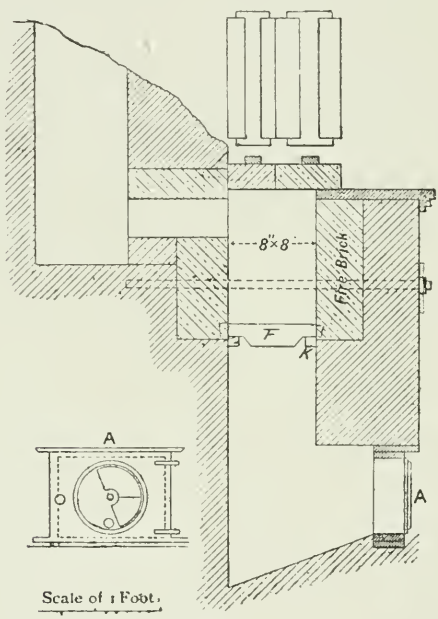


Fig. 1.

fire—that is, there should be no other opening but that from the furnaces. The draught mainly depends upon the heat of the gases in the chimney, and if cold air is sucked in the flow through the fire is at once checked. The only firebricks used are the few in contact with the flame; these should be set in fireclay, the remain. bricks should be set in a mixture of equal parts of well-puddled clay and sand. Mortar is of no use, as it will not set. Cement should be used for facing the ash pit of the melting furnace. It would be as well to encase the brickwork with light sheet iron, held in place by rods through the brickwork.

The selection of the sample for assay need not be dilated upon. Select a sample which will represent the bulk parcel. This is not an easy thing to do with many ores, but if you take precautions you need not go wrong. The method of sampling sand, whether in heaps or vats, with a sampling-rod like a cheese-tester, gives bad results. The top layers are pushed down and give proportionately more upper than lower stuff. For instance, when once sampling a vat containing a two feet layer of red sand and a 2 feet layer of white sand below, the rod brought up nearly three parts of red and only a little more than one part of white sand. You can easily test your own sampling by going over your work twice, and taking a second sample. Instead of doing duplicate assays of the one sample, sample down twice, and make one assay of each sample; this will check both sampling and assaying. If you have a heap of tailings to sample, map it out into squares. Take a sample of each square, from top to bottom, and assay each one separately. Do not trust to a few samples in this case. Sampling a heap of quartz is a more difficult matter. If it has to be accurately valued the

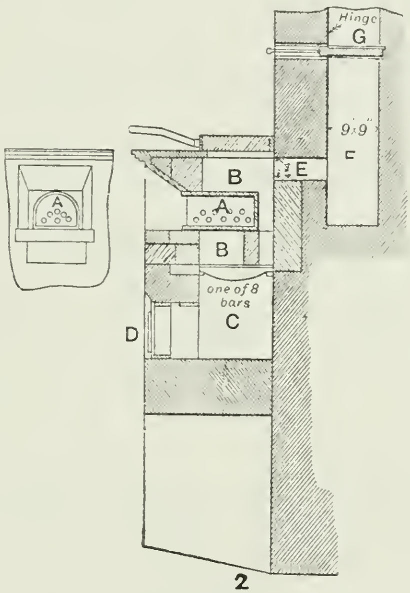


Fig. 2.

best plan would be to cut a couple of trenches right through it, and take all the material to a battery. From this the values of gold could be readily determined. If no battery is available, cut the trenches through as before, break down the stone selected to small size, cone and quarter down to a few pounds. In this case also one sample should be taken from the opposite sides of the cone, and another from the quarters at right angles to this. This to some extent checks the sampling. In any case, it means a lot of careful work. Selecting a few samples is worse than useless.

Mine sampling is a large subject, and cannot be adequately dealt with in an article of this character.

Assuming a sample of quartz is to be assayed. It may weigh from 2 to 20lb. In the latter case it is broken down to a small size, say, about the size of split peas. This is quartered, the two opposite quarters taken and kept separate from the other two. Each two quarters or one-half is again quartered, and half selected as before. This process is again repeated after crushing the selected portion more finely, if there is coarse gold present. The next subdivision gives a sample from each lot, weighing about 2½lb. This is weighed. These are kept separate, and each crushed down until it will nearly all go through an 80 mesh sieve. The coarse gold will not pass through. This coarse residue is carefully shaken out on to a large sheet of glazed paper, and the gold present estimated on the sample so taken. For instance, if the weight of the sample taken is 2.24lb. avoirdupois, and the weight of gold is one-tenth of a grain (0.10gr.) then, since 2.24lbs. give 0.1 grain, a thousand times the weight of the sample, or 2240lbs., should give a thousand times the weight of the gold, or 100 grains, or a ton, would give 4dwt. 4gr. This, of course, is independent of any fine gold present.

The method of getting this coarse gold will be described later on.

The next operation is to sample down the fine sand which passed through the sieve. This might be done by spreading the sample over a sheet of glazed paper, marking it roughly into squares

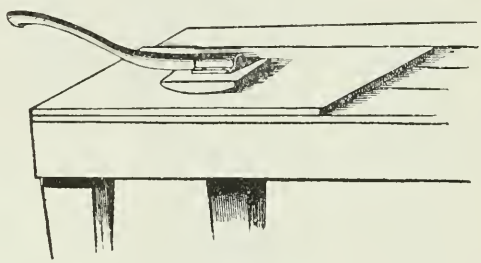


Fig. 3.

and taking a pinch out of each square with a small scoop. As a rule the gold is very fine, and tends to run through the grains of sand on to the paper below, so a better plan is to make use of the split sampler. To make such a sampler, take six strips of tinned sheet iron, about 12 inches long and 1½ inches wide. Bend them into square troughs half-an-inch deep and half-an-inch wide. Space these half-an-inch apart by soldering them to a strip at each end, which closes the troughs. These strips may be 6 inches long and the same depth as the troughs. This arrangement would give six troughs and five open spaces of the same size as the troughs. If another strip is put on the outside the numbers will be equal, and if this sampler is laid on a flat sheet of paper or sheet zinc and the powdered ore emptied on to it as much will fall into the troughs as between them. By smoothing the layers lying above the troughs down and lifting the sampler, as much will be taken by the troughs as is left on the paper. By repeating the process a small enough sample will be got for assay. Either the sample on the paper or that in the troughs may be selected.

Crushing a sample for assay in an ordinary mortar fine enough for assay is very tedious. The ordinary mortars and pestles sold by dealers are very unsuitable. A semi-circular bowl of cast-iron, close grained, and thick enough to stand the roughest usage is more suitable. A pestle provided with a long wooden handle is vastly superior to the articles generally sold. For fine grinding a slab of cast-iron about 18 inches by 24 inches and 1 inch thick, with a true surface, is almost indispensable. The grinder is a piece of cast-iron about 6 inches long, 4 inches wide, and $1\frac{1}{4}$ inches deep at the middle, and curved to the ends, where it is 7-8 inches deep. The plate may have turned-up edges and end to prevent the material going over.

The next consideration is the size of the sample to be taken for assay. This will depend on the balance used and other considerations. Supposing it is gold which is being determined then it is necessary to take a large sample when the amount of gold is small. Oertling's assay balances are guaranteed to weigh to 1-1000ths of a grain, or 0.001gr., and weighing may be done with them to .0005gr. with care. Some of Ainsworth's (American) balances weigh to one-tenth of this amount. Oertling's balances are commonly used, or others of like sensitiveness. In order to see what relation there is between the sample taken and the weighing done, suppose 1,000gr. sample is taken, and suppose the gold obtained is just weighable on the balance or 0.001gr. By simple proportion we can find the value of the ore.

As 1000gr. is to 0.001gr., so is one ton in grains to the number of grains per ton, or shortly—

1,000 : .001 :: 2,240 x 7,000 : ans.
1,000 : .011 :: 15,680,000 : ans.

There are, therefore, 15.68gr. in a ton of such material as this. Now, assuming the balance reads to .001gr., it follows that if no perceptible difference is noticed when the gold is put on the pan that the amount of gold is greater than nothing, but less than 15.68gr. per ton. This amount is recorded by an assayer, who takes such a sample with such a balance as a trace. What its true value is can only be determined by having a more sensitive balance, or taking a larger sample. Next, suppose the weight is .002gr. on the same size sample, then the value per ton will be put down as 2 x 15.68 or 31.36gr. per ton. In other words, it is greater than 15.68gr. per ton, but it less than 3 x 15.68 or 47.04gr. per ton. Its value is put down as 31.36gr. per ton, but obviously it will lie anywhere between the indications .001 and .003gr., or between 15.68 and 47.04gr., being of course nearer to the central than to the lower or higher values. If 500gr. samples are taken then anything less than the lowest readings would be called a trace. This would amount to 31.36gr. per ton, and each reading would indicate a step of 31.36gr. If only 100gr. samples are taken then anything under 156.8gr. per ton, or 6dwt., may be looked upon as a trace. If, on the other hand, 2,000gr. are taken, then one-half of 15.68, or 7.84gr., will be indicated. If 3,000gr. are taken, then one-third of 15.68gr., or 5.23gr., or less, are indicated. Or if 10,000gr. are taken, then only one grain per ton would be looked upon as a trace. Now, unfortunately, it is a common practice throughout Australia to quote very low returns of tailings on very small assay samples taken, and it is very unsafe to rely upon values which are published. Any case where it is stated that high gold values are reduced to traces by certain treatment needs most careful investigation, for in many cases misleading information is circulated.

For poor cyanide sands it is not advisable to take less than 1500gr. in each sample, the buttons obtained may be weighed separately, and then together. For very poor cyanide tailings after treatment as much as 10,000gr. may be taken by running down four assays of 2500gr. each. For such very poor material, however, assuming many assays were to be done, it would be better to obtain a more sensitive balance, and to take lesser quantities. For concentrates and richer material easily dealt with, it is not advisable to take less than 1000gr. The errors of sampling are minimised, the values obtained will be closer to the truth than if a small sample is taken. I am, of course, referring to such ores as may be easily dealt with by assay, and that the ordinary assay balance is used. The calculation of results is a very easy matter, and may be stated in simple proportion. The only thing to remember is that the grain weight is the same for all English systems; that is, a grain weight avoirdupois, troy, or apothecaries weight is the same. There are 7000gr. in one pound avoirdupois, so that there are 7000 x 2240 or 15,680,000gr. in one ton. If 1000gr. are taken and a thousandth of a grain of gold are obtained, or 0.001gr., then this means one grain in a million, or 15.68gr. in a ton. If the balance reading is 0.030, this is 30 times as great as 0.001, or 30 x 15.68 will be the amount of gold per ton, or 470gr. This is very nearly an ounce troy, so that a number like this is useful to know what the reading on a balance indicates approximately. If 500gr. be taken every thousandths of a grain in the balance must be multiplied by 2 x 15.68, or 31.36 to give the grains per ton. If 1500gr., then the multiplier is 10.45. By working out the value for different weights on the balance and posting it up behind the balance, the yield per ton can be put down as soon as the button is weighed. There is a simpler system which avoids nearly all calculation, in which a weight of a certain size is chosen, so that the reading on the balance of 0.010 indicates 1oz. per ton. This is commonly used by assayers, but as this is only written for beginners, there is no occasion to introduce any more than is necessary.

In the case of silver ores, when the values are much higher than gold, it is not necessary to take such large samples. If 15gr. indicates the closeness to which the gold assay is taken per ton, then 1oz. per ton of silver has about the same value. A sample of 100gr. would give results to within 156gr. per ton, and one of 50gr. to within less than 1oz. per ton. The same difficulty does not occur with this metal as with gold. The scales used for weighing out the ore need not be very sensitive if larger quantities are taken. For instance, if they are wrong to the extent of 10gr. in 1000, which would be a great deal, the error will only be one per cent. in the final assay.

Drying the Sample.—Samples may contain moisture, and since an assay is always made on dry material, it is necessary to dry the material chosen. It is also essential that only adhering water be expelled. If heated on a steam bath, only this water will be driven off; if heated to redness, water chemically combined and other constituents may be given off. By having a hot plate, or making some sand red hot, allowing the surface to cool, the sample placed in any convenient dish and stood on it, may be dried quickly. It may then be bottled, and the assay quantity selected from it.

Amalgamation Assay.—Assuming that free gold is present, an amalgamation assay can be readily made. For this, take 2400 grains of the finely pulverised material from which the coarse gold or metallics has been removed by sieving, place this in a large wedgwood or porcelain mortar, from 9 to 12 inches in diameter, add a globule of mercury about as large as a French bean, carefully add just as much water as will make a thick paste, and about 2 drops of nitric acid. Work the sample round with a pestle until the mercury is in the finest state of division through the thick mud. If too much water is added, the mercury cannot be sub-divided sufficiently. By grinding lightly with the pestle for about 10 or 15 minutes, the mercury will be evenly disseminated through the pulp. Then dilute slowly with water, stirring all the time. When in a thin paste, the mercury will run together, taking the gold with it. The slight acidulation with nitric acid causes it to amalgamate much better, and also to gather rapidly into one globule when diluted with water. When the mercury is all collected, the bulk of the pulp may be quietly floated off over the rim of the mortar. Run a gentle stream from the tap down one side of the mortar, and the sand will float over in some heavy particles, such as metallic iron from the grinding plate or mortar; minerals such as galena and pyrites hang to the mortar. After getting rid of the bulk of the light sand, the most of the water may be carefully poured off. Then add a few more drops of nitric acid, and grind again. Dilute once more, and the minerals will nearly all float out. If some still remain, extract the iron with a magnet, grind again, and float off; practically only the mercury will remain. This can then be run into a porcelain crucible, about 1½ inches in diameter, the bulk of the water poured out, and the remainder sucked up by clean blotting paper. Fill with distilled water, pour off again, and absorb superfluous moisture with blotting paper. The washing is to get rid of any compounds, such as salt, which contain chlorine. Take one part of pure nitric acid, and two parts of distilled water by volume, add this to the globule, place it to one side. The mercury will rapidly dissolve and leave the gold, containing some silver, in small brown, spongy lumps; sometimes, if there is much gold, small crystals of gold will be left. Allow all these to settle, and pour the liquid off into a bottle—this liquid can be kept for purifying mercury; next add some distilled water, containing some nitric acid, and pour off again. Then pour some pure nitric on the lumps or crystals, and warm up to boiling point. Do not let it bump, otherwise some may bump out. Pour off carefully; wash with water twice. In all these operations, it is essential that not one dark speck goes over with the liquid. This may be easily done by careful handling, and always pouring the liquid down a clear glass rod, held in the left hand, against the rim of the crucible, which is held with the right. When nearly all the liquid is out, the gold may be tapped down to one side of the crucible, the remaining liquid lying over it. By rolling up a bit of clean blotting paper, and cautiously touching the liquid, it may be drawn off almost wholly, the greatest care being taken not to touch any of the gold specks. Let the remaining liquid evaporate off quietly by placing the crucible in a warm place. When perfectly dry, the gold will not stick to the surface. Finally heat up to redness, and weigh the bullion obtained. As a rule the gold got by this method is of the same standard as that in the ore from which it was extracted.

To get the pure gold, it is necessary to dissolve the small quantity of silver still in it. This operation will be described later on. This method of getting the gold is preferable to taking the mercury globule and heating it, for there is a great tendency to overheat the globule, causing it to explode and scatter the gold all over the place. The practice of volatilising mercury is also a highly dangerous one, and should be avoided whenever possible. The amount of gold got in this way from most ores will agree almost exactly with that obtained by assay; but it is specially suitable for ordinary quartz with but little mineral, where the gold may be all liberated from the grains of sand. It will also work well with pyrites after roasting, but is not universally applicable. It will give in all cases a higher return than can ever be obtained by amalgamation processes on a large scale. By taking 2400 grains, each grain of gold obtained means 1lb. avoirdupois, or 7000 grains troy per ton. Suppose the bullion obtained weighed 0.524 grains, then the yield per ton in grains would be 0.524 x 7000, or 3668 grains, or working out to ounces, pennyweights, and grains.

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      { 6  | 3668
24   {       ----
      { 4  | 611 2 }
               ----  }-20gr
       20  | 152.3 }
             ----
                7oz. 12dwt. 20gr per ton.
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Metallics and Scorification.—The method of dealing with the metallics or the coarse gold, which did not go through the sieve, is to shake it out carefully on to a sheet of glazed paper. Rub the back of the sieve carefully with a hare's foot, so as to work back any bits of gold caught between the meshes, take a 2-inch scorifier, and warm it in the muffle; if this precaution is neglected, they will sometimes burst if heated too suddenly, through the water they have absorbed not getting away as fast as it is turned into steam. Run the metallics carefully into the scorifier, pour in 100 grains of granulated lead, mix by stirring with a glass rod sealed at one end, pour in another 100 grains of granulated lead. Place the scorifier in a hot part of the muffle. The lead will melt and absorb the gold and silver present. Next a portion of the lead will oxidise, and the oxide will melt and form a ring round the button. This will go on until the button becomes covered, or only shows a small eye. If now about 5 grains of powdered charcoal, wrapped in tissue paper, is dropped on the fluid mass, it will froth up and some lead be reduced again from the slag. This will carry down to the main button any gold or silver which may have passed into the slag. The amount carried down by this means is very small, since the lead itself contains the bulk of the precious metals, but it is a precaution generally advisable. As soon as the frothing has ceased, the contents are poured into a mould. When cold the button is freed from slag and placed in a cupel.

Cupellation.—The cupels are made by slightly moistening bone ash, and placing it in a mould. The top layer should be very fine; the bone ash is compressed by means of a piston, struck with a mallet. The small bone ash dish or cupel is carefully and slowly dried. Sometimes a trace of carbonate of potash is added to the water used for moistening the cupel, but this is not necessary. Cupels may be also made of dry bone ash by compressing them a little tighter. They are then ready for use after being warmed for a short time. If made too loosely, they will allow gold, lead, and silver to filter into them; if too firmly, they will not let litharge flow freely enough through them. Cupellation depends mainly on the fact that when oxide of lead melts, it is more fluid than lead, and consequently is able to penetrate a porous material like bone ash. Metallic lead, gold, and silver do not soak in to any appreciable extent. When a lead button containing gold is placed in a cupel, and sufficiently heated, the lead will melt: then the oxygen of the air will attack it. Oxide of lead forms and is apparent by the greasy looking spots which run off from the surface of the button and soak into the hot cupel. The cupel itself becomes stained. The gold stops with the still liquid lead. In course of time, there is more gold than metallic lead left, and the button becomes brilliantly colored by the films of oxide of lead running over the surface of the molten gold and disappearing in the cupel, as before. After this play of colors, all action seems to cease. The button becomes tranquil, but after a time will brighten or "blick," and solidify. The lead will be found to have practically all disappeared, and only gold will be left. If silver is present, it will behave in the same way as the gold. If both are present, an alloy of gold and silver will be left. The temperature required for cupellation can only be learned by experience. The muffle must be hot enough to melt the button, and keep oxidation going. If the button freezes, the muffle is too cold. If too high a temperature is attained, the lead may boil and spit or volatilise in copious fumes, carrying some gold and silver with it. The button obtained from the metallics should be weighed, and since it will contain silver as well as gold, the former metal must be got rid of.

Parting.—If the button is yellow, put about three times its weight or six times its bulk of pure silver with it. Wrap both in a bit of lead foil, and place in a hot cupel. A button of gold and silver will be left. After a little practice with the blowpipe, the gold and silver may be melted direct on a piece of clean charcoal, or in a cupel. It is necessary to melt them for some time, otherwise the gold may be only partly dissolved in the silver. The cupel method recommended is safer for beginners. By having about 2½ parts or more of silver by weight to one of gold, practically all the silver may be dissolved out with nitric acid. If there is only one of silver to one of gold, the acid will only superficially attack the button; if too much silver is present, the gold will be left so finely divided when the acid has dissolved the silver out that there is danger of losing some when the liquids are poured out. Strong nitric acid, as ordinarily sold, contains 68 per cent. of pure nitric and 32 per cent. water. Its specific gravity is 1.414. This acid acts too energetically on a silver-gold button. By mixing two volumes of distilled water with one volume of this acid, the strength will be about 28 per cent., and the specific gravity of the diluted acid about 1.17. This is strong enough to attack the bead if more than 2½ times the weight of silver to gold is present. After weighing the button, it should be put in a small, perfectly clean porcelain crucible, about half an inch in diameter, some distilled water poured over it, and then decanted off again down a glass rod. The dilute nitric acid then poured on it. The button will at once be attacked. Place the crucible on a hot iron plate, or on an asbestos mat, and heat nearly to boiling. In two or three minutes, the liquid will lose its brown color, and most of the silver will be dissolved. If any white flakes show through the liquid, or if it turns cloudy, either the acid or the water is impure, and the results will be bad. The flakes or cloudiness is due to chloride of silver, which should never form. Even rain water in most districts will not be pure enough. Next pour out the liquid without losing a trace of the gold, then add stronger acid made by taking two volumes of acid and one volume water. This acid will be about 50 per cent. pure nitric, and have a specific gravity of 1.32. The liquid is raised to near its boiling point; allow to stand at the temperature for about five minutes, care being taken that it does not bump. The solution may then be decanted off; the crucible filled with hot distilled water three times, and the liquid poured off each time. It is finally filled with distilled water, allowed to stand on a hot plate for a few minutes and then poured off. The drop of water remaining may be quietly evaporated off, or partly removed by careful treatment with blotting paper. When the brown powder or spongy mass of gold is dry, it may be shaken to one spot in the crucible, and then heated to redness over a clear flame. If the washing has been properly done, there will be no dark stain round the gold; and the gold will be quite loose in the crucible. The heating can be done by placing the crucible in a pipeclay triangle, resting on a tripod. A gas or spirit lamp flame is used. If the sample is not rich in gold, and if more than three times the amount of silver was present, then treatment with an acid of the strength first-mentioned is sufficient. If pure distilled water cannot be readily obtained, then the gold must be washed with dilute ammonia water to dissolve out any chloride of silver. This, however, is not advisable. If the nitric acid on hand contains chlorine, then a few drops of a solution of silver nitrate should be added to it to precipitate the chlorine as silver chloride. This should settle to the bottom of the bottle after vigorously shaking it, and the clear acid, containing a trace of silver in solution, employed for parting. This operation is one of the simplest and yet one of the most important in assaying, and the beginner should take smaller and smaller weights of pure gold, fuse silver with each lot, and then part to see whether he can recover the whole of the gold. With a little care and practice, very accurate work may be done.

Crucible Assay.—The next operation, and if there were no metallics present, the previously described work would not be necessary, is to collect the gold in the material which has passed through the sieve. The object of this assay is to melt down the whole of the materials added, and to cause the gold and silver present to unite with lead. Suppose almost clean quartz sand is being dealt with, it has been dried and passed through an 80 sieve, and the metallics dealt with separately. By mixing sufficient carbonate of soda with sand and applying heat, the mixture will fuse. About one and a-half parts of carbonate of soda are required for a fusible slag in an ordinary assay furnace. ere is often confusion as to which form of soda to use, the crystallised variety, the powdered form, or the bicarbonate. As a matter of fact, these all become the same substance when heated to redness. The crystal variety, or washing soda, is unsuitable on account of the large amount of water it contains. This causes the substance to melt readily, and boil over, if heated suddenly. It may, however, be used if it is heated first, and the dry powder taken. The anhydrous carbonate of soda, or "soda ash," is the best substance to use. The only trouble with it is that it is very often lumpy, and requires pulverisation. The commonest material, bicarbonate of soda, or baking soda, is generally fairly pure, and in the form of a dry powder, so that it is often used in place of the carbonate. On heating it gives off water and carbonic acid, and becomes ordinary anhydrous carbonate of soda, or soda ash. The main objection to this material is that it gives off nearly 40 per cent. of its weight as gas, and the outrush of this gas carries away some of the fine powder from the crucible before melting starts at all. If bicarbonate of soda is used instead of carbonate, 60 per cent. has to be added on, so as to get the same flux. For instance, if 1000 grains of carbonate of soda are required, then 1600 grains of the bicarbonate will be necessary. Metallic lead is not put in the crucible to collect the gold, because it would melt too soon, and run to the bottom; it would also be difficult to get it fine enough. A compound such as oxide of lead is more suitable. Either red lead or litharge will do equally well. The litharge should be powdered. Litharge or oxide of lead will melt freely, and in presence of silica or quartz, will at once attack it and form a fluid silicate or slag. If, however, there is charcoal present, the charcoal will combine with the oxygen in the lead oxide, and allow the lead to go free. By experiment, it is found that one part by weight of ordinary wood charcoal will reduce about 24 times its weight of lead. It does not matter how much litharge is present; when the charcoal is all used up, no more lead will separate from the litharge. A button of lead of any size may, therefore, be obtained by remembering that 4 grains of charcoal will give 100 grains of lead; if a button weighing 400 grains is desired, then 16 grains of charcoal must be added. As soon as the lead is liberated by the action of the charcoal in the hot pot, it seizes upon the fine particles of gold and alloys with them; the lead particles run together into globules, and if the slag is made fluid enough, the lead will sink to one button at the bottom of the pot. In order to get good results, the charcoal litharge and sand should be rubbed together and mixed well in a wedgewood mortar from 9 to 12 inches in diameter. The charge can be 1000 grains sand, 500 grains litharge, and 20 grains charcoal. These are intimately mixed, all being in the finest state of division. Then 1200 grains of soda may be stirred in, and mixed, and the whole lot emptied on a sheet of glazed paper, and run into a G fireclay crucible. The mortar may be rinsed out with 300 grains more soda, and this may be placed on top. A covering of ground salt, one quarter of an inch thick, or a layer of borax, may be placed on top. Neither of the last two materials are necessary in this case. The crucible may now be placed in the fire. If charcoal fuel is used, the pot should be put on a piece of brick to prevent it sinking down to the firebars, when the charcoal burns down; if coke is used, it may be placed in the fire so that its base is about 3 inches from the top of the bars. The fuel should now be built round it in small pieces, care being taken to let none drop into the pot, until the layer at the sides is well above the top of the pot. As the fire warms up the soda and sand unite, and frit together, and carbonic acid is given off quietly. If heated too suddenly, the outside melts and encloses a ball of powder on the inside, with the result large bubbles carry the material to the top, and sometimes over the top of the pot. While the sand is dissolving in the soda, bubbles are given off, but as soon as action ceases, the mass settles down into tranquil fusion. The pot at this stage should be very hot, so that the slag will not adhere to the side, and so that the molten mass may be in active circulation, and transfer any gold it still contains to the lead button below. A strip of hoop iron or iron wire may be added, so as to reduce any lead held chemically by the slag. In about half an hour from the time of placing it in the fire, the mass should be in perfect fusion. The iron may be taken out, a circular motion may be given to the pot to shake down any lead globules from the side, and the whole lot poured steadily and evenly into a conical mould. When cool the lead button may be broken off from the slag. It should weigh from 350 to 400 grains. It is cleansed from slag, and put in a cupel, whose weight should be considerably greater than that of the button. A small button of gold and silver will remain after cupellation. This button is dealt with as before, and the pure gold recovered.

The Assay of Materials Other Than Quartz.—The commonest material in auriferous rock apart from quartz is ironstone. If this is fairly pure, it will not melt with soda, as both these substances are of the same nature, or they both at high temperature will form a slag with quartz. If about two parts by weight of ironstone and some charcoal are mixed with one part of quartz and heated, a heavy fluid slag will form. It is not advisable to form such a heavy slag in assaying, so it is thinned down by adding borax glass instead of quartz sand, or borax glass and ordinary glass. The fusion is also helped by adding some carbonate of soda. As a rule, there is silica or quartz mixed with the oxides or iron, so that, remembering that if there is much more quartz than oxide of iron, soda itself will do for fluxing, say, about a weight equal to the ore taken, while if there is much more oxide of iron than quartz or silica, take a weight of dry borax equal to the ore, and half the weight of soda. Limestone, barium sulphate, or clayey material is treated in much the same way, all requiring borax or glass to make them fluid. Fortunately, the slags formed will at once show if they have been fluxed properly. For instance, a good slag should pour well, and look like a very fluid glass. It should be capable of being drawn out in long threads, which have no lumps in them; if the slag is like glass, but thick and sticky, there is not enough soda present. On the other hand, if the slag is stony-looking, there is too little quartz or borax present. Such slags do not look glassy when broken, and do not draw out into threads well. Sometimes if limestone or allied substances is present in large quantities, these materials will rise in lumps in a soda slag. This at once shows that siliceous material or borax should have been added. It is as well to remember that borax will dissolve both quartz and oxides of the metals, so that a fluid slag can always be got with it. It is also advisable to remember that no more materials than are necessary should be added to any assay, and that the bulk should be kept as small as possible.

Charcoal Necessary.—When oxide of iron or oxide of manganese is present, it is necessary to add more charcoal, otherwise no lead button at all may be obtained. For instance, 320gr. of oxide of iron will consume about 15gr. of charcoal before the oxide can unite with the quartz to form a slag. If 1000gr. of pure ironstone were taken for assay, then about 50gr. of charcoal would be required to reduce it to the state in which it enters into combination with quartz, and another 15gr. or 20gr. to reduce the necessary lead button from the litharge. This quantity, however, is largely in excess of what is ordinarily required, and it will be found that about 30gr. will generally reduce the oxides and give a sufficiently large lead button unless in very exceptional cases. Too much charcoal should not be added, for it interferes with the fusibility of a slag, and entangles lead buttons. Other reducing agents, such as flour, may be used instead of charcoal, but these offer no advantages, and most of them on decomposing give off gases which tend to carry finely-powdered ores out of the pot.

A trial flux for ordinary clayey or ironstone ores might be for two cases:—

Containing over 50 per cent. iron oxide.	Containing much less than 50 per cent. iron oxide.
Ore, 100 gr.	Ore, 1000gr.
Soda, 500gr.	Soda, 1000gr.
Glass or fused borax, 500gr.	Glass or fused borax, 500gr.
Charcoal, 35gr.	Charcoal, 25gr.
Litharge, 500gr.	Litharge, 500gr.
Pulverised borax, cover.	Pulverised borax, cover.

The borax cover, about half an inch deep, serves to minimise loss through the gases evolved carrying away some of the finely-divided ore. The ore, charcoal, and litharge, should be intimately ground together in a large mortar, the glass, soda, or borax glass then stirred in, and the whole emptied into the pot. The pulverised borax, which need not be fused, is then used to rinse the mortar out and placed on top. The pot is placed in the fire, as previously indicated, a piece of hoop iron or wire put in, and the mass fritted, then fused. In about 30 minutes the wire may be taken out, and if in tranquil fusion the button may be poured. There is no danger in getting the pot too hot in the assay furnace described; the trouble sometimes is to get them hot enough in the time specified. It is not advisable to leave pots too long in the fire, for the best slag will thicken and become pasty.

Pyritic and Other Material.—If material rich in pyrites has to be dealt with it may be assayed in two ways—the former method is to oxidise the pyrities in the pot with nitre; the second to oxidise the pyritic material in a roasting dish, and then treat it in the same way as for oxides of iron. In this case the sample where possible should be sifted to eliminate coarse gold, but as a rule such material is obtained from concentrators, and has passed over amalgamating tables, so that a portion of the gold is amalgamated. Some of this may amalgamate with the brass mesh sieves commonly employed, so that it is as well to bear this in mind. Further, if such material is very finely ground, there is danger of loss through dusting when being stirred in the roasting operation. It is preferable to take two large samples of the moderately fine concentrates and check the results against each other rather than check assays on the same finely-ground sample. There is no trouble about assaying, but there is about sampling, and the check would be on the sampling, even more than upon the assaying. The method of oxidising with nitre in a pot is not suitable for beginners, while that of roasting is.

Roasting the Sulphide Sample.—Take a sample of about 1000 grains, put it in a roasting dish 6 inches in diameter. Take about a yard of iron wire, bend about 2½ inches at right angles. This will do as a stirring rod. Rub a little chalk or whiting over the roasting dish, and wipe it out before the pyrites are put in. Spread the pyrites over the surface. Warm gently by putting the dish in the assay furnace on top of a cold fire, with the ash pit door closed. As it warms up sulphide of arsenic, metallic arsenic and sulphur will sublime. These are yellow or red compounds; afterwards the surface will start to burn. The ore must now be carefully stirred, and stirring continued until burning ceases. If the dish rises above a dull red heat, there is a danger of the mass sintering together and sticking to the dish; if it tends to get too hot lift it out and stir it in the cold air. This operation is continued until there is no smell of sulphur fumes given off. Then place the dish back on the fire, which may now be allowed to burn up, and heat the material to full redness, stirring it occasionally. The whole operation need not take more than 30 minutes. Take it out of the fire and empty it on to the cold grinding plate. The black powder on grinding becomes a rich red. Sweep it off carefully with a hare's foot and weigh—it will be found to weigh much less—perhaps, as little as 750gr. It has lost practically all its arsenic and most of its sulphur, and has taken up oxygen from the air. This material can now be fluxed like the ironstone samples. Very good returns may also be got with the amalgamation assay previously described. If the ore contains much antimony, tin or copper, these metals cause some trouble; but if the beginner deals with the simple cases given satisfactorily, he can easily understand the methods given in many of the excellent text books on assaying.

If the lead button obtained is too large for the cupel, or if it is hard, due to containing antimony, tin, arsenic, or other metals, then it is necessary to scorify it before cupellation. It is placed in a 2-inch scorifier and placed in the muffle. A pinch of borax is placed on top, and the action allowed to go on until the button is covered. Put in about 5gr. of charcoal wrapped in tissue paper, wait until tranquil again, and pour. Break off the slag and cupel the clean button. The color of the slags, the stain on the scorifier and cupels indicates to an experienced assayer what metals are present. The button is weighed, parted as before, and the resulting gold weighed, the silver being obtained by difference. When the amount of silver in the ore is not wanted, the silver necessary for parting may be added to the crucible, the scorifier or cupel, and the button parted for gold straight away. In the case of very poor ores, it is also advisable to add the silver; this will serve to collect the minute quantity of gold which might otherwise sink into the cupel and escape observation.

All litharge contains silver, some contains very little, but it is necessary to determine the exact amount. This may be done under the same conditions as the assay is done, that is, take 1000gr. of barren quartz, flux it as before, and take to the finishing point; a minute bead of silver will be the result. This may be too small for determination, but if 1000gr. of barren quartz, 16gr. of charcoal, and 3000gr. of litharge are intimately mixed and smelted, a button carrying all the silver may be obtained. The amount of silver in the 3000gr. of litharge can thus be determined.

The whole process may be followed from the study of the subjoined specimen page, from an assay notes book:—

SILVER AND GOLD CRUCIBLE ASSAY.

No. I.

Ore marked: C 21. Bullumwaal. Mineral character: Quartz sand. Reducing power: Not determined. Charge: Weight before roasting, —; weight after roasting, —.

No. II.

Ore marked: B 84. Glen Wills. Mineral character: Quartz containing iron and arsenical pyrites. Reducing power: Not determined. Charge: Weight before roasting, 1000gr.; weight after roasting, 720gr.

Crucible Charge.	No. I.		No. II.	
Ore	1,000gr.		720gr.	
Carb. of soda	1,500gr.		500gr.	
Litharge	500gr.		500gr.	
Charcoal	20gr.		30gr.	
Borax	—		Cover	
Glass	—		200	
Argol	—		—	
Nitre	—		—	
Fluor spar	—		—	
Salt	Cover		—	
Iron	Hoop at finish		Hoop iron near finish	
In fire	32min.		30min.	
To fusion	15min.		10min.	
After fusion	17min.		20min.	
Slag color	Clear green		Manganese indicated	
Appearance	Glassy		Dark glassy	
Lead button	—		—	
Weight	415		430	
Character	Soft		Hard	
Scorification	—		—	
Weight and time—				
After 1st scorification	15min.	128	12min.	240
After 2nd scorification	—		(Antimony indicated)	
After 3rd scorification	—		—	
Cupellation	—		—	
Silver and gold	.067gr.		1.220gr.	
Silver in litharge	.001gr.		.001gr.	
Gold and silver in ore	.063gr.		1.219gr.	
Gold in ore	.063gr.		.296gr.	
Silver in ore	.003gr.		.923gr.	
Gold per ton	2oz. 1dwt. 3gr.		9oz. 13dwt. 9gr.	
Silver per ton	2dwt. 14gr.		30oz. 3dwt.	

Scorification Assays.—Another method of assay is somewhat shorter than the crucible assay, but very much smaller amounts of material must be taken. It is only suitable for gold ores where an extremely sensitive balance is available, or when the material dealt with is rich, and a result to within a few pennyweights per ton is required. In this case, the ore is placed in a scorifier and mixed with granulated lead, a small amount of borax glass is placed on top, and the scorifier put in a hot muffle. The lead oxidises, and the oxide of lead either combines with the ore or dissolves it, the gold and silver present remaining in the unoxidised lead. The scorifier is heated until the lead is covered, about 5 grains of charcoal powder are dropped in, and the button and slag poured when the bubbling has ceased. The slag is detached from the lead. The same scorifier is put back again, and the lead button returned to it until sufficiently small for the cupel. This method is applicable to almost any ore, and is specially suited for silver assays.

Quartz Scorification Assay.—Take 100 grains of quartz, stir this with 500 grains granulated lead, mix well together by stirring with a dry sealed glass rod, spread another 500 grains of granulated lead on top, place in a hot muffle. In a few minutes, the lead will oxidise, the oxide of lead combine with the silica and form a fluid ring around the button. When this ring has closed over, add the pulverised charcoal as before, and after action ceases pour. Repeat the operation until the button is small enough. If oxide of iron forms the gangue, go through exactly the same operation, only add from 10 to 15 grains of fused borax on the top of the charge. If pyrites or arsenical ores, they will take from 15 to 16 times their weight of granulated lead, that is, 100 grains would require from 1500 to 1600 grains of lead, and from 40 to 50 grains of borax glass. Antimonial ores may require more lead and more borax glass. In all cases, a perfectly fluid slag must form before the buttons are poured the first time. If it is lumpy, the assay is no good, and more lead, more borax glass, or a higher temperature must be applied. Some assays, especially those containing antimony, show a tendency to bump, particles may spit out into other assays, and spoil them. Ore containing tellurides or amalgam may behave in the same way, and particles of telluride of gold or amalgam may be projected from the scorifier. As a rule, this may be corrected by heating very slowly at first, and placing some pulverised borax—not fused—on top of the assay. The borax will spread out, and to a certain extent cover the assay.

Pyrites.—When pyrites are taken, or material of a similar description, provided no loss of gold takes place in roasting—and as a rule, none does take place except by loss of fine dust—the pyrites may be placed in the scorifier to be used, then roasted gently in the muffle, stirring carefully with a piece of bent iron wire. The semi-roasted material may be then treated with a very much lesser quantity of lead, and the work shortened materially. By combining several buttons, of course larger samples may be dealt with; but from the remarks already made, the results are not close enough in a gold assay for modern requirements where an ordinary assay balance be used.

Granulated Lead.—It is necessary now to indicate how granulated lead may be prepared. Get a stout close-fitting tobacco-box; melt some soft lead, say, about 12lb.; do not heat it much above its melting point; pour it into the centre of the wooden box, not letting it touch the edges, or it will stick to them. When it just commences to solidify, seize the box and shake it to and from you rapidly, throwing the lead against one side, then against the other. Keep this up until no more grains break up. If successful, the greater part of the lead will be found to be in fine grains. Sieve these through a 20-sieve, put the coarse back into the pot, and melt down again, and repeat until the desired quantity is obtained. Sample the whole of this down, take 2000 grains, and determine the amount of silver present. Subtract the amount present in the lead from the silver and gold in the ore, as in the crucible assay. The details of the scorification assay may be followed from the form attached.

GOLD AND SILVER SCORIFICATION ASSAY.

Ore marked—66A		86A
Mineral Character—Quartz.		Pyrites.
No. 1.		No. 2.
Charge ore	100gr.	100gr.
Test lead	1,000gr.	1,500gr.
Borax glass	—	15gr.
Litharge	—	—
Scorifiers marked IV.		V.
Remarks—Fused readily.		Fused readily.
Slag color	Yellow.	Dark brown.
Appearance	Glassy.	Dull.
Button character	Soft.	Hard.

Time and weight of pourings—

	Time.	Weight.	Time.	Weight.
	Min.	Grs.	Min.	Grs.
1 ...	15	500	20	480
2 ...	10	300	10	285
3 ...	10	150	10	130

No. of grains of charcoal added before each pour—
10gr. (1), 5gr. (2 and 3).

Cupellation—	Grains.	Grains.
Gold and silver	.064	.086
Silver in test lead	.006	.009
Gold and silver in ore	.058	.077
Gold in ore	.012	.024
Silver in ore...	.046	.053

Assay Report— Contained in a ton of 2,240lb.

	Gold per ton.	Gold per ton.
	3oz. 18dwt. 8gr.	7oz. 16dwt. 18gr.
	Silver per ton.	Silver per ton.
	15oz.	17oz. 6dwt.

To Assay a Bar of Gold—Bullion Assay.—The methods of assay given in books on the cyanide process, such as Park's, Bosqui's, Turman's, and others are so simple, provided the preceding work has been done, that it would be redundant to repeat them. The method of assaying gold bullion in order to find its value is not so generally known, and therefore needs a little explanation. Assuming that a bar has been melted down, and that it contains practically only gold and silver, and that the silver is present in small proportion only, then there is very little difficulty in taking an assay sample. If the gold is very impure, there is danger of some parts of the bar being richer or poorer than others, but with ordinary gold of fair purity this does not occur. By means of a pair of cutting pliers, take a clip of about 10gr. from diagonally opposite corners of the bar, one above and one below. If the bar is too hard, then use a small sharp cold chisel, and obtain a sample. Next weigh out roughly about 10gr., taking good weight. Place this on the assay scales, and if near the weight rub down with a stroke or two across a fine file; when very near the weight, finish off on a smooth stone. With a little practice, the weight may be got exactly in less than five minutes. Do the same with the other sample chipped off.

Take one sample, wrap it in from 80 to 120gr. of sheet lead, and put it in a hot cupel, and when finished take it out and weigh again. The button left may be put on the left pan of the balance, the ten-grain weight left on the right, and the loss of weight made up by adding weights to the button, or moving the left rider until equilibrium is restored. The weights so added indicate the oxidisable metals removed by cupellation. These may be copper, iron, lead, bismuth, or any base metals. What is left in the button is only gold and silver. The other sample should have from 24 to 25gr. of pure silver added; also, assuming no copper is present, a very small piece of copper, say, 0.10gr. The whole should be wrapped in 120gr. sheet lead, and placed in a hot cupel and cupelled. The buttons are left in the muffle some time after the colored films of lead have disappeared, say, 10 minutes, in order that the whole of the oxide of lead may separate. The buttons are taken out and allowed to cool; they should be well rounded, and have a slight depression on top. They should not be loose in the cupel, nor should they stick so firmly that a lump of bone-ash breaks away with them. If the button spits on cooling, there is either too much silver present for the gold, or the button has been cooled too quickly. A button which spits should be rejected, for some loss is almost sure to have occurred. The pinch of copper added to a great extent prevents spitting, and leaves the resulting button more malleable. The button should be seized with the pliers, and gently squeezed. It will leave the cupel; some bone-ash and litharge adheres. Squeeze it laterally with the pliers, and brush this off. It should then be hammered with a clean hammer on a clean anvil, and rolled into a strip 2½ inches long. Since many works are not possessed of rolls, it may be wrapped in several folds of clean paper, and beaten with a hammer a few times, it would then be the size of a sixpence. It should then be annealed, likewise the rolled strip, by heating to dull redness. The strip should be rolled up, keeping the underside out. This is then ready for parting. Professor Mica Smith has shown that it is not absolutely necessary either to hammer out or to roll the button, for taking it as it comes from the cupel it may be parted and almost exact returns obtained.

Parting.—Take the button, the annealed flattened piece, or the cornet, and place in a parting flask, conical shape with flat bottom, of a 3oz. to 4oz. capacity, pour in some hot distilled water, and pour out again; this is to get rid of any trace of chlorides which may be on the button. Next pour about 2oz. of dilute nitric acid, two of water to one by measure of strong acid, and boil for about 20 minutes, or for ten minutes after brown fumes have ceased to come off. Pour the solution off carefully into a vessel and keep it. Wash the bulk of the silver nitrate out by pouring in some distilled water and emptying the flask again, carefully keeping the gold in. Next take about an ounce of nitric acid, made up by taking two parts by measure of strong nitric acid and one of water. This is boiled for not less than ten minutes; violent boiling is not needed in either case, and in this case with strong acid is highly undesirable, since it may lead to bumping and projecting both acid and gold from the parting flask. Some flasks are more prone to bump when strong acids are boiled in them than others. This may be prevented by taking a small clean glass tube open at both ends and standing it in the flask.

After it has boiled the proper time the acid is poured out, and the gold left washed by adding hot distilled water and decanting it out again. When this has been done three times the flask is half-filled with distilled water, and the water boiled. This is emptied out. If the gold is in one piece, it may be slid out of the flask into a clean porcelain crucible; if any fragments have detached themselves, then fill up the flask carefully with distilled water, put a porcelain crucible over the mouth of the flask, and tip the flask upside down. The gold will all drop down into the crucible. Keep the crucible in the left hand, the flask upside down, but still full of water in the right, raise the flask carefully and gradually until its mouth is level with the rim of the crucible. Then, keeping the crucible steady, move the flask horizontally, and almost at the same time bringing it into its normal position again. It will be found that after a little practice not a drop of water will overflow from the crucible, and none need be spilt from the flask. It is advisable to try this first with any small fragment of gold, for unless the hands become trained they are apt to both move together. Very fine specks of gold may be got out of a test tube in the same way.

After pouring off the water from the crucible, the balance of it may be absorbed on clean blotting paper, and the gold dried, then strongly heated. The button or cornet will be found to shrink and assume a pure yellow color. If osmiridium was present, the bottom of the button or outside of the cornet would show black specks. If platinum were present the button would not flash on cupelling, and the nitric acid solution, instead of being colorless, would remain yellow. If a check assay is made from the same bar the buttons will agree exactly, provided ordinary precautions are taken. An assay made carefully by this method would not be out by as much as one part in a thousand, or one penny per ounce. When closer results than this are needed, all that is necessary to do is to add a piece of pure gold weighing as nearly as possible the same as the gold in the sample, alloy it with the same quantity of silver and copper, cupel it side by side with the other, part it under the same conditions, and weigh the gold obtained. It will either have gained or lost in weight: a proportionate loss or gain must be added or subtracted from the sample. The loss may be due to the absorption of some gold by the cupel, the volatilisation of some gold, or the solution to a slight extent in the nitric acid: the gain is due to the presence of a small quantity of silver still left in the parted gold. An example will illustrate:—

Weight of gold taken	10.000 grains.
Weight after cupellation	9.964 = weight of gold & silver.
Loss	0.036 = oxidisable metals.
Weight of gold	9.632
∴ Weight of silver	.332

If a check piece of pure gold were put in, and if this weighed 9.600gr., and 9.603 after parting, then the gain of weight = 0.003; this additional weight per 1000 parts is known as the surcharge, and practically the same weight requires to be subtracted from the weight of gold obtained. The assay thus corrected would be in percentage:—

Gold	96.29
Silver...	3.35
Oxidisable metals	0.36
	100.00

The silver and oxidisable metals are only approximate values.

In places where chlorine or hydrochloric acid is present in the air, care has to be taken to exclude them while the assay is in progress. If this is not possible, the chloride of silver must be dissolved out in ammonia. It is also necessary to have nitric acid free from chlorine, bromine, or iodine; the last element is often present. All these had better be got rid of by adding to the nitric acid used a small quantity, say, abou 5gr., of nitrate of silver to the bottle of parting acid used to allow any precipitate to settle, and to keep this acid for parting alone.

When the bullion is lower in gold, but silver is the main alloying metal, the method of assay is much the same. In this case enough pure silver is added to bring the proportion to one part of gold to $2\frac{1}{2}$ parts of silver. For instance, if on cupellation a button weighs 9.600 and the color is rich yellow, $2\frac{1}{2}$ x 9.6gr. should be added. If the button is nearly white between 50 and 60 per cent. of gold is present, and in this case, suppose 60 per cent. is near the correct amount, then the amount of silver present in the bead will be 9.6—6, nearly, or 3.6 grains. The amount of silver which should be present is $6 \times 2\frac{1}{2} = 15$, but there are already 3.6 grains present, so 15—3.6, or 11.4, should be added. Experience will soon enable the beginner to add almost the proper quantity by mere inspection of the cupelled button. Fortunately a little more or even a little less than the $2\frac{1}{2}$ parts mentioned will not interfere materially. If much more is added, the cornet may go to pieces, while if less than two parts of silver to one of gold be present, the button will not part. After the gold has been weighed, the amount of silver which should have been added will at once be evident. For instance, if to a gold and silver button weighing 9.500 grains, 12 grains of silver were added, and the gold obtained after parting weighed 8 grains, then, according to this assay, the original 9.5 grain button only contained 1.5 grains silver; this, added to the amount of silver put in, or 12 grains, only makes 13.5 grains of silver, or the amount of gold to that of silver is as 8 is to 13.5, an amount not sufficient for parting. It would be found that the 8 grains which was weighed as gold really contained a considerable amount of silver. Where very base bullion, such as that obtained from cyanide plants, or even from complex ores or pan grindings has to be assayed, holes should be drilled through the bar and check assays made. If tin, antimony, arsenic, or much zinc be present, the alloy may require to be scorified first; the button may be then cupelled with addition of the requisite quantity of silver. In some cases the drillings may be treated direct with nitric acid in a dish and parted like an ordinary gold assay. The brown powder, or spongy lumps resulting from the treatment should be dried, then wrapped in sheet lead and cupelled. Traces of material other than gold insoluble in nitric acid will thus be got rid of. The only way of getting a reliable assay from base bullion is by taking a dip sample, that is, after the gold has been cleaned from slag it should be re-melted, and while at a high temperature well above melting point, a small blacklead or fireclay ladle, or a small crucible attached to a handle, should be made hot, dipped into the fluid mass, which has been well stirred. This may then be poured at once, with a circular motion, into a clean enamelled bucket or similar vessel full of cold water. The grains at once chilled are solidified solutions, and have the same composition as the liquid metals from which they were dipped. These can then be assayed in the ordinary way.

Assuming the assay has been completed and the gold weighs 8.000 or 80 per cent. The value of pure gold is £4 4s. 11½d. per ounce, or almost 85s. per ounce, 80 per cent. of this would be $\frac{80}{100}$ x 85, or 68s. per ounce. The old system of quoting in carats is obsolete. In this the bullion was assumed to be divided into 24 parts. If pure gold, it was called 24 carat; if 23 parts gold and one part alloying metal, it was 23 carats. Standard gold, or the gold from which our sovereigns are made, is 22 carat, or 22 parts gold and two parts copper. The percentage of gold is $\frac{22}{24}$ x 100, or 91.66, and the value per ounce is £3 17s. 10½d. Although gold is worth £4 4s. 11½d. per ounce on the basis of the sovereign being worth £3 17s. 10½d. per ounce, yet certain charges must be made for coinage, even if the gold is absolutely pure. These may amount from 1d to 2d. per ounce. Again, if the gold is alloyed with base metals, these must be extracted, and the value is only the highest amount which can be obtained for it; in other words, the cost of removing the base metals must be deducted.

Recovery of Silver from Solutions.—The liquid at first poured off on decanting from the gold contains the silver. This should be recovered. Add a solution of salt or hydrochloric acid, and shake or stir. When the liquid becomes clear again, add some more salt. If no precipitate forms, then the silver has been all thrown down, and rendered insoluble. When the top liquor has become perfectly clear again it may be poured off. This precipitate may be added to until sufficient accumulates to run down.

The Recovery of Silver from Silver Chloride. — The silver chloride, as obtained by dissolving silver in nitric acid and precipitating the metal as chloride, is not absolutely pure. It contains traces of gold, copper, lead, and other metals. To get rid of these, after washing by decantation with hot water several times, the chloride should be thrown on to a large glass funnel stopped up at the bottom with some asbestos, which will act as a filter. Hot water should be poured on and allow to drain off until the washings are no longer acid. The chloride should then be allowed to dry. A French clay crucible should be taken and borax fused in it for some time so as to fill up the pores of the pot. If this were not done, silver chloride would rapidly soak into it. When melted, take the pot out of the fire, and when cold add the dry silver chloride and put back in the fire. The chloride will melt. Heat it up until dull red and throw a small quantity of sodium bicarbonate on top; when all action has ceased, throw some more on, and repeat this until about half an ounce is added for every 10 ounces of silver chloride present. The addition of sodium carbonate is for the purpose of throwing down a shower of metallic silver globules. Any gold or platinum present will unite with these, and when the crucible is heated to redness, will form a button of metallic silver alloyed with practically all the gold and platinum. The molten mixture in the pot is then poured out into a mould, which has been lightly black-leaded and warmed. The mass is emptied when cold and boiled with water. The lump of fused chloride of silver, with the silver button still adhering, is placed in a clean glass or porcelain vessel, some pure iron, such as charcoal iron, placed in contact, and the whole moistened with 1 in 10 sulphuric or hydrochloric acid. In the course of a few days the whole of the chloride will have become metallic silver. By breaking a lump, it will be found to be white and fibrous where reduction has gone on, and still waxy where it is not altered. The button of silver which was poured out of the pot at the same time as the chloride may now be detached, and kept separate. If the chloride has not all been reduced, pour some boiling water over the mass and pour off again several times. Add some more acid and allow the iron to remain in contact until the silver has all been reduced. If not sure about the reduction, take a small quantity out, boil it with water several times, decanting or pouring off the water each time; then treat with dilute nitric acid. If it is chloride it will not dissolve; if it is silver it will all dissolve. After washing with hot water the reduced silver should be boiled with dilute hydrochloric acid—one part of acid to 10 parts of water—and then washed again until all the acid has been removed. If now dried and heated to dull redness, the silver may be kept in this state and used for parting assays. It is tough enough to hold together, and yet may be detached into pieces of suitable size and pressed together.

It may be melted down. To do this, get a clay pot of suitable size. French clays are the best. Introduce the silver, and about half its bulk of nitre, and fuse together. As soon as the mass is in tranquil fusion, add some borax glass, and when the whole lot has melted down pour into a clean mould. If nitre alone is used, the molten silver is full of oxygen, which it gives off violently after being poured, making the bar rough and hollow on top, and projecting silver globules all over the place. When the bar has partly cooled down, the slag may be nearly all detached by plunging it in cold water and withdrawing before it has cooled. The small adhering layers may be chipped off, or the silver may be heated to redness several times and plunged under water. The simplest way, however, is to soak in dilute hydrochloric or sulphuric (1 in 20) acids, and leave for some hours. The surface of the bar may be then scrubbed, and will be made perfectly clean. The bar may then be rolled out and stamped out into discs or wads of a suitable size for bullion assays, or cut into squares with a pair of shears. Silver prepared in this way will be pure enough for all mine assay requirements. If still purer silver is required, the melted bar may be treated with dilute nitric acid, until only a small quantity remains undissolved. Pour off the clean liquid through a filter, add hydrochloric acid until all the silver is precipitated, boil the silver chloride with water, and decant until the washings are no longer acid. Then add a strong caustic potash and glucose and boil with constant stirring until the chloride is all converted into metallic silver. Wash first by decantation, and then by filtration, until the washings are no longer alkaline. Dry the powder, which is metallic silver, add a small quantity of nitre and some fused borax, and smelt down in a clay pot. The silver will be almost absolutely pure. In preparing pure silver, it is not wise to start with less than about six ounces. There will be practically the same trouble purifying one ounce as the larger quantity. Small quantities of silver are usually difficult to procure except in the form of foil, which is sold at high prices. There is no reason why dealers should not supply bar silver in small lots at little more than the market rate. Do not start to purify standard silver or old jewellery in order to get your pure silver, but with the purest sample you can buy. Some impurities are most difficult to get rid of.

The Preparation of Pure Gold as Checks in Assaying.—To prepare pure gold, which is used as a check on bullion assays, it is advisable to take about half an ounce at least of cornet gold, or gold which has been parted in the ordinary way of assays. The cornets selected should have come from fairly pure gold, preferably alluvial or clean quartz bullion. Such cornets may be obtained from almost any bullion assayer at little more than the value of the gold in them. The cornets should be dissolved in aqua regia, made by mixing three parts of strong hydrochloric acid and one part nitric acid. Do not add too much, but add it little by little, and just cover the gold. When they have all dissolved in the porcelain evaporating dish in which they were placed, evaporate most of the liquid off. Allow the dish to cool and add distilled water and pour through a double filter into a Winchester quart bottle, and if about an ounce of gold is present, half fill the bottle; add about 2 grains of potassium iodide, shake up, and allow to stand for about two days. The silver which separated on dilution will have fallen to the bottom. The top liquor is carefully siphoned or poured out, the greatest care being taken to keep the silver sediment at the bottom of the bottle. The liquid containing the gold is run through a filter paper so as to separate the chloride or iodide of silver out. Heat the liquid nearly to boiling, and add a saturated solution of oxalic acid. The whole of the gold will be precipitated; allow it to settle, and pour a litre of hot distilled water on it and decant off. Repeat this four times. Then pour on about 100 cubic centimetres of ammonia water and shake round; add about half a litre of water, allow to settle and pour out, wash with distilled water once. Then add 100cc. of pure hydrochloric acid and heat almost to boiling; fill up with fresh water, allow to settle. Repeat the washings with hot distilled water until they are no longer acid. Then empty the gold precipitate on to a funnel fitted with a filter paper. Dry and add the dried gold, about half its own bulk of nitre to a French clay pot; cover with a layer of powdered salt and fuse. Pour out the gold into a well-oiled, dry iron mould. The slag will easily wash out with water, leaving a brilliant gold bar.

To Purify Mercury.—For an amalgamation assay it is necessary to have pure mercury; a similar weight to that taken for assay should be dissolved in nitric acid to see if any gold is left; mercury, even after having been retorted, often carries small quantities of gold. To purify large quantities, such as are used in gold amalgamation, the mercury should be put in an enamelled bucket and the dross which rises to the top removed with a piece of sponge until, after stirring, the surface appears bright: it should then be squeezed through chamois leather. This will deprive it of all materials except those dissolved. The methods given in most works is to allow the mercury to flow in a thin stream into very dilute nitric acid, to sub-divide it so that the base metals present may become oxidised in contact with the air, to place pieces of iron on it and retort it. None of these are satisfactory when arsenic and antimony are dissolved. A simpler method is to place the mercury in a dry retort, dry the surface, free it from all charcoal and add the residue obtained by evaporating the solutions containing the mercury in the amalgamation assay. This consists of basic nitrate of mercury mainly; on heating it will be decomposed into mercury and oxidising gases. In addition to this material a layer of about a quarter of an inch of nitrate of soda or nitre should be sprinkled on the mercury. A better material still is sodium peroxide, of which a much lesser quantity is needed. The retort should be closed and the mercury slowly distilled out. The arsenic, antimony, and like impurities are oxidised by the compounds added, and then combine with what is left, so that instead of distilling over with the mercury as they ordinarily do, they are left behind in the retort. Very impure mercury may be readily purified in this way. It need hardly be stated that the addition of metallic sodium or sodium amalgam does not purify mercury; in fact, it only enables it to take up more impurities than the quicksilver itself could.

Errors in Retorting.—The method of retorting is so well known that it should not be necessary to write anything about it, but in this case, as well as others, some blunders are unwittingly made. The cover of the retort is generally planed up to fit almost accurately, and then clamped and tightened by means of wedges. The expansion of the different parts on heating prevents the close fit it has when cold, consequently luting is added. This is generally a plastic clay. Now, while this material is excellent when wet, and even when dry, yet when it is heated it is as porous as a flower pot, and mercury vapor will pass through it with the greatest ease. In fact, the water it contains is actually driven out of it, so that it becomes porous above the boiling point of water. A great deal of mercury is lost in Australia through ignoring this simple matter: cases could be quoted where large losses took place mainly owing to the selection of such a poor lute. A better one can be made by taking some finely ground material which will not alter on heating, and moistening it with a strong solution of silicate of soda. This will form an absolutely gas-proof joint, but there is some trouble in getting it free from the retort; to overcome this difficulty an equal bulk of pulverised common salt may be added. Probably other mixtures would do as well or better, but the almost universal practice of luting with clay has little or nothing to recommend it. The slightest back pressure, such as may be caused by dipping the mouth of the condenser pipe under water, will always lead to a large leakage through the so-called clay lute. Sometimes to counteract this the pipe is not dipped into water at all, but left exposed to the air; this practice is a dangerous one, and should not be tolerated. Those who believe that because they see liquid mercury running from the pipe to the bucket below there is no escape of mercury, need only get a piece of clean gold or a sovereign, and hold it in the air a few inches away from the mouth of the pipe to be disillusioned. A better plan is to get a piece of blanket, tie it round the end of the pipe so that an open end of blanket projects a couple of inches, have this nearly level with the surface of the water in the bucket below; the blanket allows the mercury and gases to freely pass out of the retort, but acts as a flap valve, and prevents any rush of gas or air back into the retort. An experienced millman will at once know if his quicksilver is boiling too hard; in that case, there would be danger of portion of it boiling over after the manner of water from a superheated kettle. The slower the retorting is carried on the purer will be the mercury. When only a small quantity of mercury is required, it may be distilled from a hard glass tube. The addition of a small quantity of sodium peroxide will serve to free it from most metals.

When mercury has ceased to come from a retort, it must be remembered that the body of the retort is still full of the vapor of mercury. If gold is present, it will re-unite with this on cooling; if no gold is present, it may condense on the surface of the retort when cold; as a rule, retorts are opened while still hot, and there is a danger of mercurial vapors escaping, so that great care should be taken by those exposing themselves to such poisonous gases. Mercury should always be carefully weighed both before and after retorting as a check against loss.

Smelting and Refining of Gold.—In places where the gold is fairly pure, there is little trouble in cleaning amalgam: by grinding in an iron mortar or vessel that acts for a similar purpose, the gold amalgam may be freed from pyrites, pieces of copper and other metals, for these impurities will mainly float in a bath of mercury while the amalgam sinks. It is the custom at some small mills to add all this rubbish to the retorted gold. This should not be done, for it only tends to make the gold impure. The material can be easily dealt with by itself, and there is no reason for adding it to the amalgam. It is a pleasure to see some of the mill managers and amalgamators prepare retorted gold. A few points concerning the method of retorting have already been dealt with. When amalgam is retorted, the same precautions as to heating must be followed as in retorting of mercury. There is one drawback concerning large cakes of retorted gold, and that is that they must be cut up into pieces before they are smelted. Although this operation may be conducted without appreciable loss, yet there is always a danger of small pieces of gold becoming detached, while the dividing of the retorted cake always occasions unnecessary trouble to the smelter. If hard squeezed amalgam is tied up in a cloth and quietly retorted, it will maintain its shape without going to pieces; if made up into balls suitable for a crucible, each one may be kept separate from the other by simply putting layers of paper between or wrapping in cloth. The amalgam should be slowly heated, but the temperature at the finish should be full red. There is always a small quantity of mercury retained. If the temperature of retorting has not been high enough, the retorted gold will be brittle instead of tough, and the cake will break with a crystalline instead of a hackly fracture. When impure gold is being dealt with the amalgam is much more difficult to clean. The amalgam, with gold-silver compounds, is so light that it floats on mercury, and consequently impurities cannot be skimmed off as with heavy amalgam. In this case, by grinding repeatedly and working extraneous matter out, a fairly pure amalgam may be obtained. A small quantity of some oxidising material, such as nitre, may be placed in the retort with the amalgam; in this case, however, paper or rags, or other organic matter should be excluded, and the balls of amalgam separated by small sectional compartments made from thin sheet iron. A little nitre, bichromate of potash, or other oxidising agent may be added to a slight depression made on the surface of each ball of amalgam. After the amalgam has been retorted, cooled and weighed (it may be cooled by placing it on a cold iron plate, never by wetting it), the cake should be smelted straight away. No mine should send away retorted gold, there is always danger of loss, and there is never any definiteness about the value or the weight.

Smelting Retorted Gold.—If the retorted gold is fairly pure, use a black lead pot; a little experience will soon indicate the proper size. At first too large a pot is usually selected. Anneal the pot by warming it gradually by placing it upside down on top of the fuel in a cold fire; as the fuel burns the pot will gradually warm. Morgan's Battersea crucibles are very reliable, very seldom bursting even when heated suddenly. As soon as they are dull red they are safe. Next put about an ounce of borax in the bottom of the pot, and when melted swill it round so as to glaze the sides. Take the pot and place it on some dry surface, and add all the loose pieces of gold, reserving the solid lumps. The pot may be filled with pieces of retorted gold almost to the top. Add some borax, put back in the fire, and heat; the retorted gold will continually shrink; as it does keep adding the lumps with a light tongs until all the gold is in. It will be found in all cases that mercury is given off; by holding some clean gold over the top of the pot before very strongly heated, the surface at once is amalgamated by the mercurial vapors. For this reason the escaping gases should never flow into the room or be breathed by the smelter. Borax may be added in suitable quantities. Nitre, carbonate of soda, salt, or sand should not be added; there is no necessity for them, and the first two attack the crucible, causing it to corrode in a very short time, while the last only makes the slag infusible. There is no necessity to skim off the slag, as is sometimes done. The mould into which the gold is to be poured should be heated strongly, and then allowed to cool down—not putting it in water, but by standing it on a cold iron plate until it is hot enough to be uncomfortable to handle. If it is too hot, the gold will not chill at once, and blowholes will form on the under surface of the bar. The mould should be clean and wiped out with a little clean oil. In pouring large pots asbestos suits and gloves are sometimes worn. Woollen gloves, however, will protect the hands and arms. It is a good plan when large amounts of gold have to be poured for the melter to wet his hands and face just prior to pouring. He will be able to stand a temperature for a few moments that would otherwise be unbearable. The slag runs out first; the gold should be quietly and steadily poured until the pot is empty. If the temperature was high enough the whole of the gold will have run out and the slag will have no shots of gold in it. The slags, however, should be saved. By partly cooling the bar under water and then withdrawing it while still hot the bulk of the slag will crack off. The rest may be chipped off, or wholly removed by making the bar red hot and placing it in highly dilute 1 in 20 nitric acid. If necessary, the bar may be remelted in a clean pot without any flux, and when very hot and fluid poured into a clean mould. In this case there will be no trouble about cleaning, and there is practically no loss on remelting clean gold. A very clean bar may be made by pouring the metal without any slag over it into a layer of about half an inch of whale or other animal oil. The heat given off from the burning oil is, with large melts, very trying. The grease on the bar may be burnt off or cleaned off with a little soda.

When impure gold is melted, it is practically impossible to purify it by mere melting, even with the addition of fluxes. It is certain that in a blacklead or plumbago crucible the main effect of fluxes added is to eat away the crucible. Nitre, carbonate of soda, salt, and such materials only attack the impurities in the gold to the slightest extent, while the crucible is attacked violently. When retorted gold is dirty-looking or discolored, it should be melted in a clay pot. As there is a danger of these cracking, one should be procured which is dense in grain and which will ring well, showing absence from hidden cracks. To minimise danger of loss through breakage, the pot should be stood inside the bottom of a larger blacklead one, so that should gold run out it will not get to the ashes. The spongy retorted gold, if not very dirty, may be fed into the pot with some nitre; a layer of salt should also be added. Coarse salt will crack and fly all over the place when heated, but this will not occur if it is ground finely first. The salt itself has practically no action on the gold or impurities, but as it becomes very fluid when molten it prevents slags from frothing over. Nitre should be added from time to time, and the temperature kept low. The base metals will be turned into oxides, and when the gold melts it will separate from them. By prolonging the heating before melting a greater amount is separated than if suddenly melted, for once gold has melted fluxes have but little action on it. As soon as the nitre ceases to attack the base metals, the temperature is raised, the whole lot melted and poured. The slag in this case consists of salt, which is fluid, and stops on top of the mould and oxide of potash containing oxides of the baser metals. The slag will easily wash off with water, leaving a very clean gold. When the retorted gold is very dirty it should be melted down in a clay pot as before, but when it is necessary to raise the temperature borax instead of salt should be added. The borax dissolves the oxides of the metals which have separated out, and makes a fluid slag, allowing the gold shots to sink into the main mass. Sometimes if borax is not added lumps of oxide of copper or other oxides will form a crust on top and prevent all the gold being poured out. By taking care, the most impure bullion may be purified to such an extent that it contains no base metals; that is, only gold and silver will remain. It is quite a mistake to think that base bullion can be purified to any extent by fluxing alone after it has once been melted. If air is blown through the molten gold covered with borax some metals may be removed, but there is danger of loss through spitting; there is also some difficulty in arranging for a constant stream of air. Compressed oxygen would be better, but there should be no necessity for the use of such methods of purification. Chlorine will also remove both base metals and silver, but its application needs skill, or large losses may take place.

Blacklead pots will do for melting in as many as fifty times. Clay pots should only be used once for smelting. The whole of the pots and slags should be saved; if care has been taken they only contain a very small quantity of gold; but in some cases they may, with improper fluxing, be rich. The coarse gold present may be got from them by amalgamating in a berdan basin, the overflow should be saved and sold to the smelters. In some cases the bulk of the gold may be extracted from these overflow residues by cyaniding them.

Purification of Skimmings.—Skimmings may consist of amalgam containing gold, silver, copper, lead, sulphur, arsenic, antimony and other metals. These should be retorted by themselves. The black mass can then be put in a clay pot and heated. Sulphur in lumps should be added; the sulphur will combine with the base metals, and gold will sink to the bottom. As soon as all the infusible lumps are gone, and the whole of the metals are in tranquil fusion, the contents may be poured into a deep conical mould. The gold will run to the bottom, and the sulphides will form a layer above it. These sulphides should be saved to see if there is still gold in them. This may be done by melting them down again, and when molten, stirring round with an iron rod; the iron will partly dissolve and displace some copper, which will carry down the gold and silver. The slags will then, as a rule, carry only a small quantity of gold and silver. They should, however, be saved, and, if rich enough, may be sold to the smelter.

Gold Iron Compounds.—If gold is alloyed with iron, as is often the case when too much sodium amalgam is used, the alloy should be melted with galena or sulphide of lead. The iron will form a sulphide, while the lead will be reduced and absorb the gold. The gold may be subsequently got from the lead in a manner described later on.

Gold from Magnetings.—To get gold from magnetings, place them in a tub which has a drain tap from it; add about 10 per cent. sulphuric acid, allowing it to slowly drain through the iron; the cementing rust will rapidly disappear, and the outside of the iron will be dissolved off, thus loosening the amalgam. Catch the liquor that drains through, and pour it back as soon as the tub becomes empty; repeat this for three or four days, then wash the iron with water to displace the sulphuric acid and sulphate of iron; place the magnetings on a bath of mercury and work them in contact with it: the amalgam will readily pass into the mercury and the clean particles of iron may be skimmed off or withdrawn by a magnet. Squeeze the mercury through a close piece of calico or chamois leather, and retort the amalgam.

Skimmings may also be treated by first boiling them with strong caustic soda and then adding mercury to the bath—a cast iron pot would do. Stirring round well, the bulk of the amalgam will pass into the mercury.

Gold from Black or Grey Sand.—Gold may be got from black or grey sand by stirring round well with mercury in a large mortar or basin. The gold readily amalgamates. On introducing a pipe with a nozzle in it, so that water may be delivered under pressure, the volume of water may be adjusted so that the sand will just flow over the edge of the basin and the mercury remain behind. A larger vessel, such as a panning off dish, may be placed so as to hold the basin and catch the first overflow and a tub to hold the tin dish to catch the second. If any mercury was swept over, the process may be repeated, and the whole of the gold recovered. Where larger quantities of tinstone and black sand have to be treated, it is advisable to first sluice them evenly over an amalgamated copper plate having a fall of about 2 inches to the foot. The residue may then be placed in a vat, provided no pyrites are present, and treated by chlorine solutions, and the gold dissolved out. The method of applying chlorine solutions has been described elsewhere.

The Clean-up of Gold from Cyanide Solutions.—It has fallen to my lot on several occasions to recover bullion from material obtained at chlorination and cyanide works. One of the greatest troubles to beginners in these operations is caused by the impurities in their precipitates, and a few precautions taken at the outset will render their work a pleasure instead, as it often is, a source of worry, trouble, and annoyance. The first rule which must be absolutely adhered to is to have perfectly clear solutions to precipitate from. Do not neglect any precautions to carry this into effect. Always have settling vats and filter vats arranged so that if any muddy or milky liquor runs from the sand vats, it will be all clarified before it gets to the precipitating boxes. The amount of slime may appear small, but it is always largely in excess of the gold present, and it may be found at the clean-up that there is a pound of material for every ounce of gold, and to get rid of this, perhaps as much as two pounds of flux must be added. There is little chance of recovering all the gold, when it is lost in such large quantities of unnecessary slag. Assuming that cyanide solutions are to be precipitated, I would advise the beginner never to use charcoal unless he can get it properly prepared; it takes such a large amount of charcoal to precipitate such a small amount of gold that unless the charcoal has been freed from all dust and dirt the material left on burning will be so greatly in excess of the gold as to cause trouble and loss in smelting. This objection does not apply to the same extent in large works where properly prepared charcoal is in use; but in any case it is doubtful if charcoal precipitation can compete with zinc. Another trouble with charcoal is due to the fact that it is such excellent filtering material that all the dirt in the solutions is deposited on it.

Preparation of Charcoal for Precipitation.—Assuming charcoal is used, some soft variety should be chosen; it should be broken into very small pieces and then sifted through a fine sieve; the bulk of the dirt and earthy matters will go through. The coarser grains should be evenly packed, so that clear solutions only flow through. On cleaning up the cyanide solutions should be drained out and a water wash put on to displace any soluble salts. The washed charcoal should be burnt to ash, not one speck of charcoal itself being left.

Smelting the Precipitate.—The ash may be smelted down with borax alone; if sand is present the addition of some carbonate of soda will help. In case soda and such fluxes have to be used, it is better to use a fireclay crucible. Borax is about the only flux which will not attack a blacklead pot. In case clayey and other materials have got into the charcoal, it is almost impossible to smelt it. It does not pay to smelt in crucibles material which will only run three or four per cent. of gold. The gold from such material is also exceedingly difficult to amalgamate: it may be ground for hours with mercury, and only a small percentage will be recovered. Those who feed it into the battery in the fond hope that the gold will be caught there are simply deluding themselves. When once gold has got into such material, it is better to sell it straight out to the smelters.

Clarifying Solutions.—When zinc is used for the precipitation of gold from cyanide solutions, only clear solutions should be allowed to pass in. The solutions should always pass through a clarifying vat before entering the zinc boxes. One plan for making this is to have a tall vat and divide it by a horizontal filter. The filter may be tacked on to a wooden hoop or piece of wood bent round to fit the interior of the vat; hessian may be used for filter cloth, and some packing inserted between the hoop and sides of the vat; by having the filter within a foot or eighteen inches from the top of the vat, and the inlet pipe just below the filter, a slight upward pressure may be arranged for; this has the effect of keeping the solution flowing evenly through. The sediment will all fall and remain at the bottom of the vat. Another simple plan is to place the partition vertically in the vat, so that the solution in travelling from one side to the other of the vat has to pass through a vertical hessian screen. The muddy material in this case also falls on the feed side of the screen; the outlet and inlet in this case also had better be near the top. The vat thus acting as in the previous case mainly serves for settlement of fine material carried over. In some cases it may be desirable to have a charcoal filter at the head of the zinc boxes. This will serve to remove the rest of insoluble material, and the small amount of charcoal required precipitates very little gold.

To Clean-up Cyanide Solutions.—The clean-up from cyanide solutions is a simple matter if only gold is present; but in many cases where lime is used for neutralising and clarifying solutions a copious deposit of calcium carbonate takes place below the zinc and even on the zinc in the boxes. On the zinc itself arsenic and antimony will be precipitated from alkaline solutions, and copper comes down readily. Certain solutions also carry a large quantity of lead, which is precipitated with the gold. A considerable amount of lead is usually contained in the zinc; this will remain when the zinc dissolves. The clean-up is effected in the usual manner by starting at the top compartment of the zinc box. The whole of the zinc is put on a coarse, strong 8 or 10 meshes to the inch sieve, and the material teased out in a tub of water. By agitation the fines and small zinc, the crusts on the zinc, are all carried through. All cemented lumpy pieces are broken up. The material on the sieve, mainly consisting of broken zinc, is put back into the first compartment. The milky liquid which remains in the box after the zinc has been removed is run out or scooped out and placed in a second tub. All the white, milky-looking liquids go into the second, and all the black or dark slimes into the first tub. The material from the second compartment is similarly treated, and the shortage in the first compartment made up from the second. The whole of the boxes should be gone through, for although very little gold will be got after the second or third box, the teasing out of the zinc and rubbing the incrustations from its surface are necessary for effective action. After the material and slimes in the tubs have subsided, the clear liquid should be drawn off with an indiarubber siphon. This liquid and all others in subsequent operations should be run into a sump. Next half fill each tub with clean water. Take 10lb. strong sulphuric acid and pour a small quantity, say, one pound, into the tub; a violent effervescence takes place. The gases given off are very often deadly poisonous —very often arseniuretted hydrogen, phosphoretted hydrogen, and hydrocyanic acid are present. It is due to the inhalation of these gases, even in small quantities, that people at cyanide works have lost their lives; even if not almost immediately fatal, the effect on the system is lowering and depressing. Treatment with acid should be carried on out in the open air or in a hooded vessel. In case of solutions boiling over, it would be as well, at first at any rate, to place the tub over a larger one to catch any overflow. The acid should be added little by little. After some time the operator will be able to tell almost exactly how much may be safely put in. When quite tranquil, the mass may be occasionally stirred, so as to bring zinc enveloped by sediment within reach of the acid. If no action goes on, and the wooden stirring rod shows a lot of pieces of zinc still undissolved, then more acid must be added. It is better, before adding acid the second time, to lead a steam pipe —preferably of lead—into the tub and boil up with steam, and to allow the tub to become nearly full of hot water. In case steam is not available, add hot water and stir. Allow the sediment to settle and decant the clear liquor into another tub. This will serve to settle any fine gold which has passed over. The liquor from this may be run into the sump or vat mentioned before. The tub is half filled with clean water as before, more acid is added until no zinc shows, steam is passed in or hot water added, and the clear liquor decanted into the settling tub, thence into the sump. By this treatment, all the zinc should have been dissolved, but there will still remain all the lead, copper, arsenic, antimony, which were precipitated, also a good deal of zinc sulphate which has not been washed out. This should be got rid of, otherwise it will give trouble in smelting. By filling the tub up with hot water at least three times and decanting the clear liquor each time, always passing it through the settling tub, then to the sump, practically all the zinc may be removed. If it is considered desirable to remove the other metals as well, then the tub should be about quarter filled with water and nitric acid added in small quantities. The lead and copper and arsenic will readily pass into solution; the antimony will be rendered insoluble. As a rule, very little silver dissolves, while none of the gold does as long as base metals are present. The tub is filled up with water as before, and the liquid decanted off and run into the sump. The liquid still with the gold should now be got rid of. This may be done by filtering. A filter can be readily constructed by taking four laths each about 3 feet long, and nailing them at the corners. Tack on to this a yard and a quarter of unbleached calico; lay on top of this without folding a sheet of filter paper, which will just cover the cloth. Allow this to rest over a tub. Bring the gold precipitate over near it and transfer it with an enamelled dish to the filter paper. Wash everything out of the tub with water; add it to the filter paper, taking care not to break the paper. When all the water has drained out, pour over the precipitate boiling water from a bottle, taking care not to crack the filter paper. Allow all the water to drain through. Repeat this washing twice. All the soluble salts will now be washed out. Next make a strong solution of nitre. Pour this solution over the precipitate. It will soak the whole lot of it, and enable you to burn your filter paper and the organic matter in the precipitate. After this has drained, the filter should be put in a warm place and allowed to dry. All the drips and drainings, and all the washings should be added to the sump. When dry, the precipitate may be detached in cakes or dried chips from the paper. In some cases this operation may be hastened by getting an old panning off dish or camp oven and inverting the dish over the precipitate, then tilting the whole lot upside down; the wet precipitate will fall into the dish, and the filter paper will now be on top. The filter cloth may be removed, washed, and made use of repeatedly.

Drying the Precipitate.—The filter paper should be peeled off from the precipitate and dried in a separate vessel. The paper should be burnt by itself. If it is properly dry, there is no trouble about this, for the nitre has converted in into touch paper. The precipitate may be transferred partly by tipping out and partly by a pliable knife or spatula into a cast-iron vessel such as a camp oven. It may be rapidly dried in this. Borax may be fed in and the whole lot stirred round and dried. If mercury is present, as it invariably is when tailings are being treated, the heating should not be carried too far or mercurial vapors will escape. If present in quantity the mercury should be saved by putting the precipitate in a cast-iron retort and distilling off the mercury. In this case no borax should be added; the small amount of nitre present will be beneficial. The dried precipitate is taken and smelted, preferably in a clay pot, standing in the bottom of a blacklead one, the latter serving as a guard. The gold obtained in this way is as pure or purer than that obtained from the battery. With careful manipulation, there is no loss, and there is none of the fearful slag and smelting troubles such as exist when the zinc gold precipitate is roasted and then smelted in bulk.

The milky precipitate in the second vat, consisting as it often does of carbonate of lime, is treated with hydrochloric acid. If sulphuric acid is used, insoluble sulphate of lime forms; this gives much trouble in smelting. The material left in this tub is treated in the same way as in the other, the excess of wash waters and solutions going to the sump. The precipitate is thrown on to the same form of filter as before and washed. After thorough washing, a saturated solution of nitre is allowed to soak through it. The filter paper is burnt separately, and the precipitate may be added to the other.

Smelting the Precipitate.—In smelting these precipitates, the whole lot should be allowed to shrink in the melting pot before a fresh quantity of material is added. On no account should the mass be allowed to melt, otherwise the material is almost sure to overflow. The precipitate may be added to the pot while still in the fire, by placing an iron pipe wide enough to fit into the mouth of the crucible and pouring the precipitate down this. A little pulverised borax should be put on top at each addition. This spreads over the charge and prevents dusting. Should there be a tendency to boil up at any time, the addition of pulverised salt will generally prevent any overflow. It is not desirable to fill the pot too full, nor to leave it in the fire too long. When all frothing has ceased, make the fire up and heat very strongly for about half an hour. This will cause the slag to become more fluid and allow all shots of gold to settle. While still hot pour into a deep mould so that the gold may all run to the bottom. It does not matter if the slag overflows so long as pouring is continued into the centre of the mould. All slags and crucibles from these operations should be saved.

The liquid in the sump may not have any gold worth troubling over, yet since many cases where heavy losses have taken place are known to me, and since this is an operation where about the only chance of loss comes in, it is well to err on the safe side, and to be absolutely sure that no unaccounted for material has escaped. The sump may contain old scrap iron. The whole of the gold, silver, lead and copper are precipitated on the iron, which in its turn is slowly dissolved away.

I will now deal with the purification of cyanide bullion, the smelting of chlorination precipitates, and the scaling of copper plates.

When silver is present in large quantities in cyanide bullion, it may be easily separated, and the gold obtained in a pure state. In this case, assuming there is much copper in the bullion, it need not be removed by preliminary treatment with nitric acid; the precipitate may be washed free from sulphuric acid, and then dried and smelted. The base bullion should then be assayed, and if it contains 40 per cent. of gold or less it should be granulated straight away; if it contains much more than 40 per cent. of gold, some zinc cyanide gold, or ordinary chips and scraps of gold zinc from the precipitation boxes, should be melted down to a small bar with borax, and poured. The bullion rich in zinc should be added to that rich in gold, and the percentage of gold brought down to about 40. Should any lumps of zinc be to hand, it is simpler and answers just as well to add the requisite quantity as soon as the gold bullion has melted. To granulate the bullion, get a large enamelled bucket, fill with cold water. Place a dish or enamelled basin at the bottom of the bucket. Pour the bullion from a height of a few feet in a thin stream into the water, giving the crucible a circular motion all the time so that the drops do not fail in the same place. The whole mass should be broken into flat discs and hollow grains, thereby exposing a large surface for the acid to attack. If the layer of water is too shallow the shots may not be broken up sufficiently and may fuse into a pyramidal heap down below. If room enough is available the bullion may be poured on to the edge of an inclined plate; it is thus scattered and falls in flakes into water below. The bullion is then treated by washing it with water, then place it in a large porcelain crucible, add three parts by volume of water for every part of nitric acid; stand it on a sand bath made by making some sand red hot and pouring it into a sheet iron vessel. A violent action soon goes on and the zinc, silver and copper are dissolved out. When brown fumes cease to come off after warming, pour off the clear liquid and wash by decantation with hot water three times. By this treatment most of the lead is got rid of which would not be attacked by a strong acid. Next, pour one part of water and two parts of nitric acid on the brown lumps of gold and heat to boiling for about half an hour. The balance of the silver will be dissolved. The strong nitric acid solution, as a rule, dissolves but little silver, and it had better be kept for treatment of the next lot of granulations. The spongy gold should be washed about four times with distilled water, and then with some warm dilute ammonia to dissolve any silver chloride out. These washings can be added to the first solution obtained. Some dry powdered nitre may now be sprinkled on the gold; it will be absorbed. The gold may be dried in the dish. When dry put in a pot which has been glazed by borax; cover with a layer of salt and melt down. Gold of 99.6 purity may be obtained, or gold worth over £4 4s. per ounce.

To the silver solution which is now dilute a small amount (about 10 cubic centimetres) of sulphuric acid should be added; this precipitates the lead and also carries down any fine specks of gold which may have washed over. The milky solution is stirred and allowed to settle; when the top liquor is clear it is poured off into another vessel, and the precipitate remaining is thrown on a funnel and washed two or three times with water. It can be put to one side, and when sufficient quantity of these has accumulated, may be soaked in nitre, dried, and smelted with its weight in soda and some borax and a rod of iron for lead containing gold.

To the clear solution containing the silver salt is added. The silver is thrown down as a curdy precipitate. Stir this, allow to settle, add some more of a solution of salt; if no more cloudiness occurs the silver has all been thrown down. Allow to settle. Pour the clear liquid into the residue sump. Wash the chloride of silver by stirring up with fresh water, allowing to settle and pouring off the clear water. Then take the washed silver chloride and melt it in a mercury bottle with the top cut off.* Pour it out, place it with scrap iron in an acidulated solution. When reduced, wash with water several times, then boil with dilute hydrochloric acid. Smelt with nitre and borax in a clay pot, pour into a clean mould; the silver will be over 99 per cent. in purity. Personally I prefer the second method of dealing with auriferous bullion, since most of the material we have to deal with contains from thirty to forty per cent. of silver. Probably strong sulphuric acid could be used, and would be used on a large scale, instead of nitric acid. On a small scale the amount of acid used is so small in comparison with the value of the metals recovered, and the process so speedy, that dirty zinc bullion should be only produced by those ignorant of the rudiments of such technical work. The advantages of this method of treatment are that the minimum charge is made at the mints for the treatment of such gold, as against a special treatment charge on the number of gross ounces made for base bullion on poor lots. The silver may be sold at highest market price with no deductions, whereas if left in the gold the mints will not allow anything like its market value. Lastly, if sent through the post office the charge of 2d. per ounce will only be paid on the gold, instead of on perhaps three times the quantity of base bullion.

In dealing with auriferous material by chlorine solutions the same scrupulous care must be taken to keep the solutions clean. In the case of the use of the gas by Plattner's method, no impurities are added with the chlorine, and whatever comes through is usually in solution; sometimes the sand filter becomes disturbed and fine sand runs through. This should be caught in a small intermediate

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* I am indebted to Mr. Alsop for this hint. In fact, most of the information given has been acquired through sheer necessity, since the literature on the subject is wanting in necessary detail.

jar or sump. If any quantity is discovered in the precipitating vat mixed with the gold, do not attempt to smelt it. A better plan is to take the mixed sand and precipitated gold and pan it off. In this case, it is the gold which passes over, the sand being left behind. There may be a little gold still with the sand; if so, place it where the chlorine solutions will flow over it as they run from the sand vat. The gold will soon dissolve and be carried away with the other gold in solution. It is well to have an intermediate settling vat in all cases. This allows sediments and precipitates of all kinds to subside. Instead of this a vat or large jar provided with a carefully graded sand filter may be used; it must be watched to see that it does not clog, and great care against overflows should be taken. The solution should have sulphuric acid added to it if none were present before it enters the settling vat or sand filter. This is added to precipitate the lead and lime as sulphates, otherwise they would come down when sulphate of iron is added to precipitate the gold. The clear solution in the precipitating vat should have no precipitating agent added which will throw down more than the gold. Copper is often present in solution, but this is not precipitated by sulphate of iron. After the gold has settled siphon or drain the clear liquor off from the top; let it run through a sawdust filter before it escapes. Sluice the gold sludge out into a tub. Allow the gold to settle, decant the clear liquor off, add a few pounds of 1 in 10 sulphuric acid, stir round; allow to settle. See that no gold has gone into solution. This very often takes place owing to the precipitation of manganese oxides with the gold; this in saline solutions will liberate chlorine and cause the re-solution of some of the precipitated gold. Such an action invariably occurs when the permanganate method is used. Add some clear water to the tub, allow the gold to settle again, decant the clear liquor off. The washings from the tub can be added to the precipitating vat, and any small quantity of gold caught in the next clean-up. Transfer the precipitate to a filter made in the same way as recommended for cyanide precipitates. Wash by pouring boiling water from a kettle over the gold and allowing each lot to drain through before adding the next. After the third washing, pour a concentrated solution of nitre over the mass, allow to drain; partly dry the precipitate and filter paper, detach the cakes of slime gold. Dry the paper: when perfectly dry ignite, and it will burn away completely, leaving only a film of golden ash. Add this to a French clay pot, then some borax, then the precipitate, which should be dry: cover with powdered borax and smelt. The nitre prevents dusting, and will oxidise any impurity. The borax dissolves any intermingled material. A guard pot should be used as before recommended. If there is any tendency for the material to froth over, add powdered salt. The gold so prepared is almost pure, and often realises £4 4s. 9d. per ounce. Charcoal should be used, whenever possible, for the precipitation of gold from chlorine solutions. In this case a very large amount of gold may be precipitated on a comparatively small amount of charcoal. In burning the charcoal the smallest amount of fuel is necessary, also a limited draught.

If the charcoal was washed practically free from chlorine and soluble salts, there is little trouble in burning, and the amount of gold left after burning will run as high as 80 per cent. There is no trouble in melting this residue. Borax is the only flux needed. The slags and pots should be saved; these can be treated by amalgamation in a Berden pan, and the overflowing slimes settled, bagged, and sold to the smelters.

If a parcel of slag is too small to send to the smelter it may be melted down with about 10 per cent. of its weight of litharge in a large crucible. When thoroughly melted, stir well with an iron rod and allow the iron to stand in the pot. Metallic lead will be reduced. This will alloy with the silver and gold, carrying them to the bottom. Remove the iron and heat very strongly for about half an hour. Pour into a mould, allowing the slag to overflow, until the whole lot is poured. Allow the mould and contents to cool. Break the slag off. There will be found a hard lead bar or button, and on top of this a hard brittle metallic layer. This is sulphide of iron, and as a rule is almost valueless. The lead contains the gold and silver. Make a large cupel by stamping bone ash into a jam tin or similar vessel. Stand this on a roasting dish in a hot furnace half filled with glowing coal. When the cupel is very hot, place the lead bullion on it. It should melt straight away; shift the cover of the furnace so that air plays over the surface. The lead may be wholly oxidised and removed as in an assay. In such lead bullion obtained in this way or by other methods there may be arsenic or antimony. It is easier to cut the lead bullion into pieces and scorify it in large scorifiers until it is reduced in bulk and has a high lustre, and is hard and brittle. In this case it is rich in gold. The smaller bulk may then be cupelled in a large cupel placed in the muffle.

Copper Plates.—The bulk of the amalgam may be removed from copper plates by covering them with bags, and blowing in steam between the plate and the bags. After an hour or two the amalgam will soften and may be scraped off with a soft iron scraper. The plates are not injured by this treatment, and only a film of amalgam need be left on. When it is desired to remove the whole of the gold the plates are scaled. If the coating of amalgam is thick, there is little trouble in removing the gold. The plates only require to be gently heated, the mercury volatilises, and the gold may be peeled off in strips. A thin layer is difficult to remove. In this case, the plate should be slowly heated until the mercury is driven off; overheating is bad. Next wet the plate with strong hydrochloric acid and cover it over and leave for about 24 hours. Then add evenly over the plate a mixture of equal parts of powdered ammonium chloride and nitre. Place the plate over a fire and heat gently. The compounds melt, find their way under the surface of the layer of gold, and cause it to blister. As soon as the plate is red hot, it is removed and allowed to cool. The gold can then be scraped off; it may be scrubbed off with water into a tub. Many other compounds will serve the same purpose. As a rule, patches of gold, especially when not much amalgam was present, will remain. These may be retreated. Or if the plates are not wanted again, the simplest method of removing the gold is to make the plate red hot and sprinkle sulphur over it. The gold will rapidly blister up and may be readily removed.

When the plates are to be re-amalgamated, the surface should be cleaned with dilute nitric acid. This should be washed off, and the surface scoured with a bath brick and some caustic soda. Mercury should then be ground up with dry sugar and a minute quantity of potassium cyanide. This can be rubbed over the plate, when it will at once amalgamate. The amalgamated surface can be sprinkled over with sugar and left all night. It will remain bright. Next day all the coppery amalgam should be rubbed off and some silver amalgam and sugar rubbed on. The silver amalgam may be made by grinding some of the pure reduced (not melted) silver with mercury. The silver amalgam is rubbed on daily—the old material always being removed—for about 5 days. If the copper were of good quality, the plates will remain bright and will not discolor. If the plates have been properly scaled, there is practically no gold left in them: it is quite a mistake to believe that they absorb more than traces of gold. Sometimes mercury containing amalgam leaks round to the underside of a plate and coats it as well as the copper; this may always be seen when the plate is first heated.

Mining and Metallurgic Costs.

A tabulated sheet of costs for mining, milling, cyaniding, and other operations cannot be taken as a truly comparative one. Conditions are so varied in mining centres that such comparisons, without a statement of corresponding circumstances, would be misleading. For instance, take a simple case where open-cut methods are adopted. The material shifted may be massive, dense pyrites or rocky overburden, as at Mt. Lyell; it may be a granitic rock of uniform composition as at the Anchor Tin Mine (Tas.); or it may be easily mined stuff such as occurs at Mt. Bischoff, part of Mt. Morgan, and many alluvial mines. Not only is the nature of the material different, but climatic conditions, rates of wages, and conditions for the removal of the broken material all influence the total cost for this one item—mining. With regard to underground mining also, costs are only comparable on each field for similar mines. Wages, timber, explosives, fuel and machinery should all be taken into account. The adoption of the short ton also by some mines may also unintentionally mislead. Even in taking all ordinary expenses into account, it does not always follow that those mines showing the lowest costs are always the best managed. A Victorian mining manager many years ago paid £30,000 in dividends from a mine which yielded only 3dwt. 15gr. per ton; he regarded that as an achievement. Years afterwards one of his men told me that this was possible because he regarded men as cheaper than timber.

Taking milling, it would be absolutely absurd to assume that because a large tonnage is crushed that a mill is doing good work, and vice versa. Where shovelled material is fed into a battery as at Mt. Bischoff, a tonnage is put through with tiny stampers equal to that with the heaviest stamps working on hard ore. Even when the materials are hard, the toughness, specific gravity, mesh aperture, and depth of discharge, and the quantity of water used have an influence which greatly modifies the output. Mills of the same type running on stone from Ballarat, and the heavily mineralised ore at Charters Towers, will give a widely different output. Where about 20 per cent. of heavy metallic minerals is present, a battery will seldom crush more than 60 per cent. of the tonnage it would put through in the same time if the material were clean quartz.

The comparison of roasting costs would also be highly misleading unless the physical and chemical nature of the minerals fed in and the products turned out are simultaneously considered. The amount of sulphur present is but a poor guide as to the work done by a furnace, unless the other materials present are of the same nature. Roasting for smelting and even for cyaniding is a different operation from that required for chlorinating. Bearing these facts in mind, the costs given are of much value in showing what can be done on the various fields in Australia. Up to a very recent date, this branch of the work was almost ignored except by very few mining companies. Mining was looked upon, not as a business, but as speculative venture, in which luck played the most prominent part. The tendency now is to proceed on strictly business lines, and the man who has no business capacity, has no right to have control of shareholders' money, while the man who has will always try to outrival his neighbor in increasing his extractions and lowering his costs. The adoption of some uniform system for mining costs is a matter which mining engineers and managers should agree upon. At present the greatest diversity prevails, and in the case of most mines the most meagre information is afforded in reports and balance-sheets. The engineers of Western Australia have set a good example, but uniformity has not yet been arrived at.

Mining differs mainly from other business ventures in that the life of a mine is comparatively short and, in general, more uncertain, while, when the mine is worked out the stock and plant on hand only possesses a fractional value of its cost price. It is the custom on some works to write off this cost at once, on others to distribute it over some years, while many write off something annually for depreciation and wear and tear. The somewhat extraordinary method of ignoring capital originally or afterwards provided, in writing up costs-sheets, is not sound. Costs per ton may readily be reduced if the capital provided for machinery or other appliances is ignored, but in all these cases there is an economic limit beyond which any further capital expenditure is waste. It is the duty of the manager or engineer to determine what this limit should be, and in all cases his mine should be so opened up or tested that a few years' supply of ore is in sight. In all cases, capital, mining, development, treatment, management and general expenses should be considered. The omission of any of these gives returns which cannot be properly compared. The following tables were obtained from official reports, or through the courtesy of the manager in charge:—

Lake View Consols.

Ore Extraction for Year Ending June, 1903.

Costs per Ton of 2240lb. Tons Extracted, 81,785.

	s.	d.
Labor—		
Miners, truckers and mullockers, proportion of firemen, bracemen, etc.	7	0.174
Supplies—		
Explosives, timber, steel, candles	1	3.916
Assaying	0	2.977
Power	2	1.442
Repairs	0	6.110
Total	11	0.564

Milling—

Tons crushed, 81,785.
50 stamps running 344.65 days.
Stamp duty, 4.745 tons per 24 hours.

	s.	d.
Salaries	0	0.189
Labor	0	6.743
Saltwater	0	1.636
Supplies	0	1.946
Repairs	0	10.085
Power	2	2.100
Assaying	0	0.220
Ore transport—Labor	0	5.597
,, Supplies	0	0.748
Total	4	5.264

Concentrating—

Tons concentrated, 81,667.

	s.	d.
Salaries	0	0.112
Labor	0	5.076
Supplies	0	0.149
Saltwater	0	1.666
Repairs	0	1.958
Power	0	3.368
Assaying	0	0.202
Total	1	0.531

Bromo-cyaniding—

Tons treated, 77.260.

	s.	d.
Salaries	0	0.216
Labor	2	4.278
Lime	0	1.297
Zinc shavings	0	0.771
Potassium cyanide	2	5.399
Cyanogen bromide	2	1.815
Sulphuric acid	0	0.333
Saltwater	0	1.745
Supplies	0	4.147
Repairs	0	10.221
Power	3	7.765
Assaying	0	2.381
Royalty	0	9.856
Total	13	2.224

Treatment of Concentrates. Tons Treated, 4147.

Roasting—	s.	d.	s.	d.
Salaries	0	1.358		
Labor	4	1.218		
Fuel	2	8.927		
Supplies	0	1.487		
Ore transport	1	11.419		
Repairs	0	7.023		
Power	1	9.776		
Assaying	0	5.642		
			11	10.850

	s.	d.	s.	d
Cyaniding—				
Labor	2	9.430		
Lime	0	1.129		
Zinc Shavings	0	2.008		
Potassium cyanide	3	9.910		
Sulphuric acid	0	1.280		
Saltwater	0	4.292		
Supplies	0	10.692		
Repairs	1	3.277		
Power	3	10.172		
Assaying	0	8.437		
			14	0.627
Total			25	11.447

Summary of Expenses for 81,785 tons. Fine gold won, 63,215oz.

		s.	d.
Ore extraction per ton		11	0.564
Ore reduction		19	5.631
General working expenses		3	5.369
Management and General Expenses		1	2.936
Total	£1	15	2½

The percentage of gold extracted was 88.55 per cent. The residues average 2dwt. per ton. The concentrates produced amounted to 5.07 per cent. The ratio of gold in concentrates to that in slimes is not shown.

The costs for treatment at the Perseverance mine for November, 1903, where the whole of the ore is slimed, roasted, filter-pressed and cyanided, amounted to 17s. 11d.

Great Fingall Consolidated.

Costs for July, 1903. 8270 Tons Treated.

	s.	d.	s.	d.	£	s.	d.
Rock-breaking—							
Power	0	1.45					
Labor	0	1.00					
Repairs	0	1.58					
Supplies	0	0.12					
			0	4.15			
Transport to Mills—							
Power	0	0.35					
Labor	0	2.33					
Repairs	0	0.17					
Supplies	0	0.20					
			0	3.05			
Milling—							
Power	1	3.36					
Labor	1	0.71					
Repairs	0	6.33					
Supplies	0	8.81					
Mercury	0	0.31					
			3	7.52			

	s.	d.	s.	d.	£	s.	d.
Concentrating—							
Power	0	2.09					
Labor	0	0.25					
Repairs	0	0.74					
			0	3.08			
Roasting	0	1.45	0	1.45			
Grinding	0	2.25	0	2.25			
Agitating and cyaniding—							
Power	0	1.08					
Labor	0	9.34					
Repairs	0	1.17					
Supplies	0	1.01					
Cyanide	0	7.79					
Acid	0	0.61					
Lime	0	1.45					
			1	10.45			
Filling, Drying and Emptying Presses—							
Power	0	0.99					
Labor	0	3.16					
Supplies	0	0.01					
			0	4.16			
Disposal of residues—							
Power	0	0.71					
Labor	0	11.60					
Repairs	0	0.43					
Supplies	0	0.26					
			1	1.00			
Precipitating and Smelting—							
Power	0	0.33					
Labor	0	0.27					
Repairs	0	2.62					
			0	3.22			
Assaying and sampling—							
Power	0	0.06					
Labor and salaries	0	0.83					
Repairs	0	0.12					
Supplies	0	0.63					
			0	1.64			
					0	8	5.97
Stoping—							
Power	0	7.97					
Pumping	0	2.45					
Labor and salaries	5	7.35					
Repairs	0	2.33					
Supplies	1	8.49					
Assaying and sampling	0	0.34					
			8	4.93			
Management and general expenses...	3	1.47	3	1.47			
Total					£1	0	0.37

Summary of Great Fingall Costs for July, 1903.

	£	s.	d.	£	s.	d.
Reduction—						
Power	0	1	10.10			
Labor and salaries	0	3	5.55			
Repairs	0	0	10.83			
Supplies	0	2	3.49			
				0	8	5.97
Mine Development—						
Power	446	0	11			
Labor and Salaries	1972	9	0			
Repairs	240	7	5			
Supplies	386	12	4			
Proportion of assaying and sampling	54	12	11			
				3100	2	7
Plant—						
Labor and Salaries	598	8	9			
Supplies and Sundries	792	0	0			
				1390	8	9

Kalgurli Gold Mines, Ltd.

Average Cost per Ton up to October for Year 1903.

Ore treated, 12,317 tons.

		s.	d.
Rockbreaker—Driving and feeding...		0	8.26
Aerial tram		0	3.99
Power—Enginedrivers, firewood, water, etc....		2	11.59
Balls mills and crushed ore conveyor		0	10.91
Furnaces, firing, firewood and water		4	4.98
Conveyors and elevators...		0	6.97
Separation and amalgamation...		0	8.45
Cyaniding—Cyanide and caustic soda...		2	5.23
,, Fresh water		0	8.30
Agitating and filter pressing (air)		3	3.76
Oiling and attendance		0	3.88
Precipitation		0	6.81
Smelting		0	2.88
Assaying, Sampling, etc.		0	2.98
Superintendence		0	7.18
Electric lighting...		0	2.49
General Treatment		0	3.85
Total		19	6.51
Proportion administration expenses...		0	11.75
	£1	0	6.26

Hainault Gold Mines, Ltd.

Ore Raised for the Year to October, 1903: 10,318 tons.

Costs per Ton.

	s.	d.	s.	d.
Labor	2	6.39		
Loading and trucking	2	1.58		
Filling stopes	0	8.03		
Repairing and sharpening tools	0	4.09		
Superintendence	0	4.88		
Sundries	1	5.20		
Total			6	2.48
Stores—				
Tools	0	0.91		
Candles	0	1.59		
Explosives	0	6.37		
Timber	0	2.87		
Repairs and renewals	0	1.56		
Electric Lighting	0	0.19		
Assaying	0	1.07		
Sundries	0	0.18		
Total			1	2.54
Haulage—				
Enginedrivers, stores, repairs, and fuel			2	4.30
Total cost of ore at pit mouth			9	9.32
Proportion of administration			0	1.90
			9	11.22

Treatment Charges per ton for 10,318 tons at Hainault Mine.

	s.	d.	s.	d.	s.	d.
Superintendence	0	3.88				
Feeders	0	3.12				
Picking ore	0	0.29				
Amalgamating tables	0	8.10				
Concentrating Tables	0	3.52				
Separation	0	2.47				
Repairs and renewals, tailings dam	0	5.34				
Sundries	0	1.89				
Total			2	9.10		
Stores—						
Repairs and renewals	0	5.22				
Tools and oils	0	0.37				
Screens	0	0.45				
Mercury	0	0.49				
Assaying	0	0.39				
Electric Lighting	0	1.71				
Water	0	11.51				
Sundries	0	1.19				
Compressed air	0	2.21				
			1	11.54		

	s.	d.	s.	d.
Electric Power	4	1.93		
Realisation of concentrates	0	3.35		
Cost of raw ore from mine...	9	1.92		
Administration expenses	0	2.44		
Total milling per ton of ore	9	4.36		
Cyaniding Costs—				
Filling vats and mixing solutions...	1	5.62		
Belt conveyor	0	0.93		
Emptying vats	0	4.29		
Repairs and renewals	0	0.43		
Trucking from vats	0	1.01		
Sundries	0	1.16		
			2	1.44
Stores—				
Cyanide and caustic soda...	1	1.26		
Zinc shavings	0	0.80		
Power for pumps	0	1.56		
Repairs and renewals	0	0.47		
Assaying	0	1.37		
Sundries	0	0.47		
			1	5.93
Total per ton...			3	7.37

Working Costs, Mt. Morgan.

Ore treated, 262,919 tons; average value, 11.69dwt. per ton.

This was composed of 51.84 per cent. sulphide or mundic ore.
22.59 high-grade oxidised ore.
25.57 low-grade oxidised ore.

Average value of mundic ore, 10.62dwt.

Average value of oxidised ore, 12.52dwt.

	Cost of Mining.	Cost of Treatment.	Total.
	s.	s.	s.
Oxidised ore	3.03	11.69	14.72
Mundic ore	11.74	15.18	26.92
Average...	7.51	13.51	21.02

Copper recovered per annum, about 100 tons.

Average amount of chlorine per ton of ore, 4.54lb.

Average amount of chlorine per ton of sulphide ore, 5.4lb.

Or nearly 10lb. per ounce of gold.

Mundic Works.—2.59cwt. of wood burnt per ton roasted. Cost of wood, 1s. 5.81d. Ore mined, 233,568 tons. Average value, 46.51s. per ton. Ore chlorinated to May 31st, 1903, 1,986,249 tons for 2,621,318oz.; value, £10,653,432. Dividends, £6,229,166, or £6 4s. 7d. per share.

Costs for Roasting and Chlorinating Ore at the Chance Mine, Cassilis (V.)

Composition of ore—

Silica and insoluble	5.5
Iron (Fe)	41.0
Sulphur (S)	33.0
Arsenic (As)	8.7
Zinc (Zn)	4.3
Copper (Cu)	trace
Lead (Pb)	1.6
Calcite ($CaCO_3$)	1.4

Sulphur contents after roasting in Edwards' Furnace—

Sulphur soluble in water as SO_3	0.12
Sulphur in lead sulphate	0.32
Sulphur in sulphides	0.12

Cost of chemicals delivered on the mine—

Sulphuric acid	1.35d.	per lb.
Salt	0.705	,, ,,
Potassium permanganate	8.000	,, ,,

Charge for 10-ton vat—

	s.	d.
Sulphuric acid, 300lb. at 1.35d.	34	0
Salt, 240lb. at 0.705d.	14	0
Potassium permanganate, 28lb. at 8d.	19	6
Total	67	6

Cost per ton of roasted ore, 6s. 9d.

Capacity of one Edwards Furnace, 35 tons per week.

	s.	d.	
Roasting costs	18	6	per ton.
Handling ore	1	6	,, ,,
Total	20	0	,, ,,

	£	s.	d.
Fuel used in roasting per week, 20 cords at 12s.	12	0	0
Fuel used in generating power to work furnace, 5 cords at 12s.	3	0	0
Wages—			
3 men at £2 18s. 4d.	8	15	0
Emptying and filling, £1 5s.	1	5	0
Proportion of drivers' wages, £1 10s.	1	10	0
Half metallurgist's salary, £4 5s.	4	5	0
	£30	15	0

Total costs for roasting ores and treatment by the permanganate process—

	s.	d.
Roasting	18	6
Chemicals	5	4
Handling ore, etc.	1	6
Total	25	4

Cost Sheet, Anchor Tin Mine, Ltd., Lottah, Tasmania.
March 1st to March 28th, 1903.

	Wages. £	s.	d.	Average.	Explosives. £	s.	d.	Average.	Stores. £	s.	d.	Average.	Renewals. £	s.	d.	Average.
Excavation	428	6	7	11.04	85	16	5	2.21	6	13	10½	.17				
Maintenance of Plant ...	21	19	6	.56												
Transport to Battery ...	44	18	3													
Upkeep Trucks	6	2	11	15					3	2	3	.08				
Battery ...	173	11	0	4.47					32	10	6	.83	61	0	11	1.57
Upkeep ...	65	5	6	1.68												
Crushers ...	37	13	9	.97					3	17	0	.09	8	8	4	.21
Upkeep ...	9	9	5	.24												
Power ...	27	8	0	.70					1	4	0					
Sluicing and Clearing ...	6	4	9	.15												
Management and Office	29	0	0	.74												
Miscellaneous	12	13	10	.32					1	15	8					
Fodder ...									20	7	0	.52				
Freight ...									25	10	0	.65				
	924	4	3	1/10.24	85	16	5	2.21	93	17	8	2.41	69	9	3	1.79
Capital Account—																
New Addition to Trucks	2	10	9													
Fitting New Sluice Valve	3	7	0						0	17	0					
Fitting Settlers Timber for ,,	8	2	5						20	10	5					
	938	4	5						115	5	1					

Tonnage, 9311. Total cost less charges to capital and manager's salary, £1111 16s. 10d. Average cost per ton, 2/4.68. 87 stamps running. Duty per stamp, 4.46 tons. Average cost for 8 months 1903, 2/7.56 per ton.

Cassilis Gold Mining Company.

Tons Crushed, 270.

Milling and concentrating—

	£	s.	d.
Wages	30	12	0
Fuel	12	12	0
Upkeep and Stores	7	8	6
General charge...	15	12	0

Cost per ton crushed, 4s. 10.8d.

Roasting of Concentrates—

Tons roasted, 54. Two Edwards' furnaces.

	£	s.	d.
Wages	10	13	0
Fuel	12	0	0
Upkeep	0	9	0
General charges	5	0	6

Cost per ton roasted, 10s. 4.9d.

Chlorination—

Tons treated, 54.

	£	s.	d.
Wages	7	10	0
Chemicals	13	2	0
Smelting, stores	0	4	6
General charges	4	2	6

Cost per ton chlorinated, 9s. 3.1d.

Summary.

	s.	d.			
Milling and concentrating... ...	4	10.8	per ton	crushed.	
Roasting	2	1.0	,,	,,	,,
Chlorinating...	1	10.2	,,	,,	,,
Total	8	10.0	,,	,,	,,

General charges include proportion of assaying, clerical charges, and management.

The concentrates saved, according to this sheet, amounted to 20 per cent. of the ore crushed. The tailings and slimes were saved for cyaniding. The residues from the chlorination vats still contained from 7 to 14 dwt. of gold per ton, so that the method of treatment adopted cannot be looked upon as satisfactory.

Conclusion.

The mining and metallurgical progress in Australia has been so rapid that a portion of the descriptions given are no longer applicable. These alterations will be referred to shortly in this article. My visits to the centres mentioned were made during vacations, and it was not possible to visit every mining and metallurgical district; for this reason mainly only those fields presenting typical or novel methods were dealt with, it being unnecessary to include those having simple milling or mining methods. At the same time several phases of mining and metallurgy have had to be omitted, for instance, the important deep leads of Victoria, dredging for gold, the Wallaroo and Moonta mines, and others. I also desire to mention that the article on the treatment of Cassilis ore was written by Mr. W. Alpin, and that pp. 97-140, which give a great deal of statistical information on West Australia, were written in the office. In all other cases, the mines and works described were visited and the information given was obtained on the spot. It is necessary to state that valuable information was obtained from gentlemen whose names are inadvertently omitted from these pages.

A few notes on alterations in treatment methods in Western Australia will serve to show the latest developments. Messrs. Bewick, Moreing and Co. have obtained the management of several important mines, and the desire of their managers to show decreasing cost-sheets has led to several alterations in methods and details; for instance, in the Diehl process, as carried on by them in the Lake View Consols, the roasted concentrates are not mixed up with the other ore and treated by the expensive bromo-cyanide solutions, but after reduction to slime in the tube mill, are agitated in the ordinary way with potassium cyanide and filter pressed. It is also noteworthy that the Ropp furnaces have been replaced by Merton's furnaces at the Associated.

Generally speaking, it would seem as if the commonly accepted method of treatment of the sulpho-telluride ores of Kalgoorlie is by dry crushing and roasting in bulk, grinding and amalgamating in bulk, agitating the slimes with dilute cyanide solutions, and filter-pressing. Mr. Hewitson, whose opinion was given in favor of wet crushing, has altered his views on the matter now that sulpho-tellurides have become so universal at the deeper levels. The Diehl process does not seem to have been adopted anywhere else than on the mines previously indicated.

Concerning machinery, the Gates crusher holds its own, but it would seem as if the Griffin mills will have to make way for the Krupp mill; the former being more expensive as regards wear and tear. The tube mill is an excellent triturating machine, but the simpler grinding and amalgamating pans, in which two operations instead of one are carried on, are more suitable for the work done at present.

Furnaces are a highly debatable subject, but for this class of ore Edwards' and Merton's hold their own against the American types; it would seem as if Merton's furnace is more economical as regards fuel than Edwards', but with regard to roasting, they are both capable of giving excellent results.

Three-throw pumps have been used successfully for filling filter-presses. They give good results and are more economical than the compressed air montejus.

Agitation by compressed air is still used at the Kalgurli mine, but open steel cylinders are used, these being 12 feet 6 inches deep and 6 feet in diameter. The presses are filled by drawing the pulp from the bottom and forcing it into the press by means of Pearn's belt-driven three-throw pump. It is probable also that instead of the double treatment of sands by percolation in vats, the whole will be reduced to slimes and filter-pressed. This is mainly due to the reduction in the cost of filter-pressing. The total costs at this mine, including London expenses, have been reduced to 40s. 6d. per ton.

At the Associated the ore is broken with 2 No. 5 Gates Breakers and 2 Comet Crushers. Robin's belt-conveyors distribute this between 12 Ball mills, where it is crushed and passed through 900 mesh screens; it passes to 14 Merton Furnaces, thence to 20 pans; about one-third of the gold contents are amalgamated. The pulp passes over six Wilfley tables. The pulp is passed through 24 spitzkasten, 13 agitators, and six filter-presses, the treatment being the same as described. A 20 head stamp mill and concentrators have been erected for dealing with large bodies of low-grade ore.

The most notable Tasmanian alteration is the abolition of the hot blast at Mt. Lyell; the stoves have been dismantled, and the work is carried on as successfully as ever. The matte fed into the converters is also of much lower grade (about 35 per cent.) than at the date of my visit to the mine. The amalgamation of the Mt. Lyell and North Mt. Lyell companies has led to the use of the bulk of the Mt. Lyell ore as fuel and flux, but unfortunately the grade of ore available in the North mine was much lower than that previously removed. The Mt. Lyell Company is about to undertake the manufacture of sulphuric acid in Melbourne from its low-grade pyrites, and also to manufacture superphosphates.

The most notable alteration in Queensland is the gradual transition of the Mt. Morgan gold mine into a low-grade gold and copper one. Different methods of treatment will be adopted, but the processes described in detail are still carried on successfully.

The articles on New South Wales are the latest, but there will be notable changes in ore treatment at Broken Hill within a short time. A summary of the output of Broken Hill and the dividends paid will emphasise the richness of this unique metalliferous lode.

According to Coghlan, the quantity of silver and silver-lead produced in New South Wales to 1902 was—

Silver, 11,088,554oz., valued at £1,718,345

Silver-lead (ore), 3,374,369 tons 10cwt., (metal) 444,379 tons 10cwt., value £32,063,411.

Total value, £33,781,756.

The aggregate output of the mines in the Barrier country to the end of 1902 was valued at £30,945,073, thus leaving £2,836,683 the value of the output of the other mines in the State.

The Broken Hill Proprietary from the commencement of mining operations in 1885 to the end of May, 1903, treated 6,544,468 tons of silver and silver lead ores, producing 119,564,327oz. of silver and 598,835 tons of lead, valued on the London market at £25,688,000. Dividends and bonuses to the amount of £7,592,000 have been paid, besides £2,320,000, the nominal value of the shares from the several blocks. To these figures may be added the output and bonuses for the past half-year to the end of December.

British Broken Hill.—Dividends to 1900, £127,500. The accounts to the 31st December, 1901, showed a loss of £11,270 on the half-year, reducing the former credit to £12,474. Mill shut down since July, 1901, owing to the low price of the metals. At June 30th, 1902, the credit balance was reduced to £4705

Broken Hill Junction M. Co.—Second reconstruction of the Broken Hill Junction Silver M. Co.—Dividends and bonuses (3d.) paid by the former company, 17s. 3d., representing £86,250. This was on a capital of £100,000 in 100,000 shares. The present capital is £100,000 in 200,000 of 10s. each.

Broken Hill Proprietary Block 10 Co., Ltd.—Capital, £1,000,000 in 100,000 of £10 each. Dividends and bonuses to May, 1901, £9 4s. per share, amounting to £920,000. No dividends declared since. Reserve fund to September, 1902, £40,000. Credit balance to same date, £17,809.

Broken Hill Proprietary Block 14.—Dividends to March, 1901 £3 6s. per share, amounting to £330,000.

North Broken Hill, £17,500.

Broken Hill Junction North, old company.—Dividends, 4s., representing £26,000; present company, no dividends.

New Australian Broken Hill Consols, Ltd.—Dividends, 2s. per share, amounting to £59,640 paid in 1891 by the old company.

Broken Hill South Silver Mining Co.—Dividends to May, 1904, 21s. 6d. per share, amounting to £260,000.

SUMMARY OF DIVIDENDS AND BONUSES.

	£
B.H. Proprietary...	7,592,000
B.H. Blocks shares	2,320,000
B.H. Prop. Block 10	920,000
B.H. Prop. Block 14	330,000
B.H. South	260,000
British B.H.	127,500
B.H. Junction	85,000
New Australian B.H. Consols...	59,640
B.H. Junction North...	26,000
North B.H.	17,500
	£11,737,640

The opportunity may be here taken to make the following corrections:—On page 12 there is a reference to the Sullivan Diamond Drilling Company. This should read the Goldfields Diamond Drilling Co., agents for the Sullivan drills. On page 63, the cost of roasting is given as from 7s. to 9d. per ton, instead of 7s. to 9s. per ton. On page 416, lead in Block 10 tailings 5. not .5 per cent. On page 423 the dimensions given of the Broken Hill Proprietary Co.'s smelting works stack at Port Pirie should be 21 feet inside diameter, not 41 feet.

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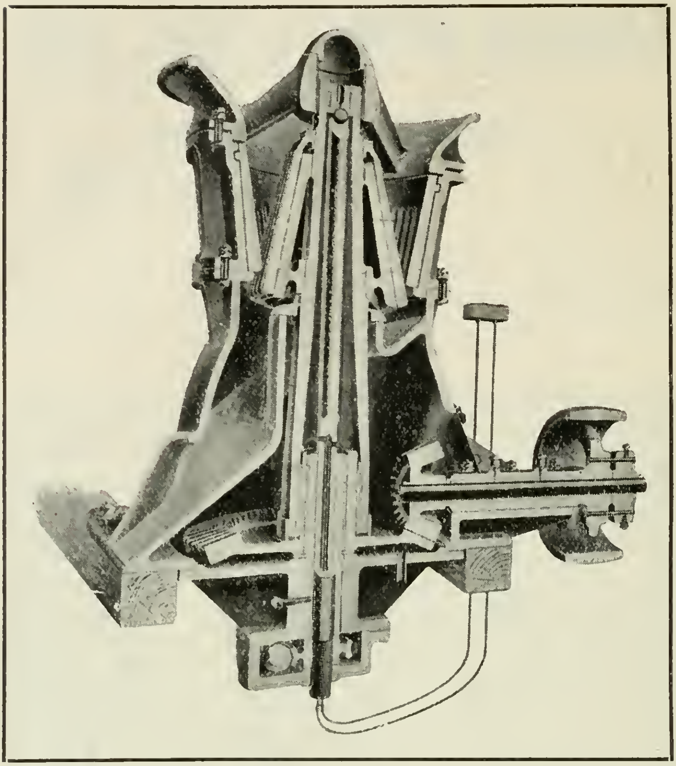

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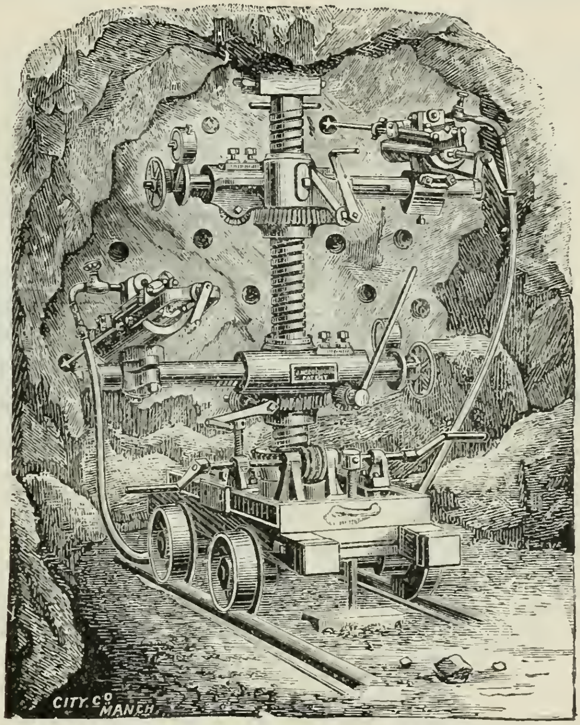

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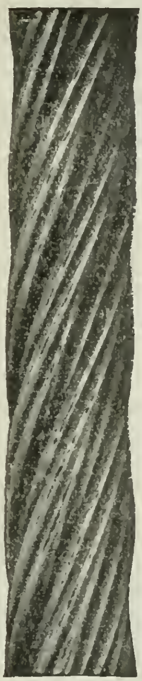

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