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OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

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VOL. X.

OCTOBER, 1909.

No. 4.

Proceedings AT Ordinary General Meeting, October 16, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, October 16th, Mr. A. McArthur Johnston (President) in the chair. There were also present:—

63 Members: Dr. J. Moir, Messrs. F. F. Alexander, K. L. Graham, E. J. Laschinger, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, R. G. Bevington, A. F. Crosse, Prof. J. Yates, H. A. Adams, G. S. S. Anderson, F. W. Bentley, E. Blume, W. Broom, W. G. Brunton, J. Chilton, F. W. Cindel, C. A. Corder, J. B. K. Dodds, G. G. Ferris, E. A. Furner, J. H. Harris, B. J. Hastings, Dr. J. McC. Henderson, S. A. Herbert, C. B. Hilliard, H. Howard, A. J. Johnson, J. H. Johnson, J. A. Jones, T. W. Jordan, G. A. Lawson, H. Lea, R. Lindsay, R. MacGregor, J. W. Morris, M. T. Murray, S. Newton, F. B. Ogle, F. D. Phillips, W. S. V. Price, J. Pyles, O. D. Ross, H. A. Scarf, W. Sharp, S. Shlom, T. W. Simmons, J. A. Smith, R. Stokes, C. T. Sydenham, J. A. Taylor, W. A. C. Tayler, A. Thomas, K. Tonnesen, C. Toombs, F. W. Watson and L. J. Wilmoth.

13 Associates and Students: Messrs. J. Cronin, D. Dowling, D. W. Greig, W. J. R. Hunter, A. King, N. N. Newland, F. J. Pooler, C. A. Robinson, J. M. Robinson, S. A. Smith, G. G. Thomas, M. O. Tillard, and W. O. Williams.

14 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the September *Journal*, were confirmed.

NEW MEMBERS.

Messrs. R. G. Bevington and F. F. Alexander were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that

all the candidates for membership had been unanimously elected, as follows:—

ANDERSON, PERCY, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Surveyor.
 BUTLER, WILLIAM MAHONY, Simmer Deep, Ltd., P. O. Box 50, Germiston. Cyanider.
 DOYLE, MICHAEL JOSEPH, Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.
 HESOM, WILLIAM WADE, Sheba G. M. Co., Ltd., Eureka. Amalgamator.
 PIELE, WILLIAM, P. O. Box 138, East Rand. Mining Engineer.
 WALSH, PETER ARTHUR, P. O. Box 2473, Johannesburg. Engineer.
 WILLIAMS, WALTER OWEN, Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg. Cyanide Manager. (*Transfer from Associate Roll.*)

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

BARRETT, SIDNEY STEPHEN, Sheba G. M. Co., Ltd., Eureka, Transvaal. Amalgamator.
 BRIDGERS, MARCUS MILTON, Sheba G. M. Co., Ltd., Eureka, Transvaal. Ore Roaster.
 CLARENCE, JOHN VAUGHAN GERALD, Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg. Mining Engineer. (*Transfer from Student Roll.*)
 PASEA, ARTHUR EDWARD, Sheba G. M. Co., Ltd., Eureka, Transvaal. Vanner Foreman.
 RAMSDEN, ERNEST, Sheba G. M. Co., Ltd., "Bellevue," Eureka, Transvaal. Cyanider.
 WESTWOOD, WILLIAM, 4, Central Road, Fordsburg. Cyanider.
 WOOD, SYDNEY, Sheba G. M. Co., Ltd., Eureka, Transvaal. Amalgamator.

As Students.—

BASS, KENNETH CHRISTIAN, Three Sisters Mine, Barberton. Assay Learner.
 CRAIG, ARTHUR WILLIAM, Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Learner.
 LOTTER, REGINALD GODFREY, Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanide Learner.
 McCAFFERY, CECIL J., Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Learner.
 McNIEL, WILLIAM, Wolluter G. M. Co., Ltd., P. O. Box 1160, Johannesburg. Cyanide Learner.
 ROBINSON, JOHN MARTYN, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Tube Mill Learner.

GENERAL BUSINESS.

THE SOCIETY'S JOURNAL, VOLUME IX.

The Secretary announced that the binding covers for Vol. IX. of the *Journal* were expected that week, and that members who desired to have their copies bound should send them in to him as soon as possible. The price of binding covers was 3s. 6d., but if the *Journal* were sent in to be bound, the inclusive price would be 6s. per volume, and 1s. extra if necessary to post the volume to members on completion.

He had received from a member in Rhodesia, the following:—

A SUGGESTION RELATING TO THE VALUING OF GOLD MINES.

Mr. R. A. Wade (*Member*): In order to simplify all calculations of the number of tons of ore in blocks of ground, and also take the dip of the reef into account as affecting the payability of a block of ground, I wish to suggest the measurement of reef widths vertically instead of at right angles to the dip. By this method to calculate the tonnage in a block of ground, the horizontal area \times the vertical width of the reef is a simpler and more accurate method than the one at present in use. By this method the steeper the stope the greater its width, the assay value remaining the same as before. The question whether a stope is payable or not certainly is affected by its dip in some cases, but this is not considered in the present method of valuing a mine.

LEAVE OF ABSENCE.

The President: I have been fortunate in securing leave of absence, and the Council has given me permission to hand over the affairs of the Society during that time to our Vice-Presidents. I may say that Dr. Moir will take the chair at the next meeting, Mr. Dowling at the meeting following, and Mr. Saner at the third meeting.

THE BARBERTON GOLDFIELD.

By A. RICHARDSON, M.I.M.M. (*Member of Council*).

Geography.—Barberton is situated at the south-eastern extremity of the Kaap Valley at the foot of the Makongwa mountains, which form the northern boundary of Swaziland, and is by rail 283 miles east of Pretoria and 136 miles west of Lourenço Marques, being the terminus of a short branch from the main line at Kaapmuiden. The Kaap Valley is a circular basin with a diameter of roughly 30 miles, the floor of the basin having a mean altitude of 2,500 ft. above sea level,

whilst the almost unbroken rim of mountains rises abruptly in places 3,000 ft. above it. The valley is drained by three rivers, the North Kaap, South Kaap and Queen's, the two last soon uniting and joining the first near Noordkaap Station, proceeding on from there as the Queen's River.

History.—The occurrence of alluvial gold in the Kaap Valley was first reported in 1875, and in 1882 several hundred diggers were at work on the Kantoor and in the creeks in the vicinity, whose numbers, owing to rich finds, rapidly increased until in 1884 there were 2,000 diggers scattered over the De Kaap district, finding gold everywhere and content to leave reef possibilities disregarded. However, some of the more enlightened began to prosecute a search for reefs, with the result that one of the brothers Barber discovered one at the back of what is now the town bearing his name, or as the then Mining Commissioner describes it: "So in February, 1884, in the presence of several diggers, I broke a bottle of gin—champagne being unobtainable—on the rock containing the gold-bearing quartz and named the prospective township Barberton after the discoverers of the reef."* The Umvoti Reef was found the next day, and later on followed the discovery by Edwin Bray of what was truly named "The Golden Quarry," the outcrop of the Sheba Reef. The Sheba Gold Mining Company was floated in 1886 with a nominal capital of £15,000 in £1 shares, and the first crushings returned 8 oz. over the plates, the tailings going another 4 oz.† On the strength of this the shares rose to £120, and anything in the shape of a reef in the district was readily floated by company promoters for capitals varying from £10,000 to £500,000, and shares in many of these "mines," with the grass roots still undisturbed, ran up to twenty or thirty times their nominal value. Disillusionment quickly followed, and the population of the town, which had risen from 1,000 in 1885 to 10,000 in 1886, rapidly dwindled again, many going to the Rand, then just discovered. Although the history of Barberton district may be said to have begun with the diggers of 1882 there were many evidences discovered by them, such as the remains of properly graded roads, mining works, and old smelting pits containing small quantities of smelted gold, suggesting an Arabic civilization of pre-Mahommedan date, and also old workings probably of kafir origin of a few generations back.‡ It is highly probable that the operations of some earlier race, of which we have only the faintest records to-day, may in some measure

* David Mackay Wilson. "Behind the Scenes in the Transvaal."
† T. Reunert. "Diamonds and Gold in South Africa."

account for the comparative poverty of modern South Africa in alluvial auriferous deposits.

Geology.—The district lies in what is known as the Swaziland System, a system that extends from the Pietersburg district round the eastern Transvaal to the north of Vryheid, and over much of Rhodesia and Manicaland. It consists of intrusive biotite-granite, schists, slates, shales, quartzites, sandstones and conglomerates, all traversed by numerous diabase dykes. The basin of the Kaap Valley is made up of this granite, while the upturned edges consist of the other rocks mentioned, the schists being next the granite. All these sedimentary and metamorphic rocks dip at steep angles away from the granite, and have a strike tangential to its periphery. The numerous long serrated ridges which break up the floor of the valley are composed of diabase, the most prominent instance forming the Devil's Knuckles. On the west the series is overlaid unconformably by the flat-lying Kantoor sandstones of the Black Reef series and the lower beds of the Dolomite Series. The strata on the north and south of the valley join together on the east and swing sharply off in that direction to the vicinity of Komati Poort; this bend to the east takes place just west of the Sheba.

The succession of strata is well exemplified along the bridle path from Barberton to Pigg's Peak, and has been worked out by A. L. Hall,* from whose account the following description is taken. Leaving the granite the lower slopes above Barberton begin with vertical, soft, probably igneous, schists, containing a thin calcareous band; then follow very hard highly sheared green quartzites; these and grey or pale red phyllitic beds extend as far as the blockhouse, and after them come softer quartzites to the summit of the first and highest range. The higher portions of this range are formed of dark-coloured quartzites with coarse conglomerates made up of cherty quartzitic pebbles; after these come soft reddish ferruginous shaly beds to the Lomati River, beyond which the second range consists of massive quartzites containing pebbly bands, near the Little Lomati River. The ascent to the "Zig Zag" traverses altered shales and schistose quartzites which change into arenaceous rocks right up to "Heights," about half way to Pigg's Peak. Just beyond "Heights" a small granite intrusion comes in, then a conspicuous banded chert bar, followed by banded ironstones with red jasper bands, then coarse conglomerate at the Devil's Bridge; from here to the hill overlooking Pigg's Peak greyish quartzite with bands of sheared green quartzite predominates, then

banded ironstone and then a ridge of sheared conglomerates and sheared grits. This succession of strata occupies some 25 miles; the dip is to the south-east, and varies from vertical at each end to 70° in the central portions. Mr. Hall mentions a calcareous band, which is noteworthy, as the formation generally is characterised by the great preponderance of silica and the almost entire absence of lime.

A portion of the field, on Moodies, lying about eight miles west of the town has been the scene from time to time of considerable mining activity, and a section, from north to south, across the formation at this point is here given. Fig. I. (see p. 124). After leaving the granite of the Kaap Valley, schists and shales occupying a width of 2½ miles are traversed; these dip 87 deg. south and form the foot hills of the range rising to an elevation of 4,500 ft. above sea level. The talcose argillaceous schists which make up the greater part of this series contain some narrow beds of chloritic schist, and are separated from the overlying shales and slates by a massive quartzite bar, in places 300 yards wide, which can be traced through the country for many miles. The shales and slates have a width of some 600 yards and a vertical dip. Following on this is a belt of quartzitic sandstone about 900 yards wide, bounded on the south by a mass of conglomerates and hard ribbon quartzites some 600 to 900 yards wide, which forms the backbone of the range, and rises not far west of the point of section to an extreme elevation of 5,800 ft. above sea level. This ribbon quartzite is the magnetite-quartzite-slate or "calico-rock," a very characteristic feature of the Swaziland system. South of this is another belt of talcose argillaceous schist, which is the last of the series in which gold has been found in payable quantities, in fact one may say that the succeeding series of sandstones, conglomerates and quartzites, which cover a distance of some four or five miles, are barren.

The granite is seamed with veins of hard white glassy quartz which are rarely auriferous, and though some gold has been taken out of them they may be classified as gash veins of very little economic importance. In the first belt of schists there are several interbedded gold-bearing quartz reefs varying in thickness from a few inches up to several feet, and in colour from pure white through all the shades of grey to black. The gold occurs in irregular patches and lenses more rarely in well defined shoots with a regular dip. This irregularity is characteristic of the district, and adds considerably to the element of chance with which gold mining is there associated. Many of these reefs may be traced for miles, but they only carry gold in short sections, rarely more than 200 ft. in

* The Geology of Mount Maré, near Pietersburg, and its connection with that of Moodies, near Barberton. Trans. Geological Society of South Africa, xii., January to June, 1909.

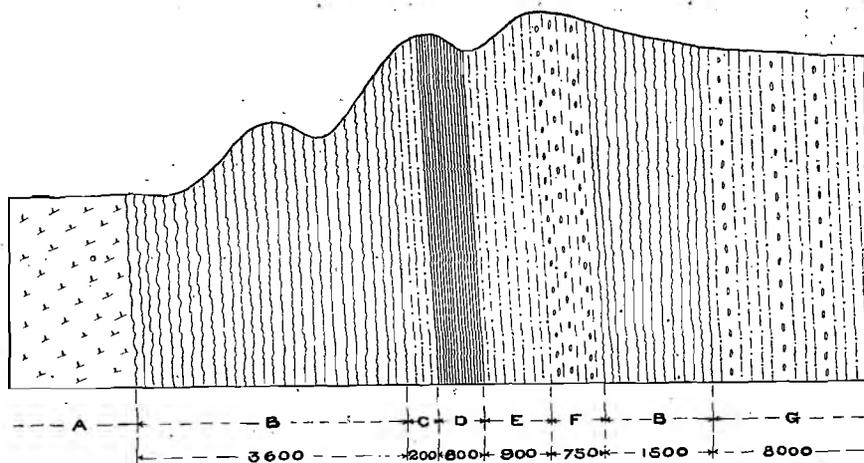


FIG. I.—Section across Moodies.

A.—Granite.
 B.—Schists.
 C.—Quartzite.
 D.—Shales and Slates.

E.—Quartzitic Sandstone.
 F.—Conglomerates and Quartzites.
 G.—Sandstones, Conglomerates and Quartzites.

Figures show widths in yards.

length, so that the reef may be sampled over a considerable length and yet the rich portions missed. This restriction seriously affects development costs, as the poorer portions between the shoots may not be worth milling. The Pioneer Reef is a white, opaque, granular quartz reef bedded in chloritic schist; it varies in width from a mere parting to 6 ft., and is of a very free-milling nature. A noticeable feature of this reef is that where the country is highly metamorphosed the reef splits up into stringers and rarely carries any values. In the shales next to the quartzite bar the principal gold carrier is the Ivy Reef, a dark grey to black quartz reef with the gold occurring in well defined shoots; the width varies from a few inches to two feet. This reef has been exploited to a depth of 900 ft. below the outcrop, and the shoots are found to maintain their regularity and value. The reefs in the next belt consist of veins of quartz in quartzite, which passes by insensible gradations into soft sandstone; they are never of any size, and have not, in this neighbourhood, been worked to a greater depth than 400 ft. below the outcrop: where, however, a mass of stringers has been found in the sandstone the country rock has carried high values and stopes 15 ft. wide with ore going an ounce to the ton have been taken out. These reefs follow the planes of stratification in the quartzite, and although cross leads are found they are not numerous and only occasionally show local enrichment. The belt of conglomerates and ribbon quartzites which lies to the south of this series nowhere in this section carries gold

in payable quantities. From traces up to 2 dwt. have been found in the conglomerate, but nothing to tempt anyone to do any work on it. The first mass of conglomerate is in places 200 ft. thick and composed of large irregular pebbles of chert, jasper, &c., in a coarse matrix. In the second belt of talcose schist, which is about 1,500 yards wide, only one reef of any importance has been found: this is known as the Brighton reef, and has had very little work done on it. It is a reef of grey quartz running from a foot up to several feet in thickness, and carrying from a few dwts. to 3 or 4 ozs. Portions of it are free-milling, but these often give place to large black ferruginous pockets several feet wide and carrying low values.

All this series from the southern schist belt to the granite have been subjected to the intrusion of numerous diabase dykes, and it is noteworthy that these are absent from the barren series of sandstones, quartzites, and conglomerates lying to the south of the second belt of schists. These dykes have a general S.E.—N.W. strike, a vertical dip, and are sometimes as much as 150 ft. thick.

There is a peculiar occurrence of alluvial gold near here, at the head of the valley, above the office of Moodies G. M. & E. Co. The valley terminates in a neck, and just below this there is a diabase intrusion, the result being a shallow basin, roughly, 1,000 ft. in diameter, the bottom of which is filled with a black clay varying in depth from nothing at the sides to 12 ft. in the deepest part. The gold occurs on the bed rock below this clay, and the closest search all over this small area has failed to reveal any source

from which the gold could have been derived. A few miles to the west of this section is the Mount Morgan mine, noted throughout the district for the refractory nature of its ore. The reef here consists of a body of dark quartz stringers generally in black shale, sometimes in grey quartzite. The ore is highly pyritic, containing iron and arsenical pyrites and disseminated graphite.

To the east of the section is the Rosetta Sheba, which is now being actively developed: this, from north to south, shows the following succession:—Shale and schist, white reef, 2 ft. to 17 ft. thick, chloritic schist 20 ft. to 40 ft. thick with white quartz stringers of no value, blue reef 15 in. to about 60 ft. wide, then shale and schist again. The White reef contains copper salts and galena, and the Blue reef bands of galena, copper salts and graphite in films, pockets, and grains, but chiefly on the south wall, in veins from 1 in. to 1 ft. thick. These reefs dip with the strata 80° south. Just to the west of the town lie the Moodies Fortuna and Abbott's, the former working a free-milling grey quartz containing some magnetite, and the latter a white quartz which at 50 ft. from surface shows, in order of importance, copper pyrites, limonite, copper glance, and copper carbonate.

At the back of the Market Square is the Thornton-Little working a reef 2 ft. or 3 ft. wide consisting of ramifications of black quartz in quartzite somewhat resembling that of the Sheba in structure. To the east are the Elephant's Kloof, Kidson, and several others, and beyond them the famous Sheba, 12 miles away. As the Sheba has been milling longer than any other mine in South Africa, and as it is undoubtedly one of the most interesting mines in the world, it merits a somewhat extended description. The accompanying cross-section, Fig. 2, explains the formation and the relative positions of the ore bodies. It will be noticed that there are three beds of quartzite known respectively as the Sheba Bar, the Southern Cross Bar, or Zwaartkopje North Bar, and the Zwaartkopje Bar, and all the rich deposits of the mine are associated with one or other of these quartzite bars: it is also noteworthy that all the reefs lie in a zone defined by a series of diabase dykes crossing the strata in a general N.N.E.—S.S.W. direction. Starting at the north of the section, there are in the shales, slates, and schist, a series of mineralised fracture planes striking in no particular direction and of low grade and erratic nature. These are represented by the Eureka, Mamba and Margaret reefs. The Sheba reef is a crushed or sheared quartzite, the planes of fracture being recemented with secondary white and black quartz, the black being the gold carrier while the white is barren.

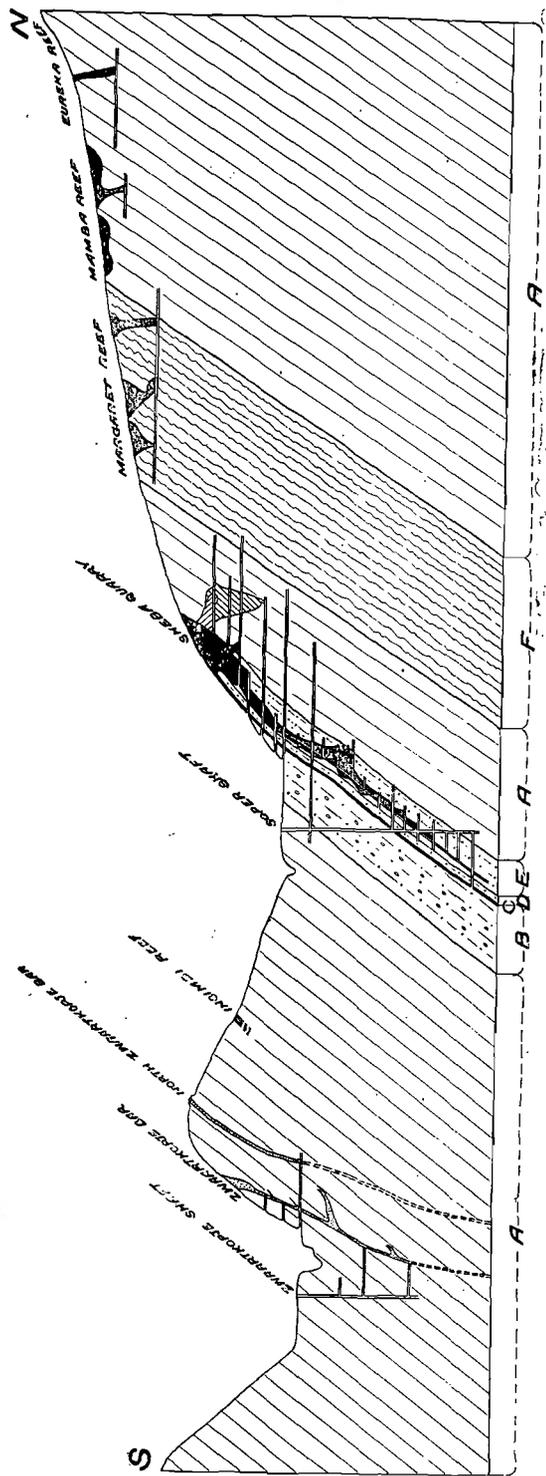


FIG. 2.—Section across the Sheba Mine. Scale 1 in. = 920 ft.
 A.—Shales and Slates. D.—Sheba Bar, Quartzite.
 B.—Sandstones and Conglomerates. E.—Sheba Reef, Quartzite.
 C.—Clay Seam. F.—Sandy Schist.

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The dark colour of this quartz has been attributed to the presence of bituminous particles,* but a more probable explanation is that of total internal reflection from strain surfaces. The presence of quartz of two different colours, one rich in gold and the other barren, is due probably to there being two periods of movement and silicious deposition. In the first the auriferous quartz was deposited, and in the second the newer barren white quartz, while at the same time the auriferous quartz was subjected to considerable pressure, leading to the development in it of a multitude of strain surfaces. A similar phenomenon has been observed in the quartz of the Dharwarian series of the Kolar field and the Hutti mines of India, a series the geological features of which closely resemble those of the Barberton formation.* This quartzite of the Sheba reef is continuous on either side but the gold is only in the shoot. The stoping width reaches as much as 100 ft., but is generally about 30, while the length along the strike below the quarry varies from 250 ft. at the upper to 110 ft. at the lower levels. This shoot has been worked for a distance of over 1,300 ft. on a dip of 55° to the 16th level, below which it is unpayable. The Insimbi reef is a quartz body in a fissure striking to the north-west from the N. Z. K. Bar through the shales and slates. The Zwaartkopje ore body is a quartz spur from the footwall of the Z. K. Bar, and though rich at the bar the values gradually disappear as it penetrates into the underlying strata. North of the Sheba there are numerous cross veins, some of which form rich cross leads in Thomas' bar as at Joe's Luck mine. This is one of the few instances where these cross veins have proved payable.

In the North Kaap and Jamestown districts there are several interesting deposits. The Verdite is working a lenticular reef of steatite which is 14 ft. wide in one place, the hanging-

* M. A. Bordeaux. Etudes sur les Champs Aurifères de Lydenburg. De Kaap et du Charterland, *Annales des Mines*, vol. xi., 1897.

* J. Malcolm Maclaren. Gold: Its Geological Occurrence and Geographical Distribution.

wall being diabase and the footwall talc schist: this reef contains some arsenic, antimony, cobalt and nickel, becoming refractory in places. Its dip is 80° and it strikes with the formation. The Verdite gold mine is several miles from the original discovery of Verdite stone, the name being kept on when the present gold claims were acquired. Verdite stone is one of the monoclinic micas, Fuchsite, its beautiful green colour being due to the presence of oxide of chromium to the extent of from 1.4%. The chief mine in this locality, or at the present time in the whole district, is the Worcester. Here the country rock enclosing the reef is composed of chloritic schist and dips 70° in a N. E. direction. The principal ore body in the mine (see Fig. 3) has been opened up on the 500 ft. level over a length of 850 ft., carrying payable values over practically the whole distance, but not over the full width at all points; the width of the reef at the main shaft on this level is 40 ft., and this gradually narrows down to the S.E., until at a distance of 250 ft. from the shaft it is only 6 in. wide. The width at 150 ft. N.W. of the shaft is 60 ft., and beyond this point at 250 ft. the body splits up, stoping being done on the hanging-wall portion only, the footwall portion being unpayable. The width of the reef at the face of the N. W. drive, which is still being driven, is 20 ft. Although payable rock is found throughout the whole length of the body rich shoots occur extending over a length of 100 to 200 ft. and dipping in a N. W. direction.

The following two typical examples show the distribution of values from wall to wall:—

Footwall.—2' @ 2.0, 2' @ 3.0, 2' @ 1.5, 2' @ 25.0, 2' @ 4.5, 2' @ 1.5, 2' @ 2.5, 2' @ 18.5, 2' @ 22.0, 2' @ 3.5, 2' @ 9.0, 2' @ 13.5, 2' @ 2.0. Hanging-wall.

Footwall.—2' @ 10.5, 2' @ 12.0, 2' @ 16.0, 2' @ 4.5, 2' @ 3.0, 2' @ 3.0, 2' @ 26.0, 2' @ 13.0, 2' @ 1.5, 2' @ 7.0, 2' @ 10.0, 2' @ 3.5, 2' @ 18.0. Hanging-wall.

About 30 miles east of Barberton, at Louw's Creek, is the Three Sisters mine, one of the most beautifully situated as well as one of the most.

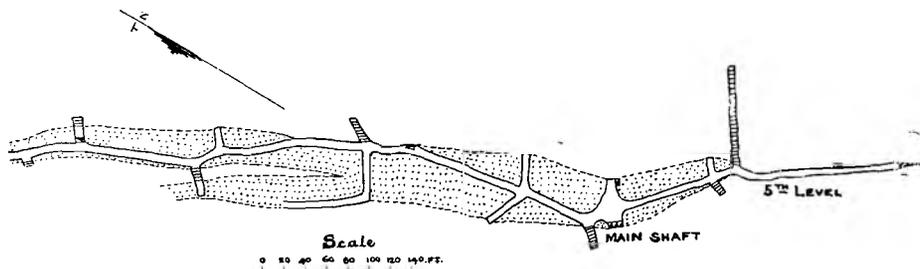


FIG. 111.—Sectional Plan of Ore Body at the 500 ft. Level of the Worcester Mine.

interesting geologically. This mine takes its name from three bold conglomerate mountains which hem it in on one side of the creek, and from the top of any one of which magnificent panoramic views can be obtained. This conglomerate, which has received a very large development at this point, on both sides of the valley, is a highly indurated schistose rock showing few signs of weathering even on the highest summits: the matrix is a coarse grit and the pebbles, which are small, are so severely flattened and crushed as to give it the appearance of a breccia. The reefs at present worked are in a foothill which butts up against the mountains at the head of the valley forming a tongue running down into it, the axis of the hill being roughly N.E.—S.W. A section across it from the northern side gives first green slate dipping 80° S. E., then talcose schist, and finally green slate again. The northern slate contains a reef of which a section is shewn in Fig. 4. This reef

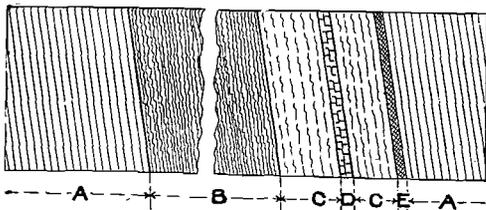


FIG. IV.—Section across N. Slate Reef, Three Sisters Mine.

- A.—Country Rock of Green Slate.
- B.—Chloritic Schist, 30 ft.
- C.—C. Quartz Schist, 2½ ft.
- D.—Honeycombed Quartz, 3 in.
- E.—Limonite, 2 in.

is a quartz schist with numerous quartz laminae (several to the inch), sericite partings and scattered iron pyrites. The talcose schist contains several white quartz reefs and leaders. The reefs, which vary in thickness from a few inches to 16 ft., dip generally 80° S. E., conformably with the foliations of the schist. These reefs contain chlorite, calcium and magnesium carbonate, some silicate, and a little strontium. Crossing these at fairly regular intervals are white quartz leaders 1 in. to 8 in. thick, whose strike is slightly inclined to that of the reefs. These leaders dip 45° N. W. across the formation, and have a peculiar wavy appearance in cross section, the distance from wave to wave being 18 ins. and from trough to crest 12 ins. as a general average; where they cut through the white quartz reefs there is no disturbance or faulting and no local enrichment. In the southern slate is a wide reef consisting of grey quartz with iron pyrites in numerous thin bands which give a lamellar appearance to the rock.

These reefs in the slate are very pyritic and contain a high percentage of silver. The white quartz reefs are free-milling and the leaders particularly so, the bullion obtained from them being very pure.

This slight geological sketch would not be complete without some reference to the mines of Barrett's Berlin and Coetzèestroom on the Kantoor, near Kaapsche Hoop. These are quasi-reef deposits consisting of irregular quartz veins, bunches and stringers, in a soft rock which is the result of the decomposition of the lower beds of dolomite. Sometimes the quartz carries the gold and sometimes the rock above and below to the extent of a few feet. The workings of the former mine are 30 feet above the Kantoor sandstone, on which the dolomite rests, and the workings of the latter are 80 above the same horizon.* Alluvial is still worked to a slight extent on the Kantoor in and a few feet under the surface; this gold originated probably from the disintegration of the lower dolomite beds. The "President" nugget, weighing 187 oz., was found here in 1887 six inches below the surface. Generally speaking the alluvial either here, on Moodies, or at Jamestown, only holds out possibilities of wages; the ancient race that worked this alluvial gold probably skimmed most of the cream off, and the diggers of the 80's cleaned up what was left.

Prospecting.—Although most of the properties on which development was done in the early days are now held or being worked by tributors and small syndicates, the district has by no means been adequately prospected, difficulties of transport standing much in the way. During the summer months the grass in the low-lying parts would interfere considerably, as also probably would the rain and the hot relaxing climate, but in the winter months the climate is a perfect one, and the grass fires lay the formation bare. Being a mountainous district with extensive foothills and ravines good sections and exposures can generally be got, and although plenty of chipping and scratching has been done everywhere there is still scope for the man with a good eye for a reef. In prospecting, as indeed in much other work, the ignoring of the obvious has led to the disregard of many significant indications. This has been exemplified in all mining countries, so a few Barberton illustrations will suffice. In the Sheba section (see Fig. 2) the rich portion of the outcrop of the Zwaartkopje Bar, which when found showed nuggety gold freely and stood out above the surrounding surface, and from which a lot of 100 tons went 23 oz. to the ton, lay for nearly ten years

*A. L. Hall, Transvaal Mines Department: Report of the Geological Survey for the year 1905.

with the Sheba battery crushing close to it just over the other side of the hill. Another case was that of a reef discovered on the Swaziland bridle path close to Barberton in 1896. This reef, which crosses the path at right angles, owing to its superior weathering qualities formed an awkward step for horses and a bit of a scramble for human beings, and yet it was ten years before anyone observed that where it crossed the path it carried "visible" freely. Another case was the finding within the last two years of a reef within a few hundred yards of two established mines and an hotel, which from time to time had been the point of call of numerous prospectors. This was discovered by its present owner who while in the course of a Sunday afternoon stroll stopped to admire the scenery, a commendable proceeding in this hilly country, and kicked a projecting knob of quartz off and found it to contain visible gold. One good way of prospecting is to follow up a road party. A reef was discovered recently in this way from which a trial consignment of about 1,500 lbs. yielded 125·35 oz. per ton.

To acquire claims the simplest way is first to consult the Government maps, which are compiled by the Surveyor-General and the Register of Mining Rights: these can be bought, the price ranging from 3s. 6d. to 10s. 6d., or can be consulted at any Government Mining Office; then to find out whether the particular piece of ground is open for pegging or not consult the compilation plan in the Beacon Inspector's office. The next thing is to expend 5s. in a prospectors' permit, which enables one to peg an area not exceeding 2,000 ft. by 2,000 ft.: having pegged the area there is no further fee and three months grace is allowed in which to start work. The work done must be sufficient to satisfy the Mining Commissioner, but exactly what constitutes "work" in this connection has never yet been clearly defined. An old prospector told the writer that two boys kept fossicking and visited once a week would probably meet the case, but some official statement on this point is clearly needed. In the Canadian North-West a somewhat similar performance costs \$5, and \$100 has to be either paid or expended each year; when \$500 has been expended or paid the locator may, on having his ground surveyed and upon complying with other requirements, purchase the land at \$1 per acre, paying a royalty of 2½% on sales.

If claims are held under the ordinary 2s. 6d. per claim per month system then it is worth noting that 15 claims, or under, can be exempted from any licence if worked by the owner personally, and to the satisfaction of the Mining Commissioner. If the prospector has any doubt as to the correct procedure in any particular case

he need have no hesitation whatever in consulting the Government officials, as they will be only too pleased to put him right. There are plenty of amateur Gold Law experts in the district, but it is much the quickest and cheapest way to go straight to the fountain head.

As the name Moodies often occurs in discussing mining matters relating to Barberton, some information regarding it may be of service: the company publishes no printed schedule, but the following, which is subject to the provisions of the Gold Law, may be taken as fairly correct. Moodies G. M. & E. Co. has full mineral rights over two farms, covering 7,487 morgen and three mynpachts. Since the advent of the small syndicate, terms have been altered and each individual case is treated on its merits. The usual working basis in the case of prospectors is to charge £1 per month for an option over an area equal to a claim area of from 6 to 12 claims. The ground has to be continuously prospected to the Company's satisfaction and the period of option is for six months; this is sometimes, if reasonable cause can be shown, extended for a further period of six months. At the end of this period the prospector is called upon to peg such ground as he requires and pay £1 per claim per month. With ground on which development work has already been done, no cash payment is asked for such work, but an additional 2½% royalty on gold won is charged for all *useful* development, based on a valuation of 10s. per foot run. Here again in case the prospector decides to put up a mill to work certain of his claims but is uncertain about the remainder, the Company may protect these for him for a definite period, so that he may reap the full benefit of his work. A royalty of 2½% is charged on the gross value of the gold won. A mill site is granted with the right to use water for reduction purposes, a charge of £5 per annum being made for this: the right to use water for power is also given, subject to the risk of curtailment, should it be required for reduction purposes. A charge for firewood at the rate of 5s. per month per white habitation is made. Timber, if required, is supplied by the Company at the rate of 1s. per stick 8 ft. long × 6 in. at butt, other sizes *pro rata*: these have to be cut in the Company's gum plantations and transported by the user at his own cost: the timber in the natural bush is not allowed to be cut in order to conserve the water supply as much as possible. Electric power is supplied at a charge of £30 per b.h.p. on motor shaft, the Company erecting and maintaining the high tension side of the plant up to and including transformers; the consumer supplying and maintaining low tension leads, motor switchboard, etc. In the case of a consumer of a small block of power a guarantee has to be given

that power will be used for a definite period, long enough to cover the outlay involved in the erection of a branch line to his plant. The Company's plant runs 365 days in the year and power has to be used continuously, no allowance being made for a breakdown, or stoppage, on the low tension side of plant.

Mining.—The mountainous nature of the country, furrowed by deep ravines, enables the reefs to be developed by means of adit levels driven on or to them at considerable depths below the outcrop, which obviates hoisting and pumping expenses and enables the rock to be gravitated to the reduction works with practically no handling. Where the reef cannot be opened from the surface by levels on it cross-cuts are put in from the face of the hill. This method has, however, been greatly overdone in the past: in one case a cross-cut 800 ft. long was put in, which gave only 80 ft. of backs. The excessively high cost of power in the early days doubtless was the cause of this, anything being done to avoid hauling and pumping expenses. Shafts are not common, their sinking and equipment being, as a rule, a little bit beyond the small man; but the increased depth of mining has necessitated them at some of the older companies and at those which are situated in flat or gently undulating country. The walls stand remarkably well, little or no timber being ever required in the stopes, and heavy ground is usually only encountered near the mouths of adits or in the somewhat treacherous talcose schist of the low-lying Jamestown district.

All methods of stoping are employed, the governing factor being the run of the gold which, in most cases, is so erratic as to preclude the possibility of laying out development beforehand with any degree of certainty. With these steeply dipping reefs levels 120 ft. apart can be carried without any other trouble than a little water in the winzes as they get near the next level, the practice of rising to connect being rarely followed. Ventilation gives little trouble, though occasionally home-made centrifugal fans worked by hand are installed in unconnected levels. All the mining work is done by hand labour: the rock drills at the Sheba have been put out of commission and there is only one running on development at the Worcester. The following description in conjunction with Figs. 3, 5 and 6 will explain the methods of development and stoping followed at the Worcester.

The drives are usually driven along the richest portion of the reef and cross-cuts about 150 ft. apart are put in from the drives from wall to wall, to prove the width and value of the ore body. The levels are arranged to give backs of about 100 ft., and vertical winzes are sunk in the ore body about 200 ft. apart. The reef is mined by the underhand system, and the levels are protected by leaving a rib of ground 6 ft. above and 10 ft. below them. The hanging-wall is strong, and requires very little protection, although pillars varying in size from 8 ft. to 20 ft. square, according to the width of the stope, are left where considered necessary, generally midway between levels and 50 ft. to 60 ft.

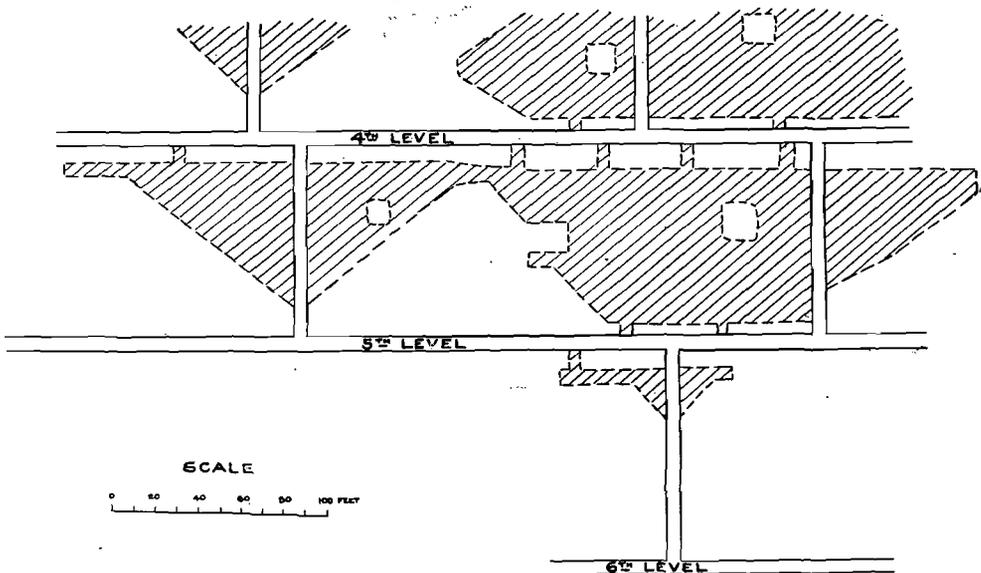


FIG. V.—Longitudinal Section, showing method of Stoping at the Worcester Mine.

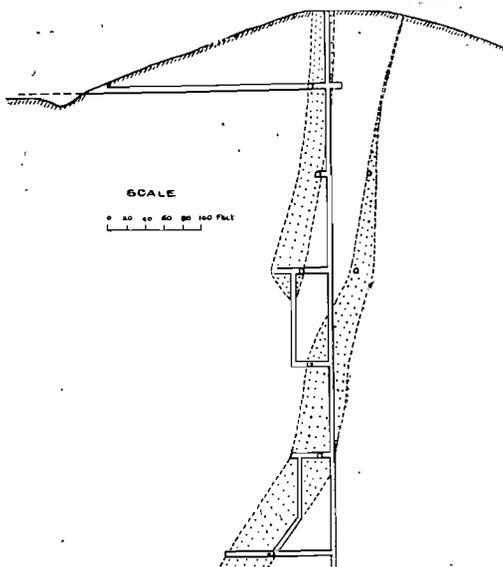


FIG. VI.—Cross Section of Ore Bodies at the Worcester Mine.

apart. Where the main shaft passes through the reef it is protected by a rib of ground of 20 ft. on each side. The reef is harder than that of the Rand, but it breaks better and requires less explosive. All roof holes are put in by double hammer, as it is impossible to drill them single handed. Holes 3 ft. deep constitute a native shift, but a few of the best boys put in holes up to 4 ft. 6 in. 5 ft. holes were tried in the stopes, double hammer, but the numerous fracture planes causes so many fast holes that the practice had to be abandoned. The amount of gelignite used in stoping is on the average 25 cases per month, or slightly under 2 cartridges per ton of ore broken. In development, blasting gelatine is consumed at the rate of one case to every 15 ft. of advance. The average amount of ore broken per hammer boy per shift is $1\frac{3}{4}$ tons. As there are several connections ventilation is good. The quantity of water amounts to 100,000 gallons per day, and is handled by an 8 in. Cornish pump in two lifts.

Transport.—This will always be one of the minor difficulties in the district, owing to the rugged nature of the country. All the mines on the main range make use of the humble but effective donkey to carry stores from the foot of the hills. The first cost in this case is practically the only one, as the donkey feeds itself and is almost immune from practically all the diseases that affect stock in South Africa. To get heavy machinery to the top of the hills the following method is adopted. It is brought by waggon to the nearest available point at the foot of a ridge

whose surface is as free as possible of boulders and rock outcrops, this being a point of more importance than the grade. The front wheels are then taken off the waggon and a rough body bolted on to their axle with two stout sharpened poles as trailers. The machinery is then securely fastened to the frame and, if sufficient natives are available, is pulled direct to the top of the mountain. If there are not enough natives to pull direct, strong poles are fixed at intervals up the ridge and block and tackle employed. Loads up to 4 tons are easily taken up 1,000 ft. of mountain in a day by this method. If the load is moving along the edge of a ridge, side ropes are held by natives to prevent it capsizing, and should any unforeseen accident occur, such as the breaking of a rope, the pointed trailers take up the load and prevent the outfit from careering down into the valley again.

Reduction.—The small high grade reefs of the district with ore easily distinguishable from the country rock lend themselves readily to sorting, which is done to the extent of 80% in extreme cases. The majority of the reefs are free milling, such mines as the Consort, Mount Morgan, Rosetta, etc., being exceptions, but still it may be said that Barberton ores are refractory when those of the Rand are taken as a standard for comparison. In the more refractory mines plates are dispensed with altogether, practically no gold being recoverable on them, while with others the plate recovery may run up to as much as 80%, indeed, in one instance, the Zwartkopjes ore body at the surface, the plate extraction reached 90%; such high extractions are, however, rare.

The fluctuations in recovery by cyanidation are also considerable, and may run from 20% to 80% as a rough estimate.

The mills now running vary in style from the humble 1-stamp of the Tiger Trap, driven by a home-made water wheel to the 40-stamp turbine driven equipment of the Worcester, and as the cyanide plants also show similar differences, all that can be attempted is a description of one or two plants.

The Worcester mill consists of 40 stamps of 1,250 lb., running 96 drops per minute of 8 in.; 900 mesh screening is used, and the duty is 3.75 tons. No amalgamation difficulties are experienced, and the plate recovery is 45% of the screen assay. The cyanide plant consists of 6 circular steel treatment and 3 collecting sands vats, each of 180 tons capacity. The treatment follows closely that generally adopted on the Rand, and occupies from 7 to 8 days. The extraction is 75% of charge value. The slimes vats are 6 in number, of 30 ft. diameter, 8 ft. deep, with a 2 ft. cone. Two are used as collectors, two as treatment vats, one as a residue

vat, and one as a spare solution sump. The extraction is at present 75%, and it is anticipated that this will be improved to 80% shortly. The total extraction from all sources is between 80% and 85%. The bulk of the slimes is exceptionally heavy and thick, and there is also a very fine slime (probably talc) which is most difficult to settle. The Adair-Usher process is used. This fine slime also comes down with the sands solutions and passes through the precipitation boxes where, when the ore from the mine happens to be at all acid, it coats the zinc and seriously interferes with precipitation, though with fresh ore it appears to exercise very little deterrent effect. Other mines in the district, as well as in Rhodesia, have also had trouble from the same cause.

The Thornton-Little may be quoted as representative of many of the smaller plants. This has a battery of 5 stamps of 750 lb. weight, running 96 drops of 8½ in., the screen being 900, and the duty 2.5 tons. As is generally the custom with small plants, the ore is fed into the mortar box by hand, a practice that is not conducive to a high duty, as the coloured gentleman who does the feeding likes to fill the box up well so that he may have time for thought before the effort need be repeated. The sands are treated in wooden vats, and the slimes are collected, dried and mixed with them. The plate extraction is 50% and the cyanide 46%, the total from all sources being 75%. In this battery the daily addition of sodium amalgam in the mortar box has proved very beneficial.

The Sheba possesses a fine plant of 120 stamps, frue vanners, roasting furnaces, and cyanide and slimes plant, but as this company has fallen on evil days, which is to be sincerely regretted, and is in a state of transition, present day figures are not of much value. The returns for the year ending June 30, 1908, show 80 stamps running, and 90,650 tons crushed of an average value of 6.518 dwts. The charge values were, concentrates 63.61 dwt., sands 2.76 dwt., and slimes 3.62 dwt. The plate extraction was about 40%, concentrates (roasted and cyanided) 84%, sands 57%, slimes 66%, the total extraction from all sources being 74%. The ore contains iron pyrites, a little arsenical pyrites, and some oxide of tin.

Settling tanks are generally used on the smaller plants and spitskasten very seldom, and the treatment vats are either of wood or corrugated iron. The latter construction stands well up to a capacity of 50 tons of sand; beyond that they are apt to collapse concertina fashion even though buried up to the rim in sand. The ore makes a good deal of slime, but slimes plants are beyond the means of the small man; in a

few cases Scoular or Wilfley tables are used for concentrating. The method of discharge is by shovelling over the side, preferably into the bed of a creek. In some instances the plate extraction is so high that cyaniding is somewhat in the nature of a superfluity, and in some others the old hands fight shy of such a mysterious agency, relying on the simple formula of "put your mill up and shove your ore through it," to accomplish the desired result. Refractory arsenical and other sands give good extractions if subjected to years of weathering, but current sands of this nature decidedly do not, and no process of simple application seems to have been yet devised to overcome the difficulty. Roasting would be one solution, but to erect the necessary plant would mean a considerable expenditure, which, with fuel costs, would require the resources of a company. In the case of graphitic ores probably a gentle roast would be sufficient to eliminate the occluded hydro-carbon gas in the graphite which—and not the graphite itself—is the real precipitating agent.* This question of refractory ores has not received the attention in this district that it calls for, and the erection of a central reduction works to treat refractory high grade concentrates would be a great boon to the small worker.

Cost of Plant—The cost of plant on a 5-stamp basis, including erection, would be about: mill (second-hand) £150 to £180, suction gas engine (15 h.-p.) £370, cyanide plant and accessories £150. In addition to the foregoing cost, a further £800 to £1,000 will be required in most cases to be put into the mine and to provide for contingencies, but £2,000 may be considered a fair estimate in most cases as capital required to reach the producing stage free from debt. Two cases have been brought to the writer's notice which reached this desirable point on a capital expenditure of under £500, and have paid well ever since: these, however, are quite exceptional. There has been in the past no trouble experienced in buying second-hand 5-stamp mills, requiring only new plates, and there is little likelihood that in the future the supply will run short, as there are in the district more silent than producing properties; moreover, there is a tendency among the more experienced tributors to buy their mills on the Rand because of the heavier stamps obtainable there, and with the new mills going up with 2,000 lb. stamps and big amalgamations and consequent reorganisation a scrapping of out-of-date plant, there is likely to be a good supply available on the Rand without drawing on Barberton at all. For new plant a local firm quotes the following prices:—One 5-stamp mill, 850-lb. stamps, with crusher,

* See this *Journal*, July, 1909, p. 24.

ore feeder, copper plate, A frame, shafting, pulleys, belting and tightener, delivered f.o.r. Barberton, £580; one 5-stamp mill, ditto, but with 1,050-lb. stamps, £610. For driving the 850 lb. stamp mill a suction gas engine, developing 20 b.h.p. at the coast, equivalent to 15 b.h.p. at Barberton, is recommended. One suction gas engine of this size, with magneto electric ignition gear, variable timing device, gas producer, tanks, accessories, pipes and spares, £320 f.o.r. Barberton. For the heavier mill an engine capable of developing 32 b.h.p. at sea level, or 23 b.h.p. at Barberton would be required; this, with the usual accessories as above is quoted £460 f.o.r. Barberton. Another firm quotes for a 3-stamp mill of 750 lb. to 900 lb. falling weight, £200; and for a 3-stamp mill of 250 lb. falling weight, £150. These prices of course do not include the cost of erection; this in the case of a gas engine would mean another £25 to £50. Charcoal costs £4 10s. a ton, and the consumption may be taken as varying from 1 lb. to 1½ lb. per b.h.p. per hour. Mines burning their own charcoal obtain it for 2s. 6d. per bag of 70 lb. weight. Natal anthracite costs £2 5s. a ton f.o.r. Barberton, but is too clinkery and tarry to use alone in the gas producer. A mixture of one-third anthracite and two-thirds charcoal is fairly satisfactory, but charcoal by itself is the best fuel and probably the most economical in the long run.

Working Costs.—The vastly different conditions prohibit an average working cost per ton crushed being struck. With a 5-stamp mill crushing from a large, soft reef and employing 2 whites and 25 natives, a return of 2 dwt. over the plates pays well; while in the case of a high grade, hard, narrow reef, sorting up to 65% of rock mined, crushing 250 tons a month, and employing 5 whites and 100 natives, costs will be quite 37s., or even higher. It may be taken as a fair approximation that a 5-stamp proposition, with the normal labour force of 4 whites and 60 natives and sorting 20% of the rock mined should run at about 18s. per ton crushed. Taking the larger mines, the great variability of conditions also prohibits any attempts at averaging. The Sheba costs for the year ending June 1908, were: mining 6/6·13, development 4/-, tramming, hauling and pumping 2/4·72, milling 3/6·88, vanning and roasting 1/3·82, cyaniding 2/9·76, tailings removal 0/6·29, general 2/5·44 total 23/7·04. The costs of the Worcester for the year ending, 1908, amounted to 13/11½, made up as follows:—

	s.	d.
Mining	5	7·11
Transport of Ore	0	1·14
Milling	2	3·32

Cyaniding	2	5·24
Slimes	0	5·33
Development	1	8·16
General	0	11·42
Head Office	0	5·77

13 11·49

White labour, if no special skill is required, is ample; but skilled men in mining or reduction are not plentiful, and the highly-trained technical man is rarely met with; but it is a mistake to import skilled miners from the Rand for the small mines, as the reef conditions are so totally different and the scale of operations so small that one not used to the district or to a similar class of mining is hopelessly lost for some considerable time. A manager of a small property would receive £40 to £60 a month, and miners and shiftmen £25, the latter usually working 10, sometimes 12, hours. The Worcester works an 8-hour shift underground and a 9½-hour shift on the surface: all stoping is done on the morning shift from 7 a.m. to 3 p.m.: stopers receive 17s. 4d. per shift, trammers 15s. 6d., and developers 35s. per foot on contract, contractors being charged 60s. per case for gelatine, and 2s. 3d. per shift for native labour.

In estimating the cost of stores, it may be taken as a basis that prices are from 15% to 20% higher than those current on the Rand.

Native Labour.—This is nearly all voluntary labour and, as a rule, fairly plentiful. The natives live in huts erected by themselves, and seem a very contented lot. Hammer boys get from 30s. to 50s. a month according to their efficiency; surface boys and trammers 30s. to 40s. The cost of feeding is 10s. per month per boy, in addition to the above. Engagements are generally monthly. Their rations consist of mealie meal, sweet potatoes, and meat on Sunday, an occasional beer drink being added. Swazis are not particularly good mine boys, partly owing to the intermittent periods during which they work, brought about by the counter attractions of home life across the border; there is, however, a sprinkling of coast boys, who leaven the mass somewhat. The average all-round rate of pay may be taken at 40s. per month. In drilling in hard quartz only expert boys can drill two 2-ft. holes on a shift.

Power.—While there is enough water for reduction purposes only rarely is it available for power as well in such cases as the volume is generally small and the head high—700 ft. at the Mount Morgan—the Pelton wheel is used. There is little doubt that the advent of the suction gas engine has made a wonderful difference to the small man, and it may be said to the companies also. At the Worcester the hauling

engine and Cornish pump are worked by steam and the rest by water power. In dry seasons, however, the amount of water is inadequate; consequently a 120 b.h.p. suction gas engine is to be erected to make up the deficiency. Two turbines are in use at this mine, both obtaining their power from the North Kaap River. The larger, which is 400 yards from the mill, receives its water through a race $1\frac{1}{2}$ miles long and has a head of 55 ft. This is a vortex twin discharge machine with a 4 ft. wheel running at 168 revs. and developing 175 h.-p. From this turbine power is transmitted to the mill by means of an endless wire rope of $\frac{3}{4}$ in. diameter, 6/19 construction, running at a speed of 5,000 ft. per min. The distance apart of supports is 300 ft. and the life of the rope four months. The cost of running this turbine is 2·66d. per ton crushed or 5/9 per h.-p. per month. The smaller turbine is a single discharge machine with a 28 in. wheel and develops 75 h.-p. It is situated two miles from the mill and has a head of 45 ft., the water coming through a race $\frac{3}{4}$ miles long; the transmission is by electricity at 500 volts. The Sheba ran its mill with steam power, but there will be sufficient power available at its generating station on the Queen's River to run the reduced plant now operating.

Health Conditions.—It is necessary to discriminate between the town by itself and the district as a whole, as the former enjoys the distinction of being one of the healthiest towns in the world, the white death rate for the last three years averaging only 7·5 per thousand, or half that of the United Kingdom. The prevalence of malaria, however, renders portions of the district unhealthy, and the deaths from this cause form 38% of the total; but even with malaria the general death rate will compare favourably with other countries. The rare occurrence of enteric or pneumonia is a factor of great importance, and miners' phthisis is unknown. In the winter months the climate is delightful, but during the summer is very hot and enervating. The altitude of Barberton is 2,830 ft., and that of the highest point in the district 6,250 ft.; and the rainfall for the last two years averaged 33·49 in. During the period June, 1908 – June, 1909, the highest absolute maximum temperature registered was 96·7° F. in the month of September and the lowest absolute minimum 38·4 in July, the highest average maximum being for the month of December 82·8 and the lowest average minimum 47·3 for July.

General.—The conditions topographical, geological and metallurgical vary so surprisingly in mines situated close to one another that generalisations, such as we are used to on the Rand, are not possible. One noticeable feature is the atmos-

phere of reserve with which the tributor surrounds his operations, an attitude which is a contributory cause to much of the neglect from which the district suffers. Publication of returns is essential if a country wishes to go ahead in small mining, as it is the only way of bringing its potentialities to the notice of the outer world. The Rhodesian Chamber of Mines and the papers of that country publish every month a full return giving district, name of mine, name of operator, number of stamps, tons treated and fine gold won of every mine in the country, small and large, and in a recent one these particulars covered the operations of 219 mines, 138 of which had five stamps or under. This sound policy of publicity has attracted much attention to Rhodesia, and has been largely the cause of the rapid progress of that country from a mining point of view. Again, in Barberton there is no association of tributors, and they are therefore quite unable, as a body, to make representations to the Government on points affecting their operations, a very serious disability especially in times of labour shortage.

Though the irregularity of auriferous deposition and the uncertainty of reef continuance render operations on a large scale generally undesirable, many chances offer for the small syndicate; but even with these latter considerable caution must be exercised if disaster is to be avoided. The chief reasons for the failure of small mines may be conveniently collected under the following headings:—

1. Bad selection of ground.
2. Inefficient methods.
3. Lack of capital.
4. Bad management.

In the selection of ground it is essential that the reef itself as well as the related geological conditions should be studied with extreme care, and that nothing whatever be taken for granted; information given by interested parties, though often reliable, occasionally suffers from a low standard of authenticity, and the ten ounces of the seller are apt to materialize as the two dwts. of the buyer. The breaking qualities of the reef, stability of walls, contour of surface, water supply, transport facilities, and particularly health conditions, are only a few of the many points which must be investigated. To pass a sound judgment on all these requires special qualities in the man who does the investigating a point sometimes overlooked by Rand syndicates, who more often than not are averse to spending money in these preliminaries. It is much better to spend a hundred or two in a thorough examination, and then, if unsuitable, turn the proposition down and cut the loss than to spend only a few pounds and then take the risk of losing thousands.

The manager should be a good all-round man with some business sense, no false pride, and capable of turning his hand to anything. A plant may be hung up for days because the miner is on the "bust," and nobody on the property knows how to blast the holes, or the gas engine may jib, and so on; contretemps like these frequently happen and test the adaptability of the manager. The lack of capital operates principally through shortage of development and inability to keep the mill running full time, and also necessitates the purchase of cheap plant not up to the work it is called upon to do. Inefficient methods are largely a corollary of the above; but with these ores metallurgy is of such importance that the application of wrong methods of extraction, may have very serious results.

In concluding, the writer would like to thank Messrs. A. B. Inglis, C. B. Kingston, G. W. Lowdon and R. Wormald, members of the Society, for their kindness in furnishing him with much information about the district, and for helping to make the few weeks he spent there pass so pleasantly.

The President: I am sure we are greatly indebted to Mr. Richardson for his extremely interesting paper. It deals with a district which many of us know well, and those who have spent a few months or even years in the Barberton district will appreciate the facts which the author has brought forward. There are many interesting points in the paper which we will be able to emphasise at a future meeting, and the Society will be indebted to anyone who has anything to say. I should like, therefore, to propose formally a vote of thanks to Mr. Richardson for the interesting paper he has given us to-night.

The vote of thanks was agreed to.

Mr. A. Richardson (Member of Council): I omitted to mention in the paper that I have brought with me to-night for examination specimens of the various reefs mentioned. These are not brought because they may contain visible gold, but on account of their being typical of the ore being put through the mills on the different properties.

THE ASSAY OF CYANIDE SOLUTIONS AND SLIME RESIDUE CARRYING DISSOLVED GOLD.

By A. WHITBY (Member of Council).

At the outset it must appear that I have entered on an exposition of methods which are, to some extent, classical. There is a legitimate argument, however, to explain this late appearance.

At the time (1902) when the first experiments were made, I was imbued with the idea that, simple as the process was, the ground had been sufficiently covered by Virgoe, and I therefore contented myself with a discussion of Virgoe's original process embodying therein an idea garnered from Prof. de Wilde's patented process for the recovery of gold from cyanide solutions.* It will be remembered that de Wilde's patent covered the precipitation of gold from a cyanide solution by means of a cuprous salt as auro-cuprous cyanide. It was, therefore, but a short step to supplement Virgoe's method by adding a reducing agent to complete the precipitation of gold shown to be defective in his process. This should have been sufficient, one would say, to guide assayers to a correct appreciation of my final method. Unfortunately, however, I have recorded instances of sulphate of soda having been used as a reducing agent, of ferri-cyanide instead of ferro-cyanide added as an adjunct, and lastly of solid lumps of cyanide used to strengthen the solution. It may be well to point out, at this stage, what every industrial worker in the sciences never ceases to reiterate—keep to your working conditions till you prove them at fault.

Of late, I have received many enquiries as to description of this method, and perhaps, owing to not having placed the details on record in specific form, have even found it identically described in an oversea journal as a new and original process devised by another chemist.†

These facts have determined the writing of this short note, more especially as the process has now had time to settle into working grooves and, when carried out properly, never fails to give complete satisfaction. I take it, that members will not accuse me of self-advertisement if I desire to be credited in full with the initiation of a method which compares favourably with any other existing one, and during seven years has never been assailed or condemned by any competent authority. May it be noted, however, that originality is not claimed, but merely service-adaptation of other men's ideas.

In describing the process it will be well to detail the reagents ordinarily used:—

1. Solution of copper sulphate (cryst. 15%).
2. Solution of sulphuric acid (1–5).
3. Solution of sodium sulphite (cryst. 10%).

Occasionally used:—

- a. Solution of potassium cyanide (20%).
- b. Solution of potassium ferro-cyanide (10%).
- c. Solution of sodium hydrate (5%).

In practice 20 assay tons of solution or 583·3 c.c. are placed in a capacious flask and 15

* Vide vol. iii, pp. 6–12.

† "Easy Method of Assaying Solutions," Walter Seymour Duprey, *Mining Science*, July 15, 1909. Reproduced from *Mexican Mining Journal*, July, 1909.

to 20 c.c. of No. 1 solution added. When the cyanide strength is much above 2%, more of the copper solution is required. After shaking, 7 or 8 c.c. of No. 2 are added. This produces a marked change in colour with an acid reaction. If the solution, however, from some cause or another was originally very alkaline, it is possible more acid will be required. After again shaking add 20 to 30 c.c. of No. 3 solution and shake vigorously. The mixture should now smell strongly of SO₂. Repeat the shaking occasionally during five to ten minutes, settle, decant on to a large ribbed filter paper, finally transferring the whole of the precipitate to the filter. Rinse the flask once or twice with water on to the filter, drain and transfer filter to a fire-clay crucible. Flux with about two of borax and one of litharge using sufficient reducing agent for a 25 gm. button. See that the entire filter is well covered with the flux fuse, for twenty minutes, and if filter is not entirely burnt off, finish the fusion with a wash of litharge. Ordinary cyanide solutions carry sufficient cyanide and ferro-cyanide, but the operator must be careful to see that there is a slight excess of copper in his filtrate.

In the case of waters, slime solutions or others carrying little or no cyanides or ferro-cyanides, first render faintly alkaline then add sufficient of occasional solution (a) to make up to not more than 1% KCy, five or six drops of (b) and proceed as usual.

Auric chloride solutions, however, do not yield to this treatment, on account probably of the formation of auri-potassic cyanide and difficulties attached to its reduction. There is here a point I have long desired to investigate, but have never had any opportunity. What is the precise province of the sulphite of soda, if not to reduce the gold from some higher state of oxidation? If the process is conducted in stages, *i.e.*, if the filtrate after treatment with the copper and sulphuric acid solutions be precipitated with the sulphite, the residual gold is recoverable from this further precipitate. On the other hand, why is not a solution of auric chloride in cyanide, forming, according to the books, auri-potassic cyanide amenable to the process without prolonged boiling with a reducing agent before treatment?

Slimes Pulps.—It would be invidious on my part to dilate on the history of slimes residue assays on these fields. It is sufficient to say, that under ordinary conditions of drying, the bulk of the dissolved gold fails to make its appearance in the assay. This has puzzled metallurgists and led to considerable controversy. The fact, however, remains.

Coincident with my experiments on solutions I conceived the notion of treating slimes pulps as solutions, adding to them such quantities of the

reagents as were necessary for the ratio of solution to dry slime, bearing in mind that such solution carried but little cyanide and practically no ferro-cyanide. I found that such a pulp could then be dried without loss, and the reagents added were so infinitesimal that they could not influence the weighings. Usually slimes residues were conveyed to me in screw-cap jam jars, and the necessary reagents added, the cap screwed on, the jar well shaken and the pulp afterwards transferred to the drying vessel, either a porcelain dish or enamelled pan. With this treatment the total gold, dissolved and undissolved, was recoverable.

As independent testimony of the efficacy of this method, I will quote from a communication received from an assayer, who was initially a sceptic.

Sample of partially treated slimes pulp in three portions:—

	Assay Value dwts.
No. 1. Treated as above described	... 5.02
No. 2. Dried without treatment	... 3.75
No. 3. Calculated value made up as follows:	
Solution 63.7% A.V. 1.6 dwts.,	
washed residue 2.2 dwts.	... 5.01

This, of course, is by no means typical of current slime residue pulps, but it shows a loss of 45% of the dissolved gold by simple drying.

I scarcely like to say how high I have found the difference in my own practice on .3 to .4 dwt. slimes residues.

Before concluding this short note I wish to affirm that as many as 30 to 40 solutions can be and are, comfortably precipitated, fluxed and cupelled alongside of current work daily in assay offices on the Rand, and the results so far have never been contested. I should also like to refer to my esteemed Mexican confrère who has curiously, and I hope unconsciously, advocated the use of ferro-cyanide as an adjunct to the preparation of his solutions. The addition of ferro-cyanide was originally suggested to me by Mr. Caldecott, and while its efficacy as well as the entire chemistry of the process remains obscure, it is certainly remarkable that another assayer should have so precisely imitated or originated the conditions of a test in constant use on these fields since 1902. In discussing this note, I hope members who have any suggestions to make regarding the chemical reactions involved will come forward, because discussion on these points will surely add to our knowledge of the cyanogen compounds. This is a country of strenuous workers who seldom have time or opportunity for investigations into theory and, as in this case, prefer to leave it alone so long as results are correctly obtainable.

Mr. H. A. White (*Member of Council*): I should like to propose a vote of thanks to Mr. Whitby for this excellent little paper, which deals fully but concisely with matters he has already brought before this Society some time ago.

I would point out that the basis of this method is Prof. de Wilde's method for dealing with cyanide solutions, brought before this Society in 1898 (vol. ii., p. 337). Dr. Loevy, who was then President, described the method as a perfect and complete precipitation for analytic purposes. Later on (vol. ii., p. 407) Mr. J. R. Williams supported this as the result of his own experiments. Since that time I have used this method very largely, but as my procedure was somewhat different from that described by Mr. Whitby, I will briefly describe it. I use nothing but a saturated solution of cuprous chloride in hydrochloric acid and a 5% solution of potassium ferrocyanide. The cuprous chloride may either be bought as such or the solution prepared by saturating strong hydrochloric acid with copper sulphate and reducing in a well stoppered bottle with excess of copper in the form of wire or strips. I do not find it necessary to add further cyanide even to mill water samples.

Suppose a sample of solution leaving the slime boxes is to be assayed, the method is as follows:

A very large beaker is taken, and in it are placed 2,333 c.c.s. (80 A.T.) of solution and a drop of silver nitrate solution, sufficient to part 0.10 dwt. A few drops of solution of the ferrocyanide are then added, followed by sufficient acid cuprous chloride solution to bleach the first formed precipitate of cupric ferrocyanide so that the precipitate remains nearly white when vigorously stirred. Then in order to flocculate this somewhat unsatisfactory white precipitate a further addition of solution of ferrocyanide is made until on stirring the resulting brownish precipitate settles well.

It will be noticed that I use the ferrocyanide as an indicator of complete reduction of the solution and then as a flocculating agent in order to secure rapid but complete filtration. Its use in the latter capacity was mentioned by Mr. G. W. Williams (vol. iv., p. 504), and suggested itself at once to any experimenter who noticed how much more satisfactory was the filtration when ferrocyanides were present, as is usually the case with certain solutions sent for assay.

I agree with Mr. Whitby that an addition of cuprous chloride should be made to samples of slime residue before drying down, and in accordance with Prof. de Wilde's latest solution method (vol. viii., p. 327) only a few drops of a saturated solution are required, and there is no risk of loss of gold from copper in the lead buttons. We made several comparative experiments at the

May Consolidated G. M. Co., and found a difference of 0.05 dwt. in favour of Mr. Whitby's method. This has not, however, I have been informed, the usual experience on at least one other mine, but I always recommend the adoption of this simple precaution which we certainly owe to Mr. Whitby. I call the author's attention to the correspondence in the *May Journal* on this question, as he seems to have overlooked Mr. S. H. Pearce's error in referring to a paper read before this Society by Mr. Rusden (vol. ix., p. 401).

EXPERIMENTS ON THE ASSAY OF ACID WASHES RESULTING FROM THE CYANIDE "CLEAN UP" BY THE USE OF BISULPHATE.

By L. J. WILMOTH (*Member*).

It is a common practice on many mines on the Rand to run the waste acid washes after clean up into large tanks for subsequent treatment with zinc fume for the recovery of their gold values. Mr. Donaldson's paper in the last *Journal* has prompted me to bring forward the results of a few comparative tests recently made on the assay of these acid washes, and I trust that other assayers will come forward with their experiences in the assay of this type of sample, so that we may arrive at a more or less standard method of assay. On these fields with the great uniformity of sample to be dealt with—both routine and bye-product work—there is little excuse for the diversity of methods in vogue at present, and it is time we assayers settled down to some uniformity of method. This diversity of method is probably more marked in our routine work than in anything else, and is to be greatly deplored. A standardisation of our methods would tend to much greater accuracy, which after all is what constitutes our value to this great industry.

This experimental work was done some two or three months ago, and I trust the results will be of some interest to members.

In all cases the amount of solution taken for actual assay was taken by weight and not by measurement, on account of the varying specific gravity of the different samples of solutions. In practice this point is of great importance, as any error at this stage may introduce a considerable error in the results. In most samples of these solutions an amount of about 20 A.T. should be taken, though with the richer ones 10 A.T. will give quite reliable results. The main object of these investigations was to ascertain if the method in use (spoken of here as the standard method)

is accurate, and whether failing accuracy it could be improved upon by any other method.

The standard method is as follows, 20 A.T. of the solution to be assayed is acidified with 7 c.c. of a 10% sulphuric acid solution and then 25 c.c. of a 10% copper sulphate solution is added. The sample is brought to the boil and about 2 gm. of zinc fume is added. As the action starts, and in fact during the whole time of evolution of hydrogen the zinc fume shows a tendency to agglomerate, and care has to be taken that this is prevented by frequent stirring. Agglomeration of the zinc fume has a tendency to interfere with the accuracy of the results. After complete solution of the zinc the precipitated copper is caught on a filter paper which after filtering is fluxed in the usual way having soda in excess in the charge. The proportion of soda to borax is usually 2 : 1. The slags from this fusion are always very coppery, and the cupels after cupellation very heavily stained. Results from this method always agree very well.

The first step in these few tests was a comparison between the standard method and (1) evaporation with litharge, (2) boiling with an alkaline sulphide in an acid solution, (3) a slight modification.

The method No. 2 was as follows, 20 A.T. of sample were taken and acidified with strong hydrochloric acid. The sample was brought to the boil and 30 c.c. of 10% sodium sulphide added. The boiling was then continued for a further 15 minutes. A few cubic centimetres of a 10% solution of lead acetate were now added. This, on coming into contact with the solution, was immediately converted into sulphide, which acted as a collecting agent. The sample was now filtered and the filter paper and its contents fused with a little borax and litharge. The idea of this was to precipitate the gold as a sulphide and also to collect it free from any zinc and copper. Separating out the zinc first in any form would probably be unsatisfactory, as its bulkiness would probably lead to the retention of the gold to a considerable extent. The method No. 3 was merely a modification of the standard method. Thinking the copper in the slag and the lead button might have a tendency to give low results the following modification was tried: After collecting the copper precipitate on the filter paper it was treated with warm dilute nitric acid to dissolve it out. After complete solution of the copper (which is very rapid) the filter paper was run down with a little borax and litharge.

In fusing the residue from the evaporation with litharge a flux consisting of merely litharge and borax was used to demonstrate the effect of unsuitable fluxing. The following is a summary of the results of these four tests:—

Evaporation with litharge	Standard flux unsuitable method	Standard Cu dissolved	Alkaline sulphide
0.63 dwt.	1.55 dwt.	1.53 dwt.	0.90 dwt.

The low results by the evaporation method are accounted for altogether by the unsuitability of the fluxing. The only difficulty to be overcome is the presence of the zinc. Further samples were taken and another test run between the standard method and evaporation with litharge, this time using a flux suitable for the charge to be handled. The proportions of litharge and borax were altered and soda and a small portion of silica added. The results in this case were as follows:—

Standard method.	Evaporation with litharge.
0.26 dwt.	0.23 dwt.

Looking back at these methods it will be seen the results undoubtedly point to the superiority of the standard method over the alkaline sulphide method. The lowness of the results in this case can only be accounted for by the interference of the zinc contents of the solution. The evaporation method with suitable fluxing of the residue gives good results though slightly lower than the standard, here again the zinc may account for the difference. The lowness of the results with modification (3) is difficult to explain but is probably due to solution of gold as well as copper when the latter is being dissolved.

Another modification of the standard method was tried. This consisted of adding a small quantity of sodium sulphide to the solution after complete solution of the zinc, and continuing boiling for a further 10 minutes before filtering when the precipitate was treated in the usual manner. The results, however, showed no improvement.

Standard method.	Standard method and alkaline sulphide.
0.28 dwt.	0.26 dwt.

In all these comparisons the standard method gave the highest results, but the presence of the metallic copper on the filter paper undoubtedly is the weakest part of the process, as from there it passes to the crucible to be fluxed off as well as possible. This method relies upon the evolution of hydrogen by a zinc copper couple, and so a test was tried in which the copper was replaced by lead, thus giving an efficient and at the same an unobjectionable material to handle. The method tried was one described by Chiddy in the *Engineering and Mining Journal* some time back. In this test the standard method was run as usual, and the following was the lead method. 10 A.T. of solution to be assayed were taken and first 10 c.c. of a 10% lead acetate solution were added then 15 c.c. of strong hydrochloric acid. The solution is brought to the boil and 2 gm. of zinc shavings added. The boiling is continued

until all the lead sulphate (due to sulphates present in solution) is decomposed and a clear solution obtained. The precipitate is filtered and fused under borax. There are several little points of importance to be noted. It is essential to decompose all the sulphate of lead present or mechanical losses will ensue, also the acid must be added before the zinc shavings.

The results by these two methods are as follows:—

Standard method.	Chiddy's method.
2·02 dwt.	2·14 dwt.
·18 "	·22 "
·49 "	·45 "
·12 "	·13 "
1·27 "	1·44 "
Av. 0·816 dwt.	0·876 dwt

In the above comparison it will be seen that Chiddy's method gives a result higher by ·06 dwt. In only one of the above tests, which are single assays on different samples does the standard method give the higher result. It would seem that the advantage lies with the use of lead acetate instead of copper sulphate in supplying the material for the couple.

A comparison of the cupels used in the different methods is most striking, and one of the chief causes of the advantage lies in the cleanness of the lead button for cupellation. A comparison of the slags by the two different methods also suggests a further cause for the difference.

A test was run to ascertain the effect of the sediment so often noticed in the sample bottles. A bottle was taken and shaken thoroughly and then a portion taken for assay while the sediment was still in suspension. The bottle was now placed on one side for a couple of hours, and when the sediment had settled a sample of clean solution taken for assay by Chiddy's method. The results were as follows:—

With shaking.	Without shaking.
1·27 dwt.	0·23 dwt.

This illustrates the effect of the sediment on the assay value. In reviewing these few experiments it would appear that to obtain the highest results the sample bottle should be thoroughly shaken up and Chiddy's method of assay used. The use of either the standard method or the evaporation with litharge if carried out properly with care as to the use of a suitable flux will yield good results, but slightly lower. Of these two, the standard method gives the better results as the copper exerts less injurious effects in the crucible than the zinc.

From the reduction works manager's point of view this last test is of importance as it is he who has to put the assay values to their true use either in guiding his work or in working out his

reports. The assayer is merely interested in getting the true assay value of the samples submitted. The question naturally crops up what chances are there that the sample submitted truly represents the material sampled, but that comes outside the scope of this short paper, and is trespassing on the domain of the cyanide works. In any case, it is a question well worth investigating by those interested, and is one that should well repay the time devoted to it.

The President: This paper is exceedingly interesting to the assayers who are present to-night, and we are indebted to Mr. Wilmoth for bringing it forward. I should like to remark on one or two points as I have done a considerable amount of work in this connection. The solution for assay should be a weighed quantity as I have found this run as high as 1·12 in specific gravity. For the last five months, I have been carrying out some tests to check the standard method. During that time our acid solutions have been treated with zinc fume, in each lot assays being taken before and after treatment. According to the assay we ought to have got between 76 oz. and 77 oz. of fine gold, but when we had specially run it down in pots we got 86 oz. of gold bullion, which ran about 900 fine. I reckon that is about as close as anyone can expect to get with this class of solution, especially as the difficulty of sampling is great. I would propose a hearty vote of thanks to Mr. Wilmoth for bringing forward this paper to-night.

Mr. A. F. Crosse (Past-President): I should like to ask one question. In what condition do you think the gold is in these solutions?

Mr. L. J. Wilmoth (Member): I think you will find the bulk of it is in suspension. There is probably very little in the solution. The high specific gravity of the solution makes it very difficult to get it all down.

Mr. A. Whitby (Member of Council): I have much pleasure in seconding the vote of thanks to Mr. Wilmoth for these notes, especially as the subject presents difficulties, and the author has given us the record of work done. I do not in any way wish to criticise at this juncture, but I notice that the comparison instituted between the author's method and Chiddy's appears to show that the latter is to be preferred. On general grounds the comparatively large amount of copper which would necessarily find its way into Mr. Wilmoth's lead buttons, would always tend to give low cupellation results, which will doubtless explain the comparative failure of the so-called standard method. I am quite in agreement with the author on the general ground of standardisation, but the term appears a little unhappy in this case.

Mr. L. J. Wilmoth: I would like to point out that the method which in this paper I call the standard method is not an original one of my own. I believe it originated in the Goldfields Laboratory.

Mr. H. A. White (*Member of Council*): Mr. President, I join issue with both yourself and the author of this paper in regard to the weighing of these solutions. You have just now pointed out that from a certain quantity you have recovered very closely the amount called for by assay; I am, however, fairly safe in saying that you estimated that quantity, not by weighing but by measuring the volume. I consider that, as is usual on these fields, a measured quantity should be taken for assay purposes where solutions are dealt with by volume, and that the assay returns should show the result as value per fluid ton of 32 cub. ft. This method has been in use for ordinary cyanide solutions for a very long time, and disturbing it in the method proposed only provides extra labour without increase of accuracy.

Prof. G. H. Stanley (*Member of Council*): I agree with Mr. White. If actual weight is required, it is an easy matter to get it by taking specific gravity with a hydrometer, but, as he says, it is volume which is always taken.

The President: Mr. White's suggestion tends to alter assay methods, and until this is general I am afraid we cannot accept it. The assayer reports as per ton of 2,000 lb. and not per fluid ton. The calculation of this value to the measured volume is easily accomplished by the cyanide manager.

Mr. F. W. Watson (*Member*): Why does Mr. Wilmoth add alkaline sulphide to a strongly acid solution, if H_2S in the nascent state is required? The author should have stated this, since it seems to me that it would be as efficient to pass H_2S slowly through the solution.

The Secretary read the following reply:—

THE DISTRIBUTION OF THE GOLD PRODUCED ON THE RAND.

(*Read at February Meeting, 1909.*)

By T. KIRKE ROSE, D.Sc., A.R.S.M., M.I.M.M., F.C.S. (*Hon. Member*).

REPLY TO DISCUSSION.

Dr. T. Kirke Rose (*Hon. Member*): I appreciate very much the kind observations made on this paper, but have not much to say in reply to the points that have been raised.

In answer to Mr. Caldecott, (a) I have no opinion to express on this point. (b) The gold

bullion imported into the Royal Mint is for the most part refined by sulphuric acid. (c) I have no information as to the purchasers of gold exported from the United Kingdom. War chests are expensive luxuries, tending to defeat their own objects. There is nothing to show that the new supplies of gold are being hoarded except by private individuals in Egypt and India. They seem to be used generally in the Arts, for currency, and as a basis for currency.

In reply to Mr. Bevington, I see on referring to the tables that the chief changes in the imports of gold bullion into the United Kingdom from the Rest of World were as follows:—

	1905	1906	1907
	oz.	oz.	oz.
Germany ...	150	171,058	367,023
United States ...	6,363	18,576	170,006

These imports no doubt consisted of refined bullion of high grade, although the values are not given in the tables. The amounts are quite enough to account for the improvement in the mean fineness of the imported bullion, and show that the President's (Mr. Bevington) surmise is correct.

THE PILGRIMS REST GOLD FIELDS AND MINING METHODS.

(*Read at March Meeting, 1909.*)

By J. MOYLE-PHILLIPS (*Member*).

REPLY TO DISCUSSION.

Mr. J. Moyle-Phillips (*Member*): See Fig. I, "Vertical Section along the strike," should read "Vertical Section parallel to outcrop." As the arrow indicates that the general dip of the country is W, it is obvious that the section was not taken along the strike. The true bearings of the section are E.N.E. to W.S.W.

As regards Mr. Price-Griffith's contention that the Theta reef is "one well defined interbedded sheet," and not a coalescence of two or more reefs as quoted by me, it would be more convincing if he showed what became of the Bevets reef—which lies above the Theta reef and between it and the shales—when the Theta reef comes into immediate contact with the shales as it does in several places in the Clewer mine.

In quoting three reefs as lying in the dolomites viz., Bevets, Theta and Beta, throughout my paper, I am giving the general idea prevalent in this district, and also suggested by Mr. Hall in his report on the Geological Survey of this district in 1906. My own idea, after three years' observations, is that there is only one or possibly two true reefs, viz., Bevets or Theta or Bevets and Theta. Nowhere in the district as

far as I have been able to ascertain have the so-called three distinct reefs been found in the same locality, and where two of them have been found only one has ever proved payable. Of the two so-called distinct reefs in the Beta mine—Bevets (or Theta) and Beta only, the so-called Beta has been worked, the other being unpayable. Again, at Frankfort, where they have the Bevets and the Theta (or Beta) reefs only, the Bevets reef has proved payable.

I cannot agree with Mr. Price-Griffiths that the term "indicator" as applied to that particular quartz leader lying immediately above the Theta reef in the Clewer, is a misnomer, as, where the reef does not outcrop, in sinking for the reef when one strikes this leader surely it may be called an indicator from the fact that the true reef is in close proximity.

As regards the manner in which the reefs were formed, I gather from the reports of the meetings of the Geological Society that that body is not itself prepared to state in what manner these reefs were formed, hence I fear to step in where they fear to tread.

I am sorry to have to tell Mr. Mance that the particulars he asks for are not available.

I tender my thanks to Messrs. Caldecott, Richardson, Mance and Saner, for their interesting additions to the paper, which have greatly enhanced its value, and I also have pleasure in acknowledging the assistance rendered me by Mr. Hall's report of his Geological Survey of 1906, and although many of his remarks are open to argument yet the report, as a whole, gives a very lucid and descriptive account of the district.

I also tender my thanks to Mr. Leggett-Neale—General Manager—and Mr. R. Sutherland, and members of the Survey Department of the T.G.M.E., Ltd., for their kind assistance, more especially the former, as without his approval and help the paper could not have been written, and, lastly, to Mr. A. Richardson, who has so ably chaperoned the paper through all its various stages, and to whom the idea of writing it is primarily due.

SMALL MINES OF RHODESIA.

(Read at September Meeting, 1908.)

By B. I. COLLINGS (Member).

REPLY TO DISCUSSION.

Mr. B. I. Collings (*Member*): I am afraid Mr. Phillips misunderstood my remarks on the subject of the age of the ancient workings in Rhodesia. I consider that the more recent ones may be 200 years old, or possibly less, but there

is no doubt that many of the ancient workings are very much older than this.

The tables supplied by Mr. Phillips, showing the length of the payable chutes, give results which are typical of the smaller quartz mines of the country.

Mr. Kingston points out that the present day tributor does not work under the same advantages as those who started the movement some four or five years ago. It is a very different business—crushing ore from a developed property to opening up one from grass roots, and paying one's way while developing.

I trust that further Rhodesian papers may be contributed to the Society, as its mines afford innumerable problems of interest.

My thanks are due to Mr. A. Richardson for many suggestions while writing the paper, and for his kindness in reading it before the Society.

STOPE MEASUREMENTS.

(Read at May Meeting, 1909.)

By OTTO S. TONNESEN (Member).

DISCUSSION.

Mr. K. Tonnesen (*Member*): In the *Journal* for July I find that Dr. Moir has offered some criticism on this paper. As you are probably aware, the author of this paper is at the present on a trip to England and the Continent, and his reply can therefore not be expected for some time. I consider that Dr. Moir has been very unfortunate in his criticism, and should therefore like to make a few remarks in connection with it. I regret that I could not do this earlier owing to temporary absence from town.

Anyone reading Dr. Moir's criticism without being familiar with the mathematical problems in question would think that the deductions in the original paper were wrong and misleading. That such is not the case has already been demonstrated by Mr. Coaton, who in his example shows a difference of 0.158 in 278.428 between the method of calculation adopted by the author and that advocated by Dr. Moir, or about 0.057%. But says Dr. Moir, the author should have pointed out that his method was an approximation. The author, no doubt, took it for granted that his audience knew that *any* application of mathematics to practical problems involves approximations. Dr. Moir's *own* method is an approximation, and is based on certain assumptions with regard to the properties of the tape line, which are not mathematically correct.

The assumption used by the author is generally adopted on similar problems in engineering calculations, not because we do not know the theory

of a catenary curve, but because the equation of the parabola offers so much quicker and easier solutions, with an accuracy generally quite sufficient for all practical purposes.

The catenary curve would be the line of equilibrium for a perfectly flexible tape, which is loaded or weighted uniformly along its length, while the same tape would take the shape of a parabola if it was loaded uniformly along the horizontal projection of its length. When we consider how very small the difference between these two conditions is in a stretched tape, we can without further calculations realise that it is immaterial whether one or the other assumption is adopted. There are other conditions which will influence the shape of the stretched tape far more than this; for instance, the tape is never, mathematically speaking, uniform in weight even when new, much less when it gets muddy or wet; further, the tape will be more or less twisted, and the portions of it which stand on edge are less flexible than when it is on the flat. Again, if we want to be hairsplitting we should take into consideration that the tension is not constant but greater towards the end, so that the tape here will stretch more, and consequently weigh less per linear unit.

The author has in his paper on Stope Measurements figured out the area of the segment enclosed by the tape in order to prove how defective the old methods of stope measurement sometimes are. Now it is clear that if there is any appreciable difference at all in this area whether the curve is calculated as a parabola or a catenary, this difference would be in the author's favour as it would increase the area of the segment and consequently increase the errors of stope measurement with offset. If the full drawn line on the figure repre-



FIG. I.

sents a parabola, then a catenary with the same sag would resemble the dotted line. Even if we are not familiar with the geometrical properties of the catenary, we can easily realise this when we remember, as I have stated before, that a tape which takes the shape of a catenary is one which is slightly heavier loaded towards the end than one which would form a parabola. But when we increase the load of a tape at a certain point, we also increase the curvature at the same point.

I shall now deal with a couple of points in Dr. Moir's criticism, where he apparently lays himself open to counter attacks. The author in his paper is dealing with an example, taking the length of the offset line A—B as 316 ft. and the sag, projected on the reef plane, as 2 ft., and he says that the area of the parabolic segment

equals two-thirds of the circumscribed parallelogram. No reader should be in doubt as to which parallelogram this refers, as the sentence is also expressed in figures: $\frac{2}{3} \times 2 \times 316$, the parallelogram therefore must have an area of 2×316 , which is the circumscribed parallelogram I show in dotted lines on the figure.

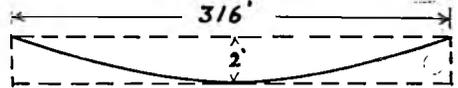


FIG. II.

Dr. Moir seems to think that the author is wrong in stating that the parabolic segment is two-thirds of this parallelogram, and assumes that the author has been mixed up with another mathematical sentence which Dr. Moir brings forward but which, however, has no direct bearing on the point in question.

Dr. Moir has then gone to some trouble in working out the segment area by means of an integration of the catenary as expressed by hyperbolic functions. Had these mathematical deductions been correct, they would not have been without a certain theoretical interest, if nothing else just to show that we have also on the Rand men-capable of dealing with higher mathematical problems. Unfortunately, however, the result appears to be wrong on the face of it, and therefore discourages any attempt to follow Dr. Moir's deductions. Dr. Moir finds the segment area to be:—

$$A = \frac{Sl}{3} + \frac{Sl^2 \sin \alpha}{6a}$$

when $S = \text{sag}$, $l = \text{length of tape}$, and $\alpha = \text{slope of same}$. If we, however, apply this equation to the case that $\alpha = 0$, that is to a tape with its two ends, both held on the same level, we get $\sin \alpha = 0$, and the equation takes the form $A = \frac{Sl}{3}$ or less than a triangle with a base = length of tape and height = sag, which is absurd. Dr. Moir's tape would have to hang something like the dotted line shown on the figure.



FIG. III.

THE EFFECT ON AMALGAMATION, OF DIFFERENT INTERVALS OF TIME BETWEEN THE DRESSINGS OF PLATES.

(Read at June Meeting, 1909.)

By G. O. SMART (Member of Council).

DISCUSSION.

The President: I may say we have adopted this method of twelve hourly dressing on the

Simmer Deep and Jupiter mill, and we find that it answers very well indeed. The plates, after seven weeks' running, are in first class condition, and we are getting the same proportionate quantity of gold as we had in the eight hours' dressing. We have also followed Mr. Smart's lead with the tube mill plates, and have adopted the eight hourly dressing with equally good results.

THE CONTINUOUS COLLECTION OF SAND FOR CYANIDING.

(Read at October Meeting, 1909.)

By W. A. CALDECOTT (Past-President).

DISCUSSION.

Mr. H. A. White (*Member of Council*): Although I notice no extravagant claims, references to "New Metallurgy" or the like, I consider that this able paper chronicles not only a departure, but a distinct advance in Rand metallurgical practice. There are two features upon which the author may be especially congratulated: not only is the plant described actually running upon a working scale, but full details are supplied of costs and results in place of the more usual sanguine anticipations with which we are all familiar. So full and complete a description demands a careful consideration and detailed criticism of the data supplied. Any alteration in ore treatment to win acceptance must show either decreased capital cost, decreased working costs, or increased extraction, and the method advocated seems to display improvement, in all three directions, but the greatest stress is laid upon smaller installation costs. I cannot, however, congratulate the author upon the clearness of the estimates of capital expenditure quoted, and no proper comparison is possible, as the items are differently grouped and moreover mutually inconsistent. For example, the ordinary system quotes 9 items totalling £63,784 for treatment vats, structure, etc., against which I only find one item "treatment vats and structure" in the new scheme, and this amounts to £23,156. Notwithstanding this great difference, it is only claimed on page 51 that 26 vats are required in place of 48, whereas the above proportion would be about 26 to 71. Tracks under vats are not mentioned for the new system nor sumps for the old one. We may no doubt have greater confidence in the difference shown for the two systems which amounts roughly to £30,000, but it would be interesting to know if the method in use on the new plants of the East Rand Mines is not cheaper still. There the difficulty of treating in the collecting vats is evaded so far

that no other treatment vats are provided. It is even possible that if perfectly slime free sand could be collected that the method would be economically sound. With regard to working costs, a figure of roughly one penny per ton is quoted, and I think it should be made quite clear that this refers to total collection cost, and not to extra cost for the new method. If such were the case the reduction in capital cost of £30,000 on 5,350 tons per day would mean the loss in working cost of £8,000 per annum. I have mentioned this point because I know that doubt has been expressed, although it seems quite clear to me that total costs are meant, and even that these are a little lower than with the usual system. With reference to extraction, the Simmer and Jack figures are most satisfactory, and though the actual assay differences are but small the tonnages and the strictly comparative nature of the test warrant the conclusion that more gold can be got in less time and that we are not dealing with a slight bias of the personal equation. Such a conclusion is confirmed by the assay values of the last drainings and an examination of the valuable curve diagrams. The figures below these diagrams show one point of interest which deserves discussion. The double treatment final drainings are steady at 0.08 dwt., and the single treatment come down to 0.20 dwt. only. Yet the dissolved gold, by difference from figures quoted, is practically the same in both cases. I cannot help thinking that a direct estimate of the dissolved gold would show very different results. This is of course quite a side issue, and does not affect the inference Mr. Caldecott has drawn, but it is possible that an apparent discrepancy may conceal some fact of real importance, as was the case in the discovery of argon in the air. A general review of the subject matter of this paper leads me to the conclusion that the success arrived at is mainly dependent upon the practically complete elimination of slime from the sand, and that this matter has not elsewhere in this country received quite all the attention its importance demands. I gather that the moisture in the collected sand will average about 15% and that in the discharged residue about 7% less. If 5,000 tons a day are treated on this basis, there will be an accumulation of weak solution of about 10,000 tons per month, enough to be embarrassing, unless Prof. de Wilde's or some similar method be employed for depriving it of gold and cyanide. Possibly, however, a better result might be obtained on the filter table by using a deeper bed with slower travel or by some other suitable method. I would suggest that further figures are required to demonstrate the gain in treating 8% of fine sand as

slime, and should include account of the dissolved gold carried out by slime residue as compared with that associated with sand, the undissolved gold in the two cases, and increased cost due to extra wear of pumps (if any). In conclusion, I have to congratulate this Society upon a paper so valuable to us and to the world of metallurgy.

The President: I expect that Mr. Caldecott will deal with Mr. White's remarks personally, but I should like to say that during the past week we have been able to run with one vacuum pump for one table, and the result has been very satisfactory, as the 15% which Mr. Caldecott speaks of is rather high. Our present results shows 12% moisture in our sands leaving the table. With regard to the point that some 10,000 tons of solution will be brought into the plant monthly, I can only say that in our plant at the Simmer Deep it has not proved to be the case. A considerable loss of water must occur through evaporation, both of the solution in itself and of that in contact with the sands during transferring. Our last four results from slimes residues gave undissolved gold values at 0.08 dwt., 0.10 dwt., 0.09 dwt., 0.08 dwt. per ton of slimes, thus showing a very satisfactory dissolution of the gold in the slimes.

A METHOD FOR THE RECOVERY OF ZINC FROM SOLUTIONS OF SULPHATE.

(Read at September Meeting, 1909.)

By WM. CULLEN (Past-President) and G. F. AYERS (Associate).

DISCUSSION.

Prof. G. H. Stanley (*Member of Council*): In connection with this paper, I mentioned last month that I had some two years ago deposited zinc from sulphate solution. I had some specimens by me, and I have brought them here for your inspection to-night. With certain strengths of solution and current conditions the zinc comes down in branching crystals, under other conditions in spongy masses more or less as are indicated on the blackboard (illustrated by means of diagrams). Some of the crystals are mounted for microscopic examination and ready for inspection under the microscopes before you, at the close of the meeting.

Mr. A. F. Crosse (*Past-President*): What would it cost to precipitate a ton of zinc? I

think experiments have been made in Germany but it was found to be too expensive.

Prof. G. H. Stanley: After making these experiments and talking to one or two Cyanide Managers, my opinion was something like that of Mr. Crosse. I scarcely think the quantity would repay the cost and trouble, except on very large plants.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution:—

Mr. Thos. Donaldson (*Member*): Reference is made in this paper to the fact that magnesite is used for the precipitation of zinc as zinc hydrate. I would like to mention another purpose to which magnesite has been successfully applied. The contact substance used in the "oleum" plant at the Modderfontein Dynamite Factory for the purpose of converting sulphur dioxide and oxygen into sulphur trioxide, is magnesium sulphate which has been impregnated with platinum chloride, and dried at a high temperature. After being in use for some time this substance loses its power to "convert" on account of its being contaminated with impurities taken up from the gases. The principal impurity is iron, chiefly in the form of sulphate, but lead sulphate is also found, although in small quantity. These impurities are objectionable, and a means for their removal was sought, and found in the use of *magnesia*. In the course of the recovery of impure contact substance, the magnesium sulphate has to be dissolved in water, in order to separate the metallic platinum from it. This solution, which contains also the iron and lead salts, is heated up by steam, and stirred by compressed air, in order to oxidise the ferrous salts. Some calcined magnesite made into an emulsion with water is then added, and the agitation is continued. Sufficient magnesite is used to neutralise any free acid present, and convert the iron into hydrate. Even if insufficiently oxidised beforehand, the iron is soon converted into ferric hydrate, and precipitated, carrying down with it any lead present in solution. With regard to the manner in which the lead is brought down, I would refer any one sufficiently interested to Mr. Wilkie's paper on this subject published in the *Journal of the Society of Chemical Industry*, June 30th, 1909. After the precipitate has been allowed to settle, the solution, free from lead and iron salts, can be decanted off, and the magnesium obtained as crystals by evaporation of the water. The working out of this short process has been chiefly in the hands of my colleague, Mr. Campbell.

THE SMELTING OF GOLD PRECIPITATES AND BULLION WITH OIL FUEL.

(Read at June Meeting, 1909.)

By ARTHUR YATES, M.I.M.M. (Corresponding Member of Council).

DISCUSSION.

Mr. A. Thomas (*Member*): It is always a difficult not to say a hazardous matter to attempt to criticise or discuss a paper dealing with the methods of work and appliances used in another part of the world, such as the one we have been favoured with by Mr. Yates, from the Redjang Lebung Gold and Silver Mine, Sumatra, for so important a part do local conditions play in the estimation of the most economical and efficient means for adoption that what may be very good practice and quite justified in one case may be altogether unsuitable in another. And so it is not my intention to enter into a discussion on the comparative merits of oil as against coal fuel in smelting, and I therefore leave this aspect of the question to those better qualified to express an opinion. However, I can quite understand that oil does possess certain advantages over coal as a fuel as mentioned by the author in respect to the ease and cleanliness of manipulation, and the ready way in which it lends itself to the more perfect control and maintenance of the heat applied. But as the author freely admits, it is solely on account of its relative cheapness in the district of the mine that he was lead to use it in preference to coke and charcoal on which he had previously relied, and the Society has reason to be extremely grateful to the author for bringing to our notice the practicability and expediency of utilising oil fuel for smelting purposes.

However, it is rather on another point raised by the author that I would like to make a few remarks. In reading over his most interesting paper I was somewhat surprised at the short life of the retort he is using, which I think works out at 4.57 charges per retort. I am not quite sure as to the exact nature of the fluxes he is using in smelting the precipitate, but it is evident from the analysis given of the precipitate and retort that its character would be basic, due to the large amount of metallic oxides present, and this would have a tendency to corrode the plumbago retort; and in order to prevent or minimise the risk of this happening I am inclined to think that a clay liner inserted into the retort would amply repay for the extra cost entailed by prolonging the life of the retort. Also, such a liner would have a tendency to eliminate a great deal of the matte which is so troublesome while smelting such pro-

ducts. My experience has been that during the process of smelting gold precipitates in a plumbago pot with a liner in every case when a liner breaks through and the slag coming in contact with the crucible a matte will invariably be found on the bullion button. This fact, which I have repeatedly observed to happen may, it seems to me, be accounted for by the action of the carbon of which the crucible is principally composed on the basic sulphates which on breaking through the liner becomes reduced to sulphides which go to make up the composition of the matte.

I offer this explanation of the possible cause and mode of formation of matte on bullion buttons, and suggest a simple means whereby the chance of its occurrence may be reduced or entirely obviated.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

(Read at May Meeting, 1909.)

By JAS. GRAY, F.I.C. (Member).

DISCUSSION.

Mr. W. R. Dowling (*Vice-President*): At the August meeting* I made the statement that the lime burners protected themselves in the sale of their lime.

The following figures confirm the statement. Nine trucks weighed during July, August and September by the Railway Department, with bags, 203,850 lb. Deducting 3 lb. per bag for 1,098 bags leaves 200,556 lb. as the nett weight of lime. The total weight as per consignment notes was 199,910 lb. The difference of 646 lb. equals about $\frac{1}{3}\%$ and might be considered less than the experimental error.

THE DEVELOPMENT OF HEAVY GRAVITATION STAMPS.

(Reprinted in this Journal, September, 1909, from the Bulletin of the Institution of Mining and Metallurgy, August 12, 1909.)

By W. A. CALDECOTT (Member).

DISCUSSION.

The President: I should strongly urge upon all members who have had any experience in the crushing of ore by these heavier stamps to bring forward their views on the matter. We

* See this Journal, Vol. x., August, 1909, p. 63.

have taken advantage of the generosity of the Institute of Mining and Metallurgy in allowing us to reprint this paper. It has been done not only for the information of members, but to bring forward discussion of the paper by men who are not members of the Institute of Mining and Metallurgy.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THERMAL PROPERTIES OF CUPELS.—"In seeking an explanation of the differences known to exist between bone-ash and patent (magnesite base) cupels when in actual use, the thermal properties of cupels have been investigated by the authors. By means of a special apparatus which is described, the relative diffusivities of heat were determined for various brands of cupel, both at 100° C., and during the actual cupellation process. The patent cupels were found to have a higher diffusivity and a greater specific heat than the bone-ash cupels, and it was noticed that during cupellation at the same muffle temperature, the actual temperature of the cupelling button was much lower in the former than in the latter case. During experiments in the cupellation of silver with lead, considerable differences were also noted in the rate of cooling and in the time required for the solidification of the silver beads. It was thus found that, after being withdrawn from the same muffle temperature, silver beads take longer to solidify and 'spit' on patent than on bone-ash cupels; and that spitting is more liable to occur when the latter kind of cupel is used."—C. O. BANISTER and W. N. STANLEY, *Institute of Mining and Metallurgy*, May 20, 1909. [Advance proof].—*Journal of the Society of Chemical Industry*, June 15, 1909, p. 606. (A. W.)

FUSIBILITY OF MIXTURES OF GOLD AND TELLURIUM.—"Gold is readily soluble in molten tellurium, a pasty solution being obtained if the proportion of gold exceed 60%. The fusion curve indicates the existence of a compound, Au₂Te₄ (43.59% of gold), solidifying at 472° C., but no other compound appears to be formed between these elements. On strongly heating the pasty mixtures, rich in gold, tellurium is driven off and finally, at 1,065° C., pure liquid gold remains."—H. PELABON, *Compt. rend.*, 1909, 148, 1176-1177.—*Journal of the Society of Chemical Industry*, June 15, 1909, p. 606. (A. W.)

CRUCIBLE ASSAYS.—*Directions for Fluxing Charges.*—(1) Examine the material submitted for assay and estimate its approximate mineralogical composition. Then write down the corresponding weight of each mineral in the quantity to be melted (usually 0.5 A.T., called 15 gm.)

(2) *Silica.* Add, if necessary, enough silica to make with that already present in the ore 2 gm of SiO₂ for each gm of CaCO₃, MgCO₃, or BaSO₄ (calcite, dolomite or barite) present, and 1½ gm for each gm of Fe₂O₃, ZnO, clay or other infusible silicate (hematite, zinc mineral, kaolin, etc. Limonite is equivalent to ¾ its weight Fe₂O₃ and

pyrite to ¾.) Consider 1¼ gm of powdered glass equivalent to 1 gm of silica.

(3) *Borax.* Add ½ gm of borax, glass for each gm of silica or infusible silicate (clay or talc) present in the ore and the fluxes. For cupel assays add four gm of borax for each gm of bone ash.

(4) *Soda.* Add 1½ gm sodium bicarbonate for each gm of SiO₂ present in the ore or fluxes. Consider fusible silicates as glass, etc., equivalent to 50 per cent. silica. For pure lead ores add at least ¾ the weight of the ore. For clay, talc, etc., use a mixture of soda and potash.

(5) *Litharge.* Add enough litharge (0–20 gm) to make with the lead already in the ore an 18-gm button and in addition for the slag, 2 gm of PbO for each gm of silica or infusible silicate in the ore. (Consider glass as equivalent to 50 per cent. silica). For cupels add 2 gm of PbO for each gm of bone ash. Increase the amount of PbO largely to keep copper, zinc, antimony, arsenic, and tellurium out of the button if they are present in the ore. For ease in correction for silver in the PbO use litharge in even multiples of 10 gm only.

(6) *Argol or nitre.* Add enough argol or nitre to make an 18-gm button. One gm of Fe₂O₃ may be assumed to prevent the reduction of about 1½ gm of lead, but the reducing power of sulphides should be determined by a preliminary assay if they are present. When more nitre than the weight of the ore is required do not use this method. Roast ores containing much pyrite, antimony or arsenic, adding charcoal for antimony and arsenic. If but little nitre is required use the nitre method in preference to roasting. For galena, etc., nails may be used with charges in which the PbO for the slag is replaced by NaHCO₃ in excess. When nitre is added in the nail method, be careful to form no FeO which will not fuse in the soda slag.

(7) Buttons brittle from antimony or zinc or showing scales of copper must be scorified.

This system of fluxing forms a bisilicate of Na₂O and PbO or Na₂Pb(SiO₃)₂. This is ample flux for a pure quartz gangue and in it the other bases dissolve forming, of course, slags of lower silicate degree. As given, the charge for pure limestone yields a slag of oxygen ratio of 1 in the base to 1.54 in the acid not counting borax. BaSO₄ gives a ratio of 1 to 1.7; MgCO₃, 1 to 1.47; Fe₂O₃ (as FeO), 1 to 1.33; clay, 1 to 1.42; etc.

Effect of the Borax.—Since borax is added to form a borosilicate it is reasonable to figure it according to the amount of SiO₂ present in the slag. The proportion given has not been finally settled upon as the best. It is sufficient to greatly lower the temperature of slag formation, to make the slag viscous during the reduction period, and to lower the temperature of final complete fluidity. It is also sufficient to serve as a general safeguard to flux a slight excess of either acid or base. On the other hand it is expensive and makes the buttons more difficult to clean.

Our charges are a little higher in PbO than usual and for some basic ores it will be more economical to substitute soda for litharge but this makes the charge more bulky. For ease in remembering it was thought better to have a constant ratio of soda and litharge to silica rather than to use the form 1 to 1½ gm of soda and 1.5 to 2 gm of PbO for each gm of silica, which is more nearly what would be used by an experienced assayer.

The amount of fluxes as given by this table are ample and without any great excess. In estimating weights to be fluxed even gm are taken, using care to

see that the weight of refractory mineral is over-estimated rather than under-estimated. The minimum amount of the sodium lead silicate required to dissolve the various bases has not yet been accurately determined. If I ever conduct a class in assaying again, this will be worked out for the more common minerals by making up charges of the pure oxides, etc. It is also desired to work out the amount of excess litharge needed for each gm of copper, antimony, etc., in the charge to make clean buttons. This PbO method has been found to work well with up to 25 per cent. of antimony and 60 per cent. of zinc as sulphides but is not so successful for copper.

Dissolving Action of Litharge.—A large excess of PbO severely attacks the crucibles and the risk of ruining the muffle by spilled charges becomes important. Therefore it is thought best to scoriify the coppery buttons as is done with the buttons obtained in the standard scorification method for copper mattes.

On account of this dissolving action of litharge upon the crucibles more litharge, as given in the directions, is of advantage in assaying clay and talc ores. The charges without excess litharge have very little effect upon Denver crucibles and attack the Battersea crucibles but slightly. The borax seems to protect the crucibles from being drilled by PbO probably by coating them with a viscous aluminium borosilicate.

We have found that by long fusion cupels can be entirely dissolved to a fairly fluid slag by much borax with some PbO and no soda or silica. This forms calcium borate and sodium phosphate as in the tests on platinum wire. The amount of bone ash in the charge is determined by weighing the cupel with the litharge and subtracting the standard weight of that size cupel when new, to give the PbO to be deducted from the weight of the stained part of the cupel. Apology is asked for the incomplete state of these data, but I thought it better to publish this now, rather than postpone it indefinitely in awaiting complete data.—A. A. STEEL, *Engineering and Mining Journal*, June, 1909, p. 1243. (A.R.)

CUPELLATION LOSSES.—“At present there is little evidence as to the comparative variations in the absorption of gold and silver by bone-ash and magnesia respectively. It would be desirable in such experiments to state what bone-ash cupels are used. The chemical composition of commercial bone-ash varies considerably, and the screen analysis varies still more. In the present paper the authors speak vaguely of ‘bone-ash’ without further specification. This is not sufficiently explicit.

The authors quote some measurements of muffle temperatures, but none of them are of much use (I include my own in this sweeping condemnation), for two reasons. In the first place the air current is variable and is in no case specified.

Secondly, the temperature of the air above the cupel cannot be taken by a thermo-couple on account of the radiation of heat from the cupel and from the walls of the muffle. The radiated heat is of course absorbed by the thermo-couple much more than by the air passing through, with the result that the thermo-couple always shows a temperature far above that of the air immediately surrounding it. These are among the causes of the great discrepancies between the different observations of muffle temperature.

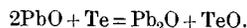
The freezing point of litharge, given by Doeltz and Mostowitsch as 906° and by Mostowitsch alone later

as 884°, is, in truth, a most difficult point to determine. A cooling curve gives no definite point of arrest. There is no doubt that at 906°, litharge is completely molten. At 640°, it is completely solid. Between these temperatures there is a long pasty stage. It needs some courage to announce the exact point above which the pasty litharge is to be called liquid, whilst below the point it is to be called solid.

A little light is thrown on the matter by my own observations that litharge is not absorbed by a bone-ash cupel maintained at 750°, but is absorbed by one kept at 825°; and by Fulton's observations that absorption does not take place at 815°, but is rapid at 883°. If absorption is to be taken as the criterion of liquefaction, then the melting point of litharge must apparently be placed at about 825°. In cupellation, therefore, the cupel must be above this, but the temperature of the air in the muffle near the cupel may be almost anything, and for the reasons given above must remain indeterminate.—DR. T. KIRK ROSE, *Institution of Mining and Metallurgy, Bulletin No. 57*, June, 1909 (H.A.W.)

BEHAVIOUR OF TELLURIUM IN ASSAYING.—The effect of litharge on tellurium was investigated in various ways. Lead buttons containing, lead 6 parts, gold 1 part, tellurium 1.5 parts were fused or soaked under litharge at a temperature of about 800° C. for 15 minutes and then cupelled. The litharge was black, lustrous and opaque. The lead button when cupelled yielded 96% of the gold. Duplicates of these fusions were left in the furnace, and the temperature raised to 1,000 to 1,100° C. for another 15 minutes, and on cupelling the recovery was only 94%. The slag was dark, red and glassy; evidently some of the tellurium had been reduced as indicated by the gold recovery. Again, equal portions of pure lead were fused separately with samples of the opaque and the glassy slags containing equal quantities of tellurium, with the result that the lead from the fusion with the opaque slag showed far more tellurium than the other. This would seem to indicate that a portion of the oxidised tellurium is reduced again when the temperature is raised.

A lead button of the same composition as previously described was dropped into litharge to which a little sand had been added, and which had been kept at a temperature of 1,000 to 1,100° C. for 30 minutes and allowed to remain in the furnace for 10 or 15 minutes more. The weight of the button was unchanged, and it was completely absorbed on cupellation. The slag was straw colour, and gave no indications of tellurium except for a slight reddish cloud, near the button. A repetition of the experiment, except that after the formation of the silicates the temperature was lowered and maintained at a low point after the introduction of the button, gave similar results. After the formation of silicates there is practically very little tendency for the tellurium to oxidise. An examination of the slags seemed to indicate that the reaction between the litharge and tellurium is represented by the equation:



and that the reverse of this reaction takes place at higher temperatures.

The effect of scorification, unless the operation is carried out at a moderately low temperature, is to concentrate the tellurium in the button. The most effective way to remove tellurium is to “soak” the button under litharge either in a scorifier or crucible at low temperature.

The effect of tellurium in a lead button containing gold and silver is to weaken the surface tension of the molten metal during cupellation. This effect of tellurium, assisting the general tendency of the film of litharge in the same direction, results in the tendency of the molten metal to "wet" the sides of the cupel. If the amount of lead is small in proportion to the silver, gold and tellurium, small portions of metal become detached, and either pass into the cupel or leave minute beads.

Lead is more readily oxidisable than tellurium, especially at high temperatures; if sufficient tellurium remains toward the end of the cupellation, complete absorption may take place, or in other words, complete absorption and subdivision are the result of insufficient lead. The tendency to become "wetter" is greater with bone-ash cupels than with patent cupels.

The presence of silver protects gold by acting as a diluent during the cupellation of buttons containing tellurium. The absorption loss due to the presence of tellurium is a function of the temperature of cupellation. An excellent bibliography is given in connection with the paper.—S. W. SMITH, *Inst. Min. and Met.*, Bull. 44.—*Engineering and Mining Journal*, June 12, 1909, p. 1,176. (A.R.)

RECENT DISCOVERIES OF GRAPHITE IN BRITISH AFRICAN COLONIES.—East Africa Protectorate.—Some specimens of graphite gneiss from the right bank of the Tsavo river, five miles below the Tsavo station, on the Uganda railway, associated with carbonate of copper were sent, but examination proved they were of no commercial value.

Uganda.—A specimen from the slopes of Ruwenzori consisted of dull greyish black graphitic schist containing disseminated flakes of graphite. The percentage of carbon amounted to 16.7 only. Another contained 31.2% carbon and 6.4% volatile matter other than water. Another only 32.68% carbon.

Nyassaland.—In the north-west of the upper Shiré district the mineral survey observed seven different parallel bands of graphitic gneiss in which the graphite was interfoliated with other minerals in large flakes. Samples contained 70.6–91.6% carbon, valued if carefully cleaned at £40 per ton. A specimen from a deposit near the Mkoma Mission, in the Angoniland district consisted of practically pure graphite in the form of coarse flakes up to 2 in. in length. It was described as soft and flaky, and classed as ordinary lump of superior quality, worth about £32 to £34 per ton.

Rhodesia.—Samples from the north-east showed 16.85% of ash, mainly felspathic material which would detract seriously from the value of the mineral.

Natal and Northern Nigeria.—Samples from these districts showed no results of economic value.—*Bulletin of the Imperial Institute*, Vol. vii., No. 2, 1909, p. 166. (J. A. W.)

NEW METHOD FOR DETERMINING ANTIMONY.—"A method for the determination of antimony, in which the decomposition of the ore is accomplished with ammonium persulphate, is described by M. F. Coolbaugh and J. O. Betterton (*Western Chemist and Metallurgist*, iv., pp. 229-232). The use of this reagent avoids the danger of loss of antimony by volatilisation during fusion.

The portion taken for analysis is fused with 8 to 10 times its weight of ammonium persulphate for 15 minutes at a low temperature. A small Erlenmeyer flask can be used for the fusion. The flask is placed on an asbestos mat on a hot plate and after the first

decomposition has taken place, the heat is gradually raised until quiet fusion is attained and allowed to remain in that condition for 10 minutes. The fusion is extracted with 20 c.c. of hot hydrochloric acid (sp. gr. 1.10) and then diluted to 100 c.c., heated, and a rapid stream of hydrogen sulphide gas passed through it for 15 minutes.

The precipitate is filtered off and dissolved through the filter paper with 40 c.c. of concentrated hydrochloric acid. A small amount of potassium chlorate is added, the solution heated and any separated sulphur filtered off. Two drops of a ferric chloride solution are added to the solution, which should be at a temperature of 90° C.; stannous chloride is then added drop by drop until the yellow colour of the ferric chloride just disappears. The whole solution is then poured into a beaker containing 10 c.c. of a saturated solution of mercuric chloride, diluted to 500 c.c. to 600 c.c. and titrated with potassium permanganate to the first pink colour. The stannous chloride reduces antimony in a hydrochloric acid solution before it will reduce arsenic. The mercuric chloride takes up the excess of stannous chloride.

The successful use of this method demands a close adherence to the details. The precipitation of the antimony should take place within 15 minutes, if the bulk of the solution is not over 75 c.c., and the acid between 1.1 and 1.15 sp. gr. The potassium chlorate oxidises the arsenic to the higher form. The reduction by stannous chloride is best made with a strong and a weak solution, using the strong solution to accomplish most of the reduction and the weak to finish with.

The amount and strength of the ferric chloride should be known in terms of the permanganate. The reduced solution should be poured immediately into the mercuric chloride to prevent the reduction of the arsenic. Silver, lead, bismuth, arsenic and tin do not interfere. Copper should be removed. The time consumed for a single determination is not more than 1½ hours. The method has given good results in comparison with other well known methods.—*Engineering and Mining Journal*, July 31, 1909, p. 209. (A. R.)

RAPID ANALYSIS OF ALUMINIUM.—"A process for rapid analysis of aluminium was described at the meeting of the French Academy of Sciences, Aug. 2.

The inventor, Mr. Kolm Abrecht, reduces the metal to the form of very fine filings. Then, after placing it in a tube, hydrogen is run through, which expels the air; and, finally, hydrochloric acid, which forms aluminium chloride. The latter is passed into a flask for analysis.—*London Mining Journal*, Aug. 28, 1909, p. 276. (A. R.)

METALLIC TELLURIDES.—"1. Sodium telluride, Na₂Te, is a colourless crystalline salt and is highly hydrated.

2. Tellurium dissolves in sodium telluride solution with the formation of sodium polytelluride. Beyond the proportions represented by Na₄Te₃, tellurium is not soluble in sodium telluride. Na₄Te₃ does not crystallise as such, but breaks down upon evaporation into Na₂Te and tellurium.

3. Potassium telluride, K₂Te does not crystallise from aqueous solution, but is precipitated from strong solution by alcohol.

4. Telluride of zinc is yellow when freshly precipitated and brown when dry. Its composition is ZnTe.H₂O. The anhydrous telluride is dark red in colour.

5. Cadmium telluride, CdTe, is precipitated as a maroon coloured substance, almost black when dry.

6. Nickel forms a telluride of the composition $\text{Ni}_2\text{Te}_3 \cdot 4\text{H}_2\text{O}$, black. When heated in hydrogen it yields NiTe.

7. Cobalt forms a telluride of analogous composition and similar properties to that of nickel.

8. Lead forms the telluride $\text{Pb}_3\text{Te}_5 \cdot 4\text{H}_2\text{O}$, which is decomposed by heat into PbTe, water and tellurium.

9. Ag_2Te is precipitated by sodium telluride from silver acetate solutions.

10. Copper forms the two tellurides CuTe and Cu_2Te_3 by precipitation with Na_2Te and Na_4Te_3 , respectively.

11. A telluride of arsenic is formed by precipitation, and is soluble in excess of Na_2Te_3 solution. This indicates the existence of a telluro salt of arsenic analogous to the sulpho salts.

12. Mercuric telluride is precipitated from mercuric chloride solution, but reacts rapidly with excess of mercuric chloride, forming mercurous chloride and tellurium chloride.

13. Sodium telluride precipitates gold from auric chloride solution and platinum from platinum chloride solution, the tellurium dissolving as the tetrachloride."—C. A. TIBBALS—*American Chemical Society*, Aug., 1909, p. 912. (A. McA. J.)

METALLURGY.

ELECTRO-CHEMICAL PLATE AMALGAMATION.—

"The electro-chemical system of amalgamation is designed to extract from the sands or pulp all gold values not encased. It is entirely a new system of amalgamation, designed to do the work of the ordinary mill plate, including the treatment of slimes, and, at the same time, to extract values which cannot be saved by the usual mill practice, and which otherwise would require cyanide treatment; to do all this in one single, inexpensive and continuous operation, requiring practically no more outlay for installation and maintenance than the usual mill plate. The system also furnishes a method by which the Alaska beach deposits, black sands, hydraulic, placer and dredge tailings, mill tailings, desert deposits and many low grade propositions can all be profitably worked.

"It may be sometimes necessary to grind ore or sands to 100, 150 or even 200 mesh before the values are released, but after this has been done the pulp is simply passed over the amalgamating device, and the electro-chemical forces automatically extract the values to such an extent that usually no further treatment is profitable. In ordinary cases values are recovered at 80 mesh. In certain cases where the ore is unusually refractory, base or complex, it may be necessary to treat the tailings for hour or two by the continuous electro-cyanide process, which is an exceedingly simple and inexpensive operation, but withal very efficient.

"The electrolytic method of amalgamating calls for a sluice box of suitable width, the length being 40 ft. or 50 ft., or even longer, according to conditions. In the bottom of the box is an ordinary copper plate acting as a cathode. Graphite anodes of suitable construction are arranged to conduct the low voltage current into the water (electrolyte) passing over the device. Pulp of proper fineness, screened gravel or sand, is passed into the apparatus by suitable means. Certain solutions are also added to the water. A critical low voltage current of proper amperage is passed into the anodes; the water, carrying pulp, etc., acts as an electrolyte, and makes the electric connection with the copper plate (cathode). In operation,

the water is at all times electrified, and the copper plate is also constantly under electrical action. The device can be made of any desired capacity."

The current density used is about five-tenths ampere per square foot of cathode surface; the pressure at the generator is from 5 to 10 volts. The solutions added to the water contain mercury and sodium salts and some salts from which hydrogen is easily liberated by electrolytic action. The mercury solution is added to the water passing into the apparatus, the amount depending on the values in the pulp. About the same amount of metallic mercury as is used in ordinary amalgam practice is necessary; practically all of it is recovered in the amalgam.

Several chemical, electrical and electro-chemical processes are in constant operation in the amalgamating box, the reactions which would ordinarily occur being greatly intensified by the presence of the electric current. The most important part of the process is the even, constant and uniform deposition of mercury on the cathode plate by the electrolysis of the mercuric solution. The coating of mercury is chemically pure, it adheres tenaciously to the plate and has no tendency to 'flour.' As the mercury is liberated in its nascent state, its affinity for metals and gases is much greater than in ordinary practice; its amalgam-forming powers are intensified to such a degree that rusty, greasy or pyritic gold is readily amalgamated. 'Fouling' or 'sickening' of the plate is impossible.

Fouling of the plates is prevented by the liberation of nascent hydrogen, the cleansing and reducing powers of which are used to great advantage in electrochemical amalgamation. It destroys instantly organic, greasy and foul conditions, counteracts all impurities in the water, and cleans rusty gold. Large quantities of hydrogen are occluded in the mercury as the latter is deposited and exercise continuously a powerful cleaning and reducing effect. Metallic sodium, liberated from the sodium salt, unites with the mercury to form sodium amalgam. The conditions of its formation ensure its purity. With the hydrogen occluded in the mercury it forms hydrogen-sodium amalgam, the most powerful amalgamating agent known. Hydrogen-sodium amalgam besides holding free gold in any condition, will also hold platinum. It will amalgamate all metals not encased, which are in a comminuted state. Amalgamation takes place as soon as a particle of gold touches the quickened mercury surface; the larger particles will come in contact with the mercury very soon after entering the sluice box; microscopic values will require possibly 40, 50 or even 60 ft. of cathode surface. Experiment has demonstrated that electrolytic amalgamation will recover from 95 to 99% of all gold not encased, and in many cases the extraction reaches 100%. Usually when the pulp of refractory ores is ground to 100 to 200 mesh, the tailings, after passing the electric amalgamator, do not contain enough values to make supplementary gold-saving devices necessary."—ELMER E. CARRY, *Mining Journal*, June 12.—*Engineering Magazine*, July, 1909, p. 839. (J. A. W.)

ELECTROLYTIC REFINING OF GOLD.—"The separation of silver and other impurities from gold by electrolysis in a solution of hydrochloric acid was suggested by Charles Watt of Sydney in 1863. He proposed to apply the method at the Sydney Mint, and claimed to have discovered means of preventing the evolution of chlorine from the anode, but no steps were taken to test his process. In 1878, Wohlwill of

Hamburg introduced his process at the Norddeutsche Affinerie, and it has since been applied elsewhere in Germany. It displaced the sulphuric acid process at the Philadelphia Mint in 1902, and at the Denver and San Francisco Mints in subsequent years, and is in process of being installed at the Ottawa Mint. It has made little progress in other countries.

In the Wohlwill process the electrolyte is a solution of gold chloride containing from 25 to 60 gm. of gold per litre, and from 20 to 100 c.c. of concentrated HCl per litre. It is heated to a temperature of about 60 to 70°. The anodes consist of pure gold, containing not more than 60 or 70 parts of silver per 1,000, and the cathodes of pure sheet gold. The anodes dissolve partly as trivalent gold, forming HAuCl_4 , and partly as monovalent gold. The AuCl formed at the anode is partly decomposed into AuCl_3 , and metallic gold, which falls to the bottom, but some of the monovalent gold ions reach the cathodes. The higher the current density the more trivalent gold is dissolved and the less AuCl is formed, so that there is less gold in the slime at the bottom. The current employed in the United States is from 300 to 600 amperes per square metre of anode surface. The deposition of gold at the cathodes is greater than the amount dissolved at the anodes, owing to the presence of impurities in the unrefined gold, so that it is necessary to add chloride of gold to the solution at intervals. Silver is converted into AgCl at the anodes, and falls to the bottom of the vessel, or is in part dissolved. The anodes may contain up to 150 per 1,000 of silver, but if such a large proportion is present it is necessary to scrape the silver chloride from the surface of the plates. Platinum is dissolved with the gold and accumulates in the solution, from which it is recovered in the ordinary way. The deposited gold is nearly pure.

The electrolyte is circulated by mechanical means, and is renewed when it has been made foul by the accumulation in it of copper, palladium, etc. Lead makes the deposited gold impure, but may be rendered insoluble by the addition of sulphuric acid.

Apart from observations on the technical details of this process, the points on which information is particularly required are: (1) the time required to complete the process. This is most important, because of the loss of interest on the gold under treatment. The chief objection to the application of this process in England is the fear that the process is too slow. (2) The cost for labour, power, etc. (3) Whether, if speed is unduly increased, the gold is likely to be of bad quality, contaminated with lead or other elements which would make it brittle.—T. K. ROSE. —*Mining and Scientific Press*, June 26, 1909, p. 890. (A. McA. J.)

THEORY OF THE SETTLEMENT OF SLIME.—“A paper on the settlement of slime, by H. S. Nichols, appeared in the *Mining and Scientific Press*, July 11, 1908. From the study of clay-working slips I can add information from a different view-point. In most respects what I submit is supplementary.

In that article no reference was made to the viscosity of the liquid medium. Viscosity is defined as the measure of the internal friction of a fluid. The viscosity of water is commonly taken at a given temperature as unity. In Fig. A is reproduced graphically the variation of the viscosity of water with the temperature. The viscosity decreases as the temperature rises. As the data given in Landolt and Börnstein's tables extend only to 70° C., it was necessary to extrapolate to get the value for 88° C., the highest temperature used by Mr. Nichols. The

settling curves for various temperatures are practically straight lines, so that the intersections of the

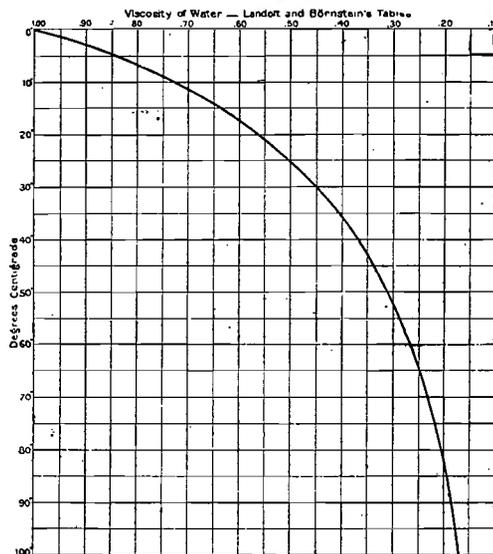


Fig. A.

ordinate for any number of millimetres settled, should be in about the same proportions. Taking the 140-mm. ordinate, the following table has been prepared:—

Temperature, degrees Fahrenheit ...	60	100	140	190
Temperature, degrees Centigrade ...	16	38	60	88
Viscosity of Water from Fig. A ...	62	38.4	26.9	18.4
Viscosity of Water multiplied by 3 ...	186	115.2	80.7	55.2
Minutes settling 140 mm. curves for heated liquids ...	194 ¹	110 ²	82 ³	50 ⁴
Minutes settling 140 mm. curves for heated and cooled liquids ...	185 ⁷	140 ⁶	80 ⁵	

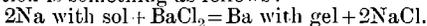
Superior figures indicate numbers of curves.

From this table it appears that, within the limits of experimental error, the rate of settling is directly proportional to the viscosity of the water, whatever the temperature corresponding. According to Fig. 1, 5, and 6 of Mr. Nichols' paper, the higher the percentage of solids in the pulp the slower the rate of settling. In Fig. B, which gives the results of experiments by A. V. Bleininger (*Trans. American Ceramic Society*, 10, 394, 1908), it is shown that the viscosity of a clay-slip increases more and more rapidly as the clay-content increases; consequently the diminution in speed of settling shown in Fig. 1, 5 and 6 of Mr. Nichols' paper can be attributed to increase of viscosity with concentration. As clays differ in viscosity at equal concentration of slips, different slimes should also be expected to vary. Consequently, a treatment adapted to one slime will require its proportions to be altered somewhat to apply to another slime.

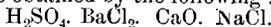
To explain the other figures, it is necessary first to discuss the various conditions of matter that may be present in a slime. The coarser particles are crystal-

loid, and a relatively dense and hard as compared with the finer matter. Upon and surrounding the crystal grains is an enveloping film of matter in the 'gel' form of the colloid condition. It resembles gelatine in properties, absorbs water, and shrinks on drying, becoming hard and cementing the grains together. It is less dense than the crystal grains, and will settle slower than them if separate.

If a large excess of pure water be used, or a proper proportion of a suitable solvent, like ammonia, soda ash, or caustic soda, the gel is dissolved and forms a turbid suspension which will not settle clear. This is called the 'sol' form. In most cases it is the soluble sodium, ammonium, or potassium compound of a very large and complex polymerised molecule, in which the sodium or other part is an exceedingly small part of the whole. This is illustrated by cases 1a, 1b, 2a, and 2b of Fig. 4 (from Mr. Nichols' paper). If to such a sol, sulphuric acid be added, the acid takes away from it the base that made it soluble, and the gel form of the colloid is precipitated and will settle. If barium chloride or lime is added the reaction is something as follows:



The insoluble barium or calcium compound is formed, and will settle readily. If salt (NaCl) is added, it merely 'salts out' the sol; that is, when salt comes into the solution the sol has to go out of the solution until a solubility-equilibrium is reached. In Mr. Nichols' experiments, as he has indicated, nearly equal results were obtained by the following:



Mg per 200 c.c. pulp containing 2.5% solids ... 16.68 83.32 55.54 166.90
Chemically equivalent proportions used ... 0.34 0.77 1.98 2.89

On figuring to a basis of chemically equivalent proportions, it is seen that salt is much less effective than the other chemicals as a cause of coagulation. The lime does not show up very well on the equivalent basis, for a reason to be explained further on.

The fourth condition of matter is that of true solution, into which form the various soluble salts must be brought in order to act as precipitants. Now matter in all four of these conditions is acted upon both by gravity and by electrostatic repulsion. The repulsion is between particles having charges of the same sign. In the case of true solutions, the size of the dissolved molecules is so small that the electrostatic repulsion far outweighs the effect of gravitation, and the particles are uniformly diffused throughout the liquid. The sol is the first step in aggregation; the particles are large enough to reflect light, and while electrostatic repulsion still predominates, the effect of gravity is to make the upper layers of the liquid somewhat less concentrated than the lower. The gel and the crystal are the irregular (or amorphous) and the regular forms, respectively, of the solid state. The opposing effects of gravity and viscosity largely predominate, but electrostatic repulsion is partly responsible for the slow settlement of the finest grains.

It was shown above that electrolytes may promote sedimentation by forming insoluble compounds. A second way in which they may accomplish the same object is by decreasing the amount of OH ions present in the liquids. Both OH ions and all forms of clay particles in water are negatively charged. The OH ions appear, with most clays, to have a marked effect in causing the clay colloid to assume the sol form; while a depression of the number of OH ions tends to coagulate the sol and form the gel. The reason for the comparatively weaker action of lime (CaO) as a

settling agent is that, while it forms an insoluble compound, it also increases the OH concentration of

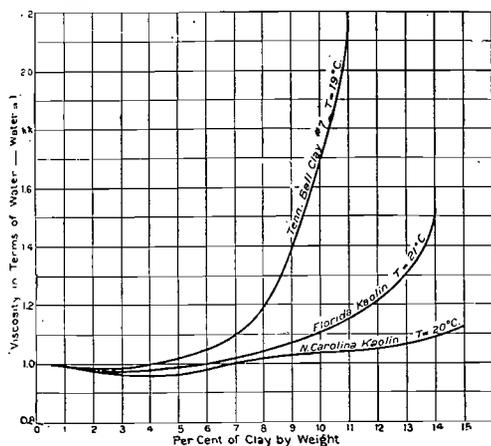


Fig. B.

the liquid, two opposite influences on sedimentation. Sulphuric acid, on the other hand, not only destroys the soluble compound, but it also decreases the OH concentration. With 4% H₂SO₄ the viscosity is about 6% greater than that of water, but these other effects predominate. Sulphuric acid is therefore a most effective coagulating agent.

Mr. Nichols' fourth deduction, that close packing of the clay particles will render the electrolyte inefficient and finally of no avail, is not exactly in accord with clay-working experience. Carbonate of soda (Na₂CO₃) is commonly added to clay slips, with the result that they require about half as much water to be capable of pouring as would be the case without the addition. Addition of acid sufficient to neutralise the soda stiffens such a slip again. This example is, however, with more dense slips than in the cases considered by Mr. Nichols.—H. E. ASHLEY.—*Mining and Scientific Press*, June 12, 1909, p. 831. (W. A. C.)

THE CHILE MILL.—“The comparison between the two styles of mill, Chile v. stamps and tube mills, is shown in the following table.

Comparison of Chile Mills and Stamps and Tube Mills.

	5 Chile Mills.	25 Stamps, 2 Tube Mills.
Capacity to 150 mesh ...	100 tons	100 tons
Cost of mills and power plant ...	\$24,000	\$17,000
Weight of mills and power plant ...	195 tons	127 tons
Power required ...	46 h.p.	95 h.p.
Wearing parts renewed ...	every 2 years	every 3 mons

The figures given in the table are based on cost of ironwork without framing, as the latter would be about the same in either case. The tubes figured were 3.5 × 14 ft., and the stamps 1,050 lb. Among the points not covered in the table is head room, the loss of which is much less with the Chile mill, thus giving greater latitude in choice of sites. Less actual ground space is also required. Supporting framework, building and erection costs, including a

simple hand crane for the Chile mills are about the same in either case, while depreciation, attendance (skilled as well as unskilled), accessory equipment, breakages and probable delays are all decidedly in favour of the Chile mill. The best quality of the stamp mill, its ability to withstand rough handling, is possessed by the Chile mill. Neither over-feeding nor under-feeding has any effect on its continuous operations, these conditions merely reducing capacity temporarily."—M. R. LAMB,—*Engineering and Mining Journal*, June 12, 1909, p. 1182. (A. R.)

TUBE-MILL LINERS.—"The following points have to be considered in connection with liners for tube mills:

- (1). They should be comparatively thin, so as not to take up much room in the mill.
- (2). They should be so arranged that there is no slip of the pebbles on their surfaces.
- (3). They must be easily put into the mill, require no attention, and be readily renewed.
- (4). They must show economy as regards wear.

The 'ribbed liner' here described has all these features.

This method of lining tube mills has been well tested at the Komata Reefs and Waihi Grand Junction batteries, and has given great satisfaction.

The quartz at the Komata mine is very hard, and very coarse screens are used on the battery (wove-wire screens, with three holes per linear inch), so the tube mills are running under very severe conditions. At present two mills, each 16 ft. × 4 ft., lined with 'ribbed liners,' are handling 100 tons per day of the 3-mesh product, and grinding it down to a very fine condition. The grades before and after grinding were respectively:—

On	Passed							
6-mesh.	10	20	40	60	90	120	200	200
%	%	%	%	%	%	%	%	%
6	14.5	18	13	10.5	13	4.5	4.5	16
—	—	—	—	1.5	16	15	8.5	59

This high efficiency of the mills is chiefly due to the system of liners, as the ribs are so arranged that there is no slip of the pebbles on the liners, and the cascade of pebbles, due to the revolving of the mill, is well developed. This is of the utmost importance in running tube mills. The mills at Komata are run with a 'returning box' in connection with them. This box is of the 'spitzkasten' type, and automatically returns the unground particles to the mills, the furnished product overflowing and going to the treatment plant.

The wear of pebbles and liners in the two Komata mills is 3 cwt. to 4 cwt. per week for each 16 ft. × 4 ft. mill, and totals 1.56d. per ton of sand.

A comparison between 'ribbed liners' and silix liners of the most improved type was made at the Waihi Grand Junction mine. The ore at this mine is not nearly as hard as at Komata. During fifteen weeks with the silix liners, 8,430 tons of sand were ground in the mill. The cost for liners was 0.66d., and for flints (13 tons 17 cwts., valued at £38 1s. 9d.), 1.08d. = 3.6lb., making a total of 1.74d. for flints and liners per ton of sand ground.

During the fifteen weeks with the 'ribbed liners,' 8,680 tons of sand were ground. Very careful measurements showed that the life of a set of liners, allowing one set of angle shoes for renewals, would be eighty weeks. The liners are of hard cast iron, and the estimated cost per ton of sand works out at 0.42d. The flints used during the fifteen weeks came to 5 tons 12 cwts., which worked out at 0.42d. = 1.44 lb. per ton of sand, making a total of 0.84d. for flints and liners per ton of sand.

The above tests show about the same tonnage in each case, as this was the total ore being crushed by the stampers during the test, but during the run with the 'ribbed liners' the grinding was very much finer than with the silix liners, the result being a reduction of 1s. 6d. per ton in the value of the sand tailings.

Owing to the 'ribbed liners' being comparatively thin, the average working-area of a 19 ft. × 4 ft. 10 in. mill is about 20 per cent. larger than with silix liners, and this, combined with the lifting action of the ribs, increases the grinding efficiency of a mill to a remarkable degree.

The arrangement suitable for a mill about 4 ft. diameter is eight ribs. It is of great importance that the ribs be correctly spaced and of correct size; roughly, they should be about 18 ins. apart, and a 4 ft. mill requires eight ribs, and a 5 ft. mill ten ribs. The ribs for a 4 ft. mill are 3 ins. × 2½ ins., and for a 5 ft. mill 3½ ins. × 3 ins. Another very important feature with these liners is the speed of the mill, as this should be slower than with the thick, smooth-surface liners, such as silix or concrete.—F. C. BROWN, *New Zealand Mines Record, London Mining Journal*, June 26, 1909, p. 795. (A. R.)

SCALING OF BATTERY PLATES.—"The high extraction recorded by Mr. Goddard was possibly due to the fact that the plates had been previously silver-plated, which prevents, to a certain extent, the absorption of the gold by the copper; and the amalgam was simply a cake on the surface and might have been cleaned-up almost as completely by the use of steel scrapers, without hammering the plates. On the other hand, had the plates been originally plain annealed copper, there would probably have been, at the time of scaling them, a much larger quantity of gold absorbed deeply into the copper, and a very small percentage would have been recovered by the method described. Consequently, if it is desired to recover the gold from annealed plates and avoid the expense of shipping them to a smelter, it becomes necessary to treat them by a more effective process. Below I give details of one by means of which practically all the gold is extracted from the plates without materially affecting them; in fact, in comparison with new ones the plates are perhaps improved, as they become more porous and therefore more absorbent to mercury, than before treatment. I give the following minute details of procedure because I have never seen the method referred to in any book:—

No. 1. After cleaning up in the ordinary way by means of a squeegee and scrapers, etc., the plate is taken up and well washed, after which it is heated all over evenly at a temperature just sufficiently high to eliminate all the mercury.

No. 2. While still warm, coat the upper surface with a mixture of finely powdered sal-ammoniac and hydrochloric acid (the mixture should be almost of the consistency of a paste and may be applied conveniently with a 3 in. paint brush).

No. 3. Then expose the plate to a moderately high temperature until evenly red hot all over.

No. 4. While red hot plunge suddenly into a tank of cold water, when nearly all the gold comes away in the form of scales, ranging from small particles up to pieces of 2 in. diameter. About 10% adheres to the surface of the plate, but is easily detached by chipping with a suitable edged tool.

The process is very simple, and, with convenient means of heating the plates and a suitable tank, I

believe a plate could be taken up, scaled and laid for re-amalgamation in about an hour.

To avoid the necessity of bending the plate the tank should be of a greater area than the plate and about 1 ft. in depth, and should have a smooth bottom to facilitate the collecting of the scales.

The scales are black in colour and contain possibly about 25 % copper, which is removed by boiling in dilute nitric acid in an enamelled cast-iron mercury bucket. The liquid is decanted, and the operation is repeated three or four times until the acid shows only faint signs of copper in solution, after which the scales are washed, dried and smelted.

Some years ago I superintended the scaling of eight large battery plates from a 20-stamp mill in the vicinity of Boksburg, Transvaal. The operation was conducted under considerable difficulties on a wood fire in the open, and with a tank which required the bending of the plates to receive them. In spite of these inconveniences the work was accomplished within 24 hours, and most of the time was taken up between straightening the plates out again and annealing them.

I do not happen to have kept notes recording the exact extraction of the gold corresponding with those given by Mr. Goddard in his paper, but the result obtained was very satisfactory, and the fineness of the gold was found to be about 5 % higher than the average of that which had been yielded from the regular clean-ups extending over a few years, and I believe this is invariably the case when the scaling is done by this method.—J. E. BREAKELL, *Institution of Mining and Metallurgy, Bulletin No. 57*, June, 1909, p. 39. (H. A. W.)

MINING.

ACETYLENE LAMPS UNDERGROUND.—“Acetylene lamps, according to W. F. Slaughter (*Proc., Lake Superior Mining Institute*, vol. xiii., p. 206), consume about one-fifth as much oxygen as candles do when giving the same amount of light. On that account, when it is necessary to work in badly ventilated stopes or headings, acetylene lamps should be used, not only because they consume less oxygen, but also because they give a bright light in air in which it is difficult to make a candle burn. Acetylene lamps are now made small enough to be worn on a miner's hat; they are used at the Saginaw mine on the Menominee range, Michigan. While the lamps require some attention in order to keep the water feeding properly to the carbide, the miners soon learn how to care for them. The lamps then give little trouble. It is said that it costs about two-thirds as much for light when acetylene lamps are used as against candles. At the Michigan mine the cost of the carbide required to last a 10-hour shift is about 2 cents.”—*Engineering and Mining Journal*, Jan. 16, 1909, p. 177. (C. B. S.)

LOADING BLAST HOLES AND DRIVING SMALL DRIFTS.—“Contributors to recent issues of the *Journal* appear to hold widely divergent views as to the proper method of loading up the drill holes. As to the proper place for the detonating cap, I am certain that Mr. Adkinson is correct in stating that a hole shows more shattering in the vicinity of the detonating cap: this is no reason, however, for placing the cap in the bottom of the hole.

As everyone knows, dynamite is detonated from a solid into a gas by pressure. When the detonating cap lets go it compresses about 2 cub. in. of the surrounding mass of dynamite sufficiently to cause dissociation; the transition from solid to gas must

occupy a definite interval of time, about one-twenty thousandth of a second. The initial explosion of 1 or 2 cub. in. of dynamite in the immediate vicinity of the cap would generate sufficient force to compress the entire charge to the dissociating pressure.

Detonation of a mass of black powder is effected by a uniformly expanding zone of flame starting from a spark. With dynamite detonation is accomplished with a succession of jolts separated by imperceptible but definite intervals of time. This explains why a strong cap and a well packed charge give definitely better results. The strong cap detonates enough of the surrounding charge to make a strong primary explosion. When the charge is well tamped the primary explosion would compress 2 ft. or 3 ft. of powder to the pressure of dissociation. With a weak cap and poorly compacted dynamite, it is conceivable that it might take three or four jolts to explode the whole charge.

The reason the ground around the primer is badly shattered is that the primary explosion of an ounce or so of dynamite has not power enough to spring the rock burdened on the hole, and as the tamping is tightly up against the explosive, the pressure crushes and shatters. In the interval that precedes the detonation of the main portion of the charge the gas from the primary explosion blows the tamping back an inch or so and cushions the main explosion to a certain extent. For these reasons I believe a cap should be in the top of the charge, because it is evident that the main or secondary explosion tears the rock out. With a cap on top a small part of the upper end of the charge is rendered useless for rock breaking purposes; with the cap in the bottom a small part of the bottom of the charge is consumed in shattering the bottom of the hole. With a lifter or back hole it would not make a particle of difference, as far as breaking ground went, where the cap was; with a cut hole, where the power is needed on the toe of the hole, a cap on top of the charge will tend to put the locus of the principal concussion a fraction of an inch nearer the bottom of the hole.

Tamping.—In breaking rocks with mud-caps, I find it makes a very material difference whether or not the detonating cap is placed on top of the dynamite. The most effective way to break a rock is to pick out a little depression and form a wall around it with plastic clay; fill this crater with dynamite, lay the cap on the centre and cover the whole with soft clay. If the cap is shoved down into the dynamite the breaking power of the mud cap is lessened. The only possible explanation of this phenomenon is that when the detonator is near the bottom of the dynamite, the detonation lifts a portion of the explosive bodily from the rock before it bursts into gas. High explosives must have a good contact with rock which it is desired to break. When a mud cap is placed on top of a stone, the explosive is usually plastered down tight against it, and the stone is broken into several fragments; if the same quantity of explosive were poked under the stone it would have poor contact and do very little execution. This is the reason why dynamite is supposed to strike down.

In regard to tamping a hole, I think that ordinarily 8 in. or 10 in. of clay tamping is enough. Strong cut holes require more tamping to get the full power of the explosive. Take the case of a 6 ft. inch hole in the face of a drift, the hole being 1½ in. in diameter and loaded with 32 in. of 60% explosive. The centre of the charge would be burdened 4 ft. and in hard granite or trap it would take one two-thousandth of a second for the concussion to travel from the explosion to the free face. This means that for a

period of one two-thousandth of a second the gases of the explosion must be confined. At the end of the period, the momentum imparted to the rock tears it away from the solid. During this period the tamping will be subjected to a pressure of 450,000 lb., sufficient to force 18 in. of clay or 3 ft. of water tamping 6 in. This shows that 18 in. of clay tamping is none too much. A 6 ft. hole burdened 30 in. would kick 8 in. of clay tamping back 6 in. before the rock cleaved from the solid.

A quick way to tamp up a round of holes is to have one of the muckers make a candle box full of clay slugs $\frac{1}{2}$ in. in diameter and 4 in. or 5 in. long. In loading twist a piece of paper around one end of the slug and they are easily shoved in and rammed home with the loading stick. I think the best results are obtained by putting in 18 in. of tamping for the incut holes and 14 in. for the other holes. Using a 2 in. time interval on the fuses, the fire in the fuse will be several inches in the tamping before an adjacent hole goes off. It is no uncommon thing for a hole to kick off 2 ft. or 3 ft. of another hole; as long as the fuse was burned down into the tamping and firmly bedded in a foot or so of clay, the spent fuse would break in case a collar blew off; otherwise the fuse might be pulled out of the hole. I noted a case in which one hole cut another hole off in the middle of the powder charge; the hole had pulled back to a slip and the dynamite was cut so smoothly that there was no sign of a hole on the slip until one of the men stuck his pick into it.

This cutting off of holes is a prolific source of accident. If the holes are overloaded as they often are, powder is thrown in the muck. Of course, there is no method of blasting that will eliminate accidents altogether, because things are liable to happen in blasting a round of holes that no man can foresee.

Drifting and Crosscutting.—In driving drifts and crosscuts where there are no good slips or floors to break to, the 'V' or pyramid incut will be found to be the most satisfactory. In narrow drifts four holes will be enough and they should be drilled so that the points nearly or quite intersect. In drifting, I think the double-screw column is much handier than a cross-bar, although it takes longer to rig up on a column than it does on a bar. In uneven ground, the machine often runs up or down on a hole; with a column and arm it is possible quickly to adjust the machine for alignment and often prevent the steel from fitchering and running tight. In very tight, hard ground, it pays to set the column as close to the face as the machine will work; drill the holes about $5\frac{1}{2}$ ft., and give them as much lift as possible. Of course, when the cut holes are drilled with the machine close in, the column has to be rigged back about 18 in. to drill the back holes and lifters.

If the ground is only moderately hard it pays to drill the cut holes so as to bottom $1\frac{3}{4}$ in.; then 4 lb. of 60% or 75% dynamite can be put in the bottom 30 in. of each hole. If the ground is extremely hard, time can be made by drilling the holes smaller, bottoming them $1\frac{1}{4}$ or $1\frac{3}{8}$ in. Not only do small steels cut faster and hold their gauge better, but it is also far easier to collar a hole on a slanting face with a small starter.

When small holes are used it will be necessary to spring the four cut holes with about four $1\frac{1}{2}$ in. sticks of 60%, then blow them out well with a blowpipe; this will chamber the holes so that they will take enough powder to throw out the incut. It takes about 40 minutes to spring four holes, (15 minutes to tamp up, 15 minutes to get the smoke out of the

drift, and 10 minutes with the blowpipe to clean and cool the holes.)

Order of Fire and Misfires.—In most cases it is a decided advantage to blast the incut with a battery, although it takes more time than if the holes are all loaded and fired in one round. When fired with a battery the cut generally kicks the collars of a few of the surrounding holes, so that after the incut round there is not the same danger of holes cutting one another off and causing mishaps and scattering powder in the muck. Again if the cut does not come out well the bottoms can be reloaded and split with the squaring holes. When the incut is blasted separately, there is not much danger of the machine-men lighting holes out of their proper order and perhaps spoiling the whole round.

If a hole that is solidly tamped with about a foot or so of clay tamping misses fire, it is certainly no joke to dig the tamping out. If the caps are always soaped, and waterproof fuse is used, such mishaps are bound to be extremely rare. One method of drawing tamping that is a little safer than picking it out with a scraper, is to burn it out by repeatedly loading up whatever collar there is to the hole; after two or three shots the concussion will jar through the tamping and set off the main charge. Another method, which I believe is safer, is to make a little tank by capping both ends of a 6 in. or 8 in. pipe about 3 ft. long, taping in a couple of inch nipples on opposite sides of the pipe. This tank would hold two or three pails of water. To blow tamping use a small tip on the blowpipe, wedge a plank or board across the drift, and after inserting the blowpipe in the hole, lash it fast to the board so that it cannot kick back. Now run the blowpipe hose to the water tank and run the machine hose from the air pipe to the upper nipple of the tank. Use a valve 200 ft. or 300 ft. back from the face for turning on the air pressure. This method works well in an upper or flat hole; it will often wash out a foot of tamping without shifting the blowpipe."—G. C. McFARLANE.—*The Engineering and Mining Journal*, Jan. 23, 1909, pp. 225-6, (C. B. S.)

A SELECTIVE ELECTRIC FUSE SPITTING DEVICE.

—"Spitting holes in shafts, especially in wet ground, is hazardous work, and the danger has been greatly increased by the introduction of electric hoisting. With electric-power systems the loss of power is sudden and without warning, while in the case of air or steam there is a gradual dying away of the power. Hence, there is danger of the failure of the electric power between the blasting signal warning the engineer to be ready to hoist and the final signal for hoisting the miners away from the blast.

Since even the best of electric-power systems is liable to fail, especially when the transmission line is long, the solving of the problem of blasting without the attendant danger of being caught when the holes are fired, has been given much study. Of course, simultaneous blasting by electric current is possible, but that is not desirable in shaft work where timbers generally have to be kept close to the bottom and where the cut holes must be fired first to get the best results.

The device described in the following paragraphs was perfected at the Hecla mine, in the Comr d'Alene district, Idaho, for selective firing of holes from a distance by means of electric current. In sinking a big three-compartment shaft at that mine, the air plant being taxed to its full capacity, the hoisting had to be done from the main plant, which was operated by electric power generated at Spokane

Falls, Washington, over 100 miles distant. The frequent temporary interruptions in the transmission of the power got on the nerves of the shaftmen, for they feared that the power might fail when they were spitting heavy blasts in the shaft. The result was that the company electrician devised a plan of spitting the fuse at a safe distance so as to eliminate this danger.

One Miss in a Thousand Shots.—The first device used was not satisfactory, but by rebuilding it and using a higher voltage, P. C. Schools, electrician at the mine, has succeeded in bringing the machine to such a state of perfection that the misses amount to only 1 in 1,000.

The perfected system consists of a firing board, where the operator tests his circuits and 'spits' his holes in the order desired; a reel, on which is wound the cable carrying the wire used in spitting; and firing blocks attached to the end of the cable. The holes are charged and primed in the usual way, and the spitting wire is inserted in a slit cut in the fuse near its end. The fuse is wrapped tightly with electrician's tape, and thoroughly coated with axle grease, so that the juncture is practically waterproof, and the spitting can be done successfully under water.

The spitting ends are all prepared before going into the shaft, and the fuses are all cut the same length, as the operator gives the time interval between holes when he inserts the plug at the firing board. Each fuse has two leads of a spitting wire projecting from its end. The cable containing the wires with the attached firing blocks, which is kept on a reel in the station on the next level above the shaft bottom, is now lowered to the bottom, and the two No. 16 annunciator wires projecting from the fuse of the first hole to be fired are securely wrapped around the two heavy copper leads of block No. 1. This gives hole No. 1 direct connection with the firing board on the level above. Holes Nos. 2, 3, 4, etc., are then attached to the numbered blocks in the order in which they are desired to explode.

The system is designed for a 24-hole round, which, of course, can be used for fewer holes if desired; the number of holes can easily be increased, but that rarely would be necessary.

Tests Insure Ignition of the Fuse.—When all the holes are ready to be fired, the men are hoisted to the firing station, and the circuits are tested out. To test the circuits, the main line switch is closed and care taken that the single-pole firing switch is open, for it is impossible to spit a fuse unless the firing switch is closed. This firing switch is kept in a box under lock and key, and only one man on each shift has a key to open it. The flexible cable and plug is then inserted into each of the holes in the firing board numbered to correspond to the holes below to be fired. If the circuits are closed, and ready to be fired, the lamps at the top of the board will light. If the lamps should not light, then there is something the matter with the circuit that must be remedied. If all the circuits tests closed, then the shots are ready to be fired.

To fire the shots, the main line switch is closed, the firing-switch box is unlocked, and plug inserted into No. 1, the lamp lights, the firing-switch is closed. This short circuits the lighted lamps causing them to go out, and at the same time applies 440 volts directly across the No. 26 tinned iron wire in the fuse at the bottom of the shaft. This wire melts with a blinding flash, spits the fuse, and burns itself free. The firing switch is then opened immediately, so that if an arc is maintained at the fuse it will be

smothered by the cutting in of the lamp resistance. With the plug still in No. 1, and the firing switch open, failure of the lamps to light indicates that the spitting wire at the bottom did its work, and the fuse is now burned; but if the lamps again light up brightly, it indicates that the fuse did not spit, and that the firing switch must again be closed. It is seldom, if ever, that the firing switch has to be reclosed.

Little Additional Time Required.—The operator then allows his time interval—a few seconds between the holes—which in most cases is simply time enough to change his plug to the next hole. He then proceeds with second hole as above described, retesting the circuit-firing resistance to see if the operation was successful, and continuing until all the holes are spit. The melting of the fuse wire leading to each hole disconnects the firing blocks so that the lower end of the cable is free, the upper end is then detached from the firing board, the cable wound on to the reel, and set aside until the next round. The fuses are all ignited and the shots go in the order desired without any attendant danger.

While seemingly complicated in description, this device can be cheaply installed where the current is available. In making this device nothing is required besides the ordinary material and apparatus kept at a mine where electric current is used for power. The original device was patented by one of the miners who was working in the shaft when it was used, but as neither of the inventors have taken out a patent, the perfected form is not covered by the stolen patent.

This device is as simple to operate as a telephone switch board, while the attaching of the firing blocks to the fuse takes little more time than would be required in spitting a fuse with a torch, and is quicker than spitting with a hot iron; but, of course, it is not speed that is important, but safety.—ROBT. N. BELL, *Engineering and Mining Journal*.—*Queensland Government Mining Journal*, March 15, 1909, p. 129. (C. B. S.)

MONAZITE.—"The extensive application of the oxide of thorium in the manufacture of mantles for incandescent gas lights creates a large demand for minerals containing thorium. Of the latter, monazite is found to be far the most abundant, and is therefore the principal source of thoria. Dr. Charles Baskerville mentions eighty minerals in which thorium has been found. Some of these would possess high commercial value as a source of thorium if found in sufficient quantity. Among these may be mentioned thorianite with 58 to 89% of thoria, auelrite with 70%, calciorthorite with 59 to 60%, thorite with 48 to 72%, uranorthorite with 48 to 52%, and thorogummite with 41 to 42%. Auelrite has been found in small quantities in the zircon mines of Henderson County, N.C. Frank L. Hess describes the occurrence of several thorium-bearing minerals along with the other rare earth minerals in the famous Baringer Hill locality, Texas. These are mackinoshite, thorogummite, nivenite, and yttrialite; they are of scientific interest rather than of commercial value as a source of thoria. Thorianite and thorite, especially the former, are found in quantities of commercial value in Ceylon.

Monazite is a phosphate of cerium, lanthanum, praseodidymium, and neodidymium with a variable percentage of silica and thoria. The proportion of thoria in monazite ranges from less than 1% to more than 20%. The average quantity in monazite obtained for commercial purposes varies between 3

and 9 per cent. The colour of the monazite ranges from grayish to yellow, to reddish, to brownish, and greenish. The lustre is resinous, and is especially brilliant on cleavage faces. Monazite is opaque to translucent, and subtransparent. The specific gravity ranges from 4.9 to 5.3, and is generally over 5, so that the mineral is readily concentrated by ordinary methods of washing. The mineral is brittle, with a hardness of 5 to 5.5. Monazite generally occurs in small crystals with brilliant faces in the original rock matrix. These crystals are rounded by attrition when set free from the rocks and deposited in gravel beds by streams.

The bulk of the world's supply of monazite for the preparation of thorium nitrate used in the manufacture of incandescent gas mantles comes from Brazil. The Brazilian monazite is exported to foreign countries, principally Germany and Great Britain, for manufacture. The plants manufacturing thorium nitrate from monazite in the United States are supplied entirely by the domestic production. Up to the present time all the monazite produced for commercial purposes in the United States has come from North Carolina and South Carolina. Monazite occurs in a number of other States, and apparently promising deposits have been tested in Idaho and Georgia. The monazite region of North Carolina and South Carolina is a north-east-south-west belt twenty to thirty miles wide and over 150 miles long. This belt lies wholly within the Piedmont Plateau, and borders closely on the Blue Ridge, to whose general course it is roughly parallel.

Monazite occurs in quantities of commercial value in the Carolinas in the form of sand in alluvial gravel deposits. These deposits are situated in the bottom lands along creeks and streams, in the stream beds, and in some cases in terrace formations. The gravels of these deposits vary in richness, and probably do not often run over 1% monazite. The gravel beds range from one foot to several feet in thickness, and they cover areas varying from a few square yards to many acres. Occasionally, the rock formations similar to those from which the monazite is derived carry an appreciable amount of monazite. In no cases, however, have these rocks been found rich enough for profitable working on a large scale.

The usual method of mining monazite is by concentration in sluice boxes. In some cases Wilfley or other concentrating tables, operated by gasoline motors, are used to separate the monazite from the gravel. The crude sand thus obtained ranges from 15% to 60 or 70% monazite. By treatment with electro-magnetic machinery, the grade of the sand is raised to about 90% monazite with an available content of 4½ to 5% of thorium. The monazite is shipped in this form to the manufacturers, and there turned into nitrate with a thorium content of 48 to 50%.—DOUGLAS B. STERRETT.—*Mineral Resources, U.S.A.*, 1908, *London Mining Journal*, June 5, 1909, p. 700. (A.R.)

COST OF UNDERGROUND WORK.—"The general manager of the Golden Horseshoe Mine has furnished the subjoined statement relating to the underground department, and containing some interesting particulars of the operations of the mine during the year ended 31st December, 1908.

(Tons wherever shown are of 2,240 lb.)

Work done by 3½ in. Rock Drill Machines.—The total average number of machines in constant use throughout the mine was 30.81. The following are particulars of work done in stopes and on development:—

Stoping.—

Machines in use per shift :	24.81
Number of holes bored	72.338
Average depth per hole bored	6.92 ft.
Average number of feet bored per drill per shift	22.53 ft.
Tonnage of ore broken per drill per shift	10.74 tons.

Explosives.—

Stoping: Explosives used per ton of ore broken	0.745 lb.
Drives: Explosives used per foot driven	14.09 lb.
Crosscuts: Explosives used per foot cross-cut	9.66 lb.
Passes, Rises and Winzes: Explosives used per ton 'risen' or sunk	9.38 lb.
Shaft Sinking: Explosives used per foot sunk	16.16 lb.

Ore Broken and Raised.—

Average tonnage per man per shift (based on total number of men engaged in the underground department, viz., 495)=1.6825 tons.

Average tonnage per man per shift (based on total number of men actually engaged in stoping and filling ore into shoots, viz., 275 men for 298 1.3 days, breaking 238,552 tons)=2.9077 tons."

Monthly Journal, Chamber of Mines, W.A.—Australian Mining and Engineering Review, June 5, 1909, p. 294. (A. R.)

MISCELLANEOUS.

AN IMPROVED TAPE REEL FOR ENGINEERS AND SURVEYORS.—"This new design is intended to overcome the many objections, difficulties and annoyances incident to the use of the present designs of reels—objections which, in many cases, even a novice in their use has easily realized, but which somehow have escaped the attention of the reel makers.

Among these defects are (1) slow winding; (2) difficulty in preventing the unwinding of the tape; (3) easy clogging along friction surface with dirt and grit. Smaller defects may be found in the large friction surface, short winding crank, etc. The makers of this new reel, which they call the "Ideal," claim to overcome all these and other defects. Thus, where it takes—with the present tape-reel—about 130 turns of the winding crank to wind up a 100-ft. tape, consuming from thirty-five seconds to a minute of time, by means of an arrangement of gearing, giving three revolutions of the tape drum to each revolution of the winding crank, only 40 revolutions are necessary to wind up the 100-ft. tape. This can be done with ease in ten seconds. This feature of quick winding is particularly desirable in working on city streets and other places where interference from traffic is constant, and many broken tapes would be avoided by such a rapid winding device. The second defect in the present reels mentioned above is obviated by the introduction of a 'tape arrester.' This is merely a rod working in a tube attached to the reel-plate. Normally, the inner end of this piston is free from the gear wheel, but on pressing it with the forefinger the inner end engages the teeth of the larger gear wheel and instantly arrests the unwinding of the tape. This feature, taken in combination with a leather handle attached to the reel, permits the operator to make accurate measurements without the necessity of laying the reel aside or otherwise disposing of it—it will save cut fingers, broken tapes, and inaccurate operations, to say nothing of the temper of the operator.

The third defect mentioned, that is, the clogging of the friction surface, as well as the other minor defects, are overcome by the form and method of

construction. All the gearing is well protected and the form of the plates has been made to fit the hand and thus prevent slipping during the operation of winding. The drum is light, strong and free from friction surface, revolving about an axle attached to the smaller gear; there is nothing to be clogged, and any grit that may collect about the axle can be easily removed. Altogether the advantages claimed for the new reel are so striking that every tape user should investigate its merits."—*The Engineering Digest*, Feb., 1909. (C. B. S.)

BONDING NEW TO OLD CONCRETE.—"For connections made after a lapse of 24 hours or more, break back the surface concrete to firm material, and clean the fresh surface with steam, air blast or forceful water streams so as to remove all fine loose material. Saturate well, but not so that water stands on the surface or oozes from the material. Paint completely with neat cement grout, mixed to the consistency of thin cream, just before new concrete is deposited, and see that the latter is of proper mixture, containing a proper proportion of mortar, which should be worked against the joint so as to be certain that no voids exist in its vicinity.

For connections made after long intervals, so that the old cement has set hard, and where the expense of rough-pointing the whole surface is greater than is required because of the nature of the desired bond, use commercial muriatic acid, diluted with clear water, 1 to 5, or the commercial bonding powders, dissolved in clear water at the rate of 5 lb. of powder to 10 gal. of water. First wet the old concrete surface with so much water that a fresh wetting is not immediately absorbed. Remove any excess of moisture, and, when the surface appears as if commencing to dry, paint on the old surface three successive coats of acid one after the other. Let this remain for about 30 min., after which carefully clean the surface of unspent acid, soluble salts, and fine material, with plenty of water, finally cleaning with a steam jet or air blast if obtainable. Just before the fresh concrete is to be deposited, and while the old material is still very damp, apply a thin coat of neat cement grout mixed to the consistency of thin cream just before the new concrete is deposited, and see that the latter is of proper mixture containing a proper proportion of mortar, which should be worked against the joint so as to be certain no voids exist in its vicinity.

Generally, if proper bonding methods are adopted, with acid and grout, work may be stopped along 45° planes at any point in a beam, at pleasure, between the ends and the third points; such planes would better be diagonal compression planes. The writer prefers such joints to vertical ones at the centers of beams, where direct tensile and compressive stresses are larger. Further, it is to be remarked that nothing but ample reinforcement, consisting of stirrups or numerous diagonal rods, can compensate for the detrimental effect of a horizontal joint left along or near the central axis of a beam. The writer prefers numerous stirrups, well connected to the tension rods, and amply anchored in the top concrete."—From a paper read by Mr. E. P. Goodrich before the American Society of Civil Engineers, March 3, 1909. —*The Engineering Digest*. (C. B. S.)

THE GAS ENGINE AND LOW-GRADE FUELS.—"The value of a combustible as a fuel for power generation depends not only on its thermal efficiency, but on local conditions affecting cost. While coal and petroleum are the most efficient and concentrated

fuels available for power purposes, they may not always be the most economical.

There is an increasing effort to make available the lower grade of fuels at the places where they are found and transportation costs do not have to be considered. Efforts in this direction have met with considerable success especially in the adaptation of the gas producer to the generation of gas for gas engines, from materials that have usually been considered as having little fuel value. The power being generated by exploding the gas with air, the attainment of high temperatures required in steam generation is not necessary, or desirable, sufficient heat only need be generated in the producer to distill the volatile matter from the fuel and to burn the solid carbon to the combustible carbon monoxide. The completion of the combustion takes place in the cylinder of the engine when the mixture of gas and air is exploded.

Producer gas through its use in gas engines is becoming an important factor in solving the power problem in those regions where water is too scarce or contains too many impurities for satisfactory use in steam boilers."—*The Mining World*, June 26, 1909. (C. B. S.)

MEANING OF THE KILOWATT HOUR.—"Many are puzzled to understand the peculiar significance of some of the terms used in denoting the measurement of electric current. The words 'amperes,' 'volts' and 'watts' slip easily off the tongue of those who are familiar with the language of electrical science, but it is not very well appreciated how many are unable to comprehend their precise meaning. The 'kilowatt hour,' however, gives more trouble than any of the other terms.

To explain this term needs first a clear definition, and then a comparison. Everyone will understand that a certain amount of force must be used to drive electric currents through a circuit. This force is measured by volts; thus we have 110-volt currents and 220-volt currents, the one expressing just twice the force of the other. But the quantity of current passing through a circuit depends upon the force and the resistance, and so the quantity is expressed by a different term, viz., 'amperes.' Now the efficiency of the current depends upon both force and quantity, and to express this efficiency or united action we multiply the force by the quantity—that is, the volts by the amperes—and express the result in watts. Thus 110 volts multiplied by five amperes is 550 watts, or 220 volts multiplied by five amperes is 1,100 watts.

A kilowatt is, of course, 1,000 watts, which is equivalent to about $1\frac{1}{3}$ h.-p. In charging a battery the lighting company's bill for the use of so many watts for so many hours—thus 1,000 watts for 10 hours—would be charged as 10 kilowatt-hours, which at 5 cts. a kilowatt-hour would be 50 cts., a charge that seems little enough for 10 hours' use of $1\frac{1}{3}$ h.-p."—*Mining World*, June 26, 1909, p. 1228. (C. B. S.)

INCREASE IN WORLD'S GOLD PRODUCTION.—The London *Statist* gives the world's production of gold in 1908 as the largest in history, or \$409,000,000, against \$400,000,000 in 1907 and \$390,000,000 in 1906. The outlook is for a further expansion of gold production, because of its stable value, more economical mining with modern machinery, and the thorough extraction, compared to the loss of one-third of the gold in milling a generation ago.—*United States Consular Report*.—*Mines and Minerals* August, 1909, p. 44. (A. R.)

TEST OF LARGE REINFORCED CONCRETE BEAMS.—“A pamphlet issued by the University of Illinois gives particulars concerning an interesting series of tests conducted by Professor A. N. Talbot with the object of determining the actual properties of beams made under practical conditions of construction: The members were taken from a number of beams prepared for use on the Illinois Central Railroad, and were in the form of slabs 25 ft. long by 6 ft. 3 in. wide by 2 ft. 10 in. deep at the middle, the average weight being 33 tons each, the weight being probably expressed in American tons of 2,000 lb. It is satisfactory to observe that the behaviour of the slabs was similar to that of beams made and tested in laboratories, a result which should inspire confidence in the quality and soundness of reinforced concrete work of judicious design and constructed with care. The tests show very clearly the efficiency of vertical U-shaped stirrups and bars bent up towards the ends in withstanding diagonal tension. Ample precaution against incipient failure by diagonal tension is specially important because of the risk of sudden failure and the impossibility of detecting signs of weakness in places where the sides of beams are not open to inspection.”—*Mining, Building and Engineering Journal*, July 10, 1909. (C. B. S.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE COPPER HANDBOOK FOR 1908. Compiled and Published by H. J. Stevens, Houghton, Mich., U.S.A. \$5.

“The present is the eighth annual edition of this work which has grown since 1901 from 328 pages to 1,500 pages of text. No less than 6,767 copper mines and companies are listed and described, the descriptions varying from a few lines to several pages according to importance. There are also 23 chapters dealing with the history, geology, geography, mineralogy, chemistry, metallurgy and statistics of copper. To those persons who are interested in copper mines the work is invaluable.”—*Mining Engineering*, July, 1909. (A. R.)

HANDBOOK FOR FIELD GEOLOGISTS. By C. W. HAYES, Ph.D., Chief Geologist, U.S. Geological Survey. viii. +159, (New York: John Wiley & Sons. London: Chapman & Hall, Ltd.) 6s. 6d.

“One of the most useful geological handbooks that we have seen for a long time is a handbook for Field Geologists which has recently been received for review. In order to make the handbook as complete as possible within reasonable limits some 84 pages of general instructions are given with regard to preparation for field work, outfit, field observations and the method of making various horizontal and vertical measurements, the use and manner of recording notes, and the collecting of specimens, together with a deal of information with regard to the various details of field operations. Excellent and concise as these instructions are, the geologist who has occasion to do much work in the open will doubtless find the instructions for special investigations which are given in Part II. a most useful and thoroughly well arranged series. A number of schedules have been prepared with a view to securing system and com-

pleteness in making and recording observations. Most of them are not intended to be exhaustive or to cover all possible points on which the specialist will make observations. They are rather for the guidance and help of the specialist when he is working outside his speciality. These objects have been clearly borne in mind by Dr. Hayes, who deals separately with the different classes of rocks, igneous and sedimentary, and points out the special matters which are to be investigated and considered in each. At the end of the chapters in which these things are discussed, a summary of the instructions is given, together with notes regarding the significance of the various characteristics which are necessary to be noted. Following these there are given a collection of schedules containing every important heading under which facts should be noted, regarding the geology and structure of rock masses generally, the geological details of glaciers and glacial deposits; metalliferous ores, placer deposits, stone, road materials, cement materials and lime, clay, sand, gravel, coal and gas, and the rest. In all there are fourteen separate schedules, and it would be difficult to think of any rock characteristic or point, however small, about which inquiry should be made, which is not in one way or another mentioned or suggested in the schedules. The handbook is of convenient size, well bound, and in every way a *vade mecum* which should be in the hands of every geologist who takes his work seriously.”—*S.A. Mining Journal*, Aug. 21, 1909, p. 702. (A. R.)

IGNEOUS ROCKS. Vol. I. By JOSEPH P. IDDINGS. (New York City: John Wiley & Sons. \$5.

“Vol. I. is devoted to composition, texture, and classification of igneous rocks. It is, naturally, highly specialised and interests chiefly the petrographers. The book is well illustrated and contains copious references to the general literature on the subject.”—*Mines and Minerals*, Aug., 1909. (A. R.)

HANDBOOK ON TACHEOMETRICAL SURVEYING. By C. XYDIS, C.E. 6s. (London: E. & F. N. Spon, Limited.)

“A useful book, which comes to fill a gap in our technical literature, as hitherto very little has appeared in print on the subject handled so thoroughly in this work. We remember only a paper by the late Mr. Brough and another by Mr. Niel Kennedy.

The pros and cons of the subject we need not discuss here, as they are well known to those of our readers who are interested in such work. Unlike the majority of its kind, this work is anything but voluminous, and on that ground, in this busy age, it appeals to us; moreover, its information is couched in plain and concise terms, with many practical suggestions for methods of working, serviceable alike to surveyors and their pupils.”—*London Mining Journal*, Aug. 21, 1909. (A. R.)

PREHISTORIC RHODESIA. An Examination of the Ethnological and Archaeological Evidences as to the Origin and Age of the Rock Mines and Stone Buildings, with a Gazetteer of Mediaeval South-East Africa. By R. N. HALL, author of ‘The Ancient Ruins of Rhodesia.’ With Illustrations, Maps and Plans. 12s. 6d. (London: T. Fisher Unwin.)

“This book describes the ruins and ancient mine workings of Rhodesia. It is published in an attractive form, while the descriptions are aided by a number of most excellent photographs.

Mr. Hall does not believe in Prof. MacIver's conclusions — viz., that the ruins are comparatively modern. He endeavours to prove their antiquity, and to upset the statement made—that Nankin china was found below the walls of the oldest buildings.

Investigations of this nature present great difficulties, for we note the ruins appear to be almost characterised by their irregular design, while the absence of squares and circles in their constructive details is a feature worth noting; the apparent absence of inscriptions increases the difficulties of research work. Mr. Hall is to be congratulated upon the way he has massed and pieced together the scattered records, involving, as it must have done, a vast amount of patient labour.

It seems doubtful if the true age of these Rhodesian remains will ever be known. The extensive ancient mine workings which abound all over that territory are evidence of large quantities of gold having been mined. We think that the estimates of tonnages and values of ground treated are too problematical to be considered.

The conclusions generally favour the antiquity of these remains, and there is also evidence that the mine workings have been undertaken from ancient even up to modern times.

It would appear that Rhodesia in the past has provided a large population with the means of subsistence, and that to-day the country promises to rival its ancient condition in value and prosperity."—*London Mining Journal*, Aug. 21, 1909. (A. R.)

INVESTIGATION OF THE PEAT BOGS AND PEAT INDUSTRY OF CANADA DURING THE SEASON 1908-9. By ERIK NYSTRÖM, M.E., and S. A. ANREP, M.E. (Ottawa, Canada: Department of Mines.)

"A year ago, the Mines Branch of the Department of Mines, Ottawa, issued a report on 'Peat and Lignite, their Manufacture and Uses in Europe,' with the object of giving to Canadians as complete a review as possible of this industry in those countries in which it has been most successfully carried on. This report is now followed by a bulletin, entitled 'The Investigation of the Peat Bogs and Peat Industry of Canada during the Season 1908-9,' by Erik Nyström, M.E., peat expert. This bulletin comprises 25 pages of text, and includes six large scale maps of peat bogs.

The bulletin contains a descriptive report of each bog, showing the location, area and structure, and giving an estimate of the available supply of peat fuel, with records and analyses, calorific values, etc., and should be of particular interest to those engaged in, or connected with, the development of Canadian peat resources.

A fuel-testing plant is now being erected at Ottawa, in which the value of peat for the production of power-gas will be demonstrated, and the Department proposes to carry on a very thorough investigation of this subject."—*London Mining Journal*, Aug. 21, 1909. (A. R.)

METALLURGY OF THE COMMON (NON-FERROUS) METALS. By A. HUMBOLT SEXTON, F.I.C., F.C.S., etc., and JOHN S. G. PRIMROSE, A.G.T.C., etc. (Manchester: The Scientific Publishing Company.)

"The exhaustive treatment which has been accorded to various common non-ferrous metals by different authors rather tends to dwarf the importance of a work which, in one volume of less than

500 pages, deals with the metallurgy of a number of these metals. But it must be explained that this book is only intended as an introduction. The requirements of those engaged in metallurgical work will be better met by the larger works, a list of which is given by the authors, for it is obviously impossible in the space occupied to deal fully with the metallurgy of each metal. For instance, antimony occupies 12 pages only, whereas another author finds sufficient material to fill a complete volume on the subject. Similarly other metals are only dealt with briefly, though copper comes in for much fuller treatment. This is only meet, in view of the extensive and varied uses to which that metal is put and the number of processes employed for extracting and refining it. In fact, about one-third of the book is devoted to the subject of copper, including the mineral, the principles of copper smelting, processes for treating the ore, and refining processes. The subject of tin is summarised in about 40 pages, and then follow zinc, lead, antimony, aluminium, and nickel. It may be suggested that in such a work it would have been well to include some information on other of the metals which, while scarcely classed as common, yet are in more or less extensive commercial use. That, however, is beside the point now, for the authors have set the limitation and have worked within it, producing a book which is certainly an excellent introduction to fuller works. Naturally, there has been made little attempt to offer original matter; rather has existing information been gathered together and condensed into one volume. This, we may say, is one of the leading characteristics of the work, and the most important features of the metallurgical processes dealt with are thus presented in a concise and systematic way."—*Iron and Coal Trades Review*, Sept. 5, 1909. (A. R.)

The following reports have been received from the Mines Departments mentioned, and have been transferred, for the convenience of members and for ready reference, to the Seymour Library, Transvaal University College.

A. From the Department of Mines, Canada.

1. Report on the Chrome Iron Deposits of the Eastern Townships, Province of Quebec. By Fritz Cirkel. pp. I.-IV. and 1-141.

This is the 29th number of the valuable series now being issued by the Canadian Department of Mines under the direction of Dr. Haanel. Its primary object is to meet the demand for information on certain aspects of the iron and steel industry, which in recent years has developed so rapidly in Canada. It deals very fully with the subject of its title, and in addition there are sections on the ores of other countries, including the Transvaal (c.f. Hall and Humphrey, *Trans. Geo. Soc. S. A.*, Vol. XI., May 18, 1908), and Rhodesia, as well as the composition, statistics and chronology of the chrome industry. A chapter on the determination of the value of chromite follows, and then one on its uses. It will be of interest to our members to quote the following from p. 102: "Chrome steel is extremely hard, tough and dense. It is possessed of great tensile strength, and is superior to any metal known for the wearing parts of mills used for crushing and pulverising gold, silver and other ores. This steel is especially adapted to severe service and resistance to abrasion, and for the wearing parts of stamp mills, such as battery shoes and dies, cams, tappets and bosses, chrome steel has proven, by reason of its durability, the most economical and satisfactory material to be obtained." A useful chapter on the technology of

chromium and its compounds follows, and two appendices (a) a reprint from the Mineral Industry on the Metallurgy of Chromium, (b) Experiments with Chromite at McGill University, Montreal. The report is well printed and illustrated with fifteen diagrams and eleven plates in half-tone, and will be valuable to all who are concerned with this industry. (J. A. W.)

2. Report on the Investigation of an Electric Shaft Furnace, Domnarfvet, Sweden, etc. By E. Haanel, Ph. D. pp. 1-38.

This report forms a sequence to the reports published in 1904, 1906 and 1907 by the department on the electric smelting of iron ores, and the investigation was undertaken by Dr. Haanel in December last on the invitation of the inventors. The results given represent the latest developments of the electric smelting of iron ores. The report is divided into four parts, the first of which deals very fully with the trial runs, and it is interesting to note that the experiments undertaken show that a saving could probably be effected under Swedish conditions at Domnarfvet of about \$1.55 per ton by the use of this furnace. In addition to the above there are two valuable reports constituting Parts III. and IV., the former dealing very fully with the general arrangements and full working details of the factories for the manufacture of carbon electrodes, and the latter with modern methods of charcoal making. Part II. describes a new electric furnace for the manufacture of steel. (J. A. W.)

B. From the Department of Mines, Western Australia.

1. Report on the Progress of Mining in the districts between Leonora and Wilma. By A. Montgomery, M.A., F.G.S., State Mining Engineer. pp. 1-85.

2. Report on the Waverley or Siberia District. By A. Montgomery, M.A., F.G.S., State Mining Engineer. pp. 1-26.

3. Report of the Department of Mines for the year 1908.

4. Bulletin No. 35: Geological Survey, Western Australia. Geological Report on the Gold and Copper Deposits of the Phillips River Goldfield. By H. P. Woodward, Assistant Government Geologist, with which is incorporated a description of the Crystalline Rocks of the District. By E. S. Sampson and L. Glanert (with two maps, eight plates and seven photographs). pp. 1-175.

C. From the Department of Mines, India.

1. Report of the Chief Inspector of Mines in India for the year ending 31st December, 1908. By J. R. R. Wilson, M.I.C.E., F.G.S. pp. 1-140.

D. From the Missouri Bureau of Geology and Mines.

1. Vol. VI., 2nd Series. The Lime and Cement resources of Missouri (U.S.A.) By H. A. Buehler, B.S., Assistant State Geologist. pp. I.-XVI. and 1-254.

2. Vol. V., 2nd Series. Public Roads. Their Improvement and Maintenance. By E. R. Buckley. (J.A.W.)

Adams, Prof. Henry. Cassell's Building Construction. Comprising Notes on Materials, Processes, Principles and Practice, including 2,284 engravings and 12 Plates. Part I. 4to., sd., pp. 24. Cassell. Net 3d.

British Standard Specification for Cast Iron Pipes for Hydraulic Power. (Engineering Standard Committee, No. 44. Folio, sd. Lockwood. Net 5s.

Butler, Edward. Carburettors, Vaporisers, and Distributing Valves Used in Internal Combustion Engines. 8vo., pp. 188. C. Griffin. Net 6s.

Cayen, R. M. Systematic Qualitative Analysis. For Students of Inorganic Chemistry. Cr. 8vo., pp. 240. Blackie. Net 3s. 6d.

Colly, Albert Ladd. Reinforced Concrete in Europe. Its Applications, Economic and Endurance. The Systems, the Forms of Bars, and the Metal used in England and on the Continent. Royal 8vo. Williams & Norgate. Net 14s. 6d.

Crane, W. R. Index of Mining Engineering Literature. 8vo. Chapman & Hall. Net 17.

Harrison, Newton. Wireless Telephone. Construction. Cr. 8vo., pp. 74. Spion. Limp, net 1s. 6d.

Hatch, F. H., and Corstorphine, G. S. The Geology of South Africa. 2nd Edition, 8vo., pp. xvi., 394. Macmillan. Net 21s.

Haughton, C. E. The Elements of Mechanics of Materials. A Text Book for Students in Engineering Courses. 8vo., pp. 194. Constable. Net 7s. 6d.

Holley, C. D. The Lead and Zinc Pigments. Cr. 8vo. Chapman & Hall. Net 12s. 6d.

Koester, Frank. Hydroelectric Developments and Engineering. Illustrated. Imp. 8vo., pp. 480. Constable. Net 21s.

McDonald, J. G. Hints to South African Farmers. Cr. 8vo., pp. 88. Argus Printing Co. Net 2s. 6d.

Moor, C. G. The Recognition of Minerals. Being a Collection of Notes and Simple Tests for the Use of Travellers and Prospectors. With Monographs on Geology. Ore Deposits, etc., By Donald A. MacAlister. Cr. 8vo., pp. vii., 231. Mining Journal. Net 7s. 6d.

Morris, W. L. Steam Power Plant Piping Systems. 8vo. Spion. Net 21.

Philip, James C. The Romance of Modern Chemistry. A Description in Non-Technical Language of the Diverse and Wonderful Ways in which Chemical Forces are at Work and of their Manifold Application in Modern Life. With 29 Illustrations and 15 Diagrams. Cr. 8vo., pp. 348. Seeley. 5s.

Ramsay, John. Engineering Units of Measurements, with Symbols and Abbreviations. British and Metric Equivalents. 8vo., sd. Simpkin. Net 1s.

Raynes, F. W. Domestic Sanitary and Engineering and Plumbing. 8vo., pp. 488. Longmans. Net 10s. 6d.

Report of the Senior Analyst of the Cape of Good Hope for the Year 1908. Folio, sd., pp. 25. W. Wesley. Net 2s.

Scales, F. Shillington. Practical Microscopy. An Introduction to the Microscopical Methods. 2nd Ed. Cr. 8vo., pp. 350. Bailliere. Net 5s.

Science Chemistry Papers. Being the Questions set at the Intermediate Science Examination of the University of London. From 1892 to 1909. (University Tutorial Series.) Cr. 8vo., pp. 56. Clive. 2s. 6d.

Thomas, H. H., and MacAlister, D. A. The Geology of Ore Deposits. Illustrated. Cr. 8vo., pp. 428. E. Arnold. Net 7s. 6d.

Thomas, H. H., and MacAlister, D. N. The Geology of Ore Deposits (Geological Series). 12mo., pp. 416. E. Arnold. Net 7s. 6d.

Tower, Walter Sheldon. The Story of Oil. Illustrated. Cr. 8vo., pp. 284. Appleton. Net 4s.

Trade, Board of. Electric Lighting Acts, 1882 and 1888. Regulations: (a) for securing the safety of the Public, and (b) for ensuring a proper and sufficient Supply of Electrical Energy.

Unwin, W. Cawthorne. The Elements of Machine Design. Part 1. New Edition Revised and Enlarged. 8vo., pp. 546. Longmans. Net 7s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M. Chart. Inst. P.A.
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