Preliminary study into a mechanized resue mining method for deep level mining

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Synopsis

Resue mining was practiced extensively in the East Rand in the first half of this century. The obvious benefits were that a higher grade of ore was trammed to the surface. The disadvantages were slow rates of face advance and an extremely labour-intensive method of mining. A collaborative effort by personnel from Avgold's Target Gold Mine and Sandvik Tamrock has resulted in the design of a method of resue mining which can be totally mechanized. The purpose of this paper is to describe the methodology used and the proposed new mechanized resue mining method.

Introduction

In May 1958 R.H. Bryson, Manager, Free State Geduld Mines, Ltd presented a paper to the Association of Mine Managers in which he described resue stoping at Loraine Gold Mines, Ltd. The principle of operation was mining of the Khaki Shale, which was hand packed behind the advancing face, and then the lifting of the Basal Reef. He quotes the following as the advantages and disadvantages of resue mining as applied to Loraine.

Advantages

- *Grade*. The cleaner the product sent to the mill the more economically attractive it must become.
- *Economy*. The cleaner the product the less utilization of scrapers, rolling stock, hoists, etc.
- *Maintenance*. With definite and well-defined cycles, stope services such as winches, air and water, etc., can be maintained and extended during the waste cycles.
- *Labour control*. All labour is concentrated on the working faces and is therefore easily supervised and controlled. There are no old areas in the stope where persons can hide, idle or get lost.
- *Fire*. The fire hazard is reduced, fire being confined by the waste fill to gullies, track cuttings and working faces.
- *Support*. Although the stope width in many cases exceeds 2,4 metres comparatively small packs may be used. The waste fill is of no use as an immediate support since the shale rapidly disintegrates and loses height on its own, but it does provide a steadying base for long and thin mat-pack support.
- *Supervision*. The stoping width enables supervisors to move about readily and in comfort: labour being confined to the face permits detailed supervision of actual jobs, and well-defined separate cycles require the minimum of labour organization once a routine has been established.

Disadvantages

- *Intermittent Face Advance*. Faces are stopped every 3 metres on strike for reef lifting, etc.
- *Hanging Control*. Permanent supports can be installed only after the reef lift is completed. During the waste overcut and reef lift period temporary supports only can be used.
- *Irregular Supply of Reef*. The amount of reef sent to the mill daily depends on the number of panels on reef lift and this fluctuates widely throughout the month.
- *Labour Variations.* The time taken to complete a cycle of operations is almost a direct function of the labour available to hand pack the waste overcut. Fluctuations in labour strength are almost immediately apparent in output.

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- *Excessive Waste.* When excessive waste is present it must be trammed as waste and in many cases this necessitates holding up work on other reef panels until all surplus waste has been trammed.
- *Gold Loss*. It being impossible to eliminate all sources of waste contamination, some must be accepted, and this in turn lends itself to abuse unless constantly watched.
- *General*. Strict and conscientious supervision is essential to avoid the following:
  - Not filling waste spaces properly and tramming overflow waste as reef.
  - Blasting waste before the stope inspector has passed the stope, thus causing loss of gold in sweepings and possible reef in the footwall.
  - Tramming waste as reef when it could be trammed as waste.
  - Gold loss in waste fill.

In comments to the paper R.C. Briggs refers to resue mining as applied at Sub Nigel in the late 1920s and a further trial carried out in 1955. He states that in his experience loss of gold in the waste fill is the most important disadvantage of the resue method and can be quite considerable under normal working conditions. Supervision is the only safeguard for minimizing this loss and is the most difficult part of the operation to maintain.

The average value of the 68 cm milling width of the stope was 8 gms/ton. The average of the waste fill was 1,1 gms/ton. If just sufficient waste was mined to fill the mined out area and it was packed at a density of 1,6, the gold loss in the waste fill can be calculated as being 14%.

Briggs argues that it was more cost effective to mine the ground conventionally and not to practice resue stoping.

In his 1963 paper, R.S. Pearson, Underground Manager, President Steyn Gold Mining Company, Ltd describes a mining method that separates the reef and waste handling and he says:

'In addition to minimizing the disadvantages of resue mining already mentioned the system also lends itself to reducing the possibility of reef dilution and/or reef losses in the waste fill. This being the great disadvantage of the resue system which has led to its being abandoned in favour of other systems wherever possible.'

As mining goes ever deeper and it becomes more difficult to attract rock drill operators it becomes essential that new methods of extracting the narrow reefs are developed. Ideally these new methods should result in safer more profitable mining methods. In other narrow seam mining operations, such as coal, the route followed has been mechanization. Currently available hard rock mining equipment is too large for operation in narrow reefs. It is suggested that the future lies in 'mining wide and tramming narrow' or in other words mechanized resue mining.

Assessment of the problem

The problem is how to mine a de-stressing cut before mining the massive orebodies of the Eldorado Fan at the Target Gold Mine. The various steps in the process and the actions taken are defined as follows. However, the first point is to define the geotechnical area where the proposed stoping system will operate.

Geotechnical setting

Depth of stoping is 2300 to 2500 metres below surface. The stoping will be done in a generally high siliceous rock environment. The rock mass rating of the area in general is in the order of 80+. The dip span of the stoping area is limited and less than the critical span, even for very low stoping widths. This also has the effect that the use of backfill or waste fill will also not cause the vertical stress to regenerate in the back areas of the stopes.

Define the untouchables

These are factors, which are inviolate and cannot be contravened. They are almost always safety related, but include all parameters which are deemed to be important. These factors are defined at the beginning of the design exercise and each suggestion or stoping system proposal is tested to ensure that it conforms to the untouchables.

- *Narrow stope width*. The behaviour of the rock in a stope face is dependent on the interaction of the stress level, the competency of the rock and the stope width. In a high stress environment the failure of the rock in the face of a narrow stope, say 80 centimetres, is confined to a smaller zone and failure will occur in a more controlled manner. Face bursting is less likely to occur. Experience from working in this type of ground has taught that it is hazardous to work in a moderately to highly stressed stope face more than 130 centimeters high.
- *No pillars to be left*. There are other mineable reefs below the stoping horizon where the de-stressing cut will take place. It is vital that no pillars are left in the stope to impose a concentration of high stress on the areas below.
- *Limited leads and lags*. Leads and lags between different stope faces not only cause stress concentrations but also result in cross fracturing. A hangingwall that is fractured in different directions and orientations is difficult to stabilize.
- *Cannot retreat*. A wave of high stress is 'pushed' ahead of a stope face and access development ahead of the stope face will be exposed to this high, and eventually intolerable stress condition. (The effect of the proximity of face blasting also plays a significant role in this.)
- *Maximum 8 metre face span*. From experience of mining in this type of ground, it is known that the span between the face and some substantial 'cushioning' support, such as timber packs or fill, should not exceed eight metres in order to preserve the integrity of the hangingwall. The hangingwall in the face area itself will require evenly distributed, though lighter support, such as rock bolts.
- *Stope support*. The type of stope support used must not require major cage, rail or vehicle transport as the infrastructure available will not be able to cope with it.
- *Exposure of personnel*. No person is allowed to work in an unsupported area.
- *Mechanized operation*. The whole operation must be

mechanized as the human resources will be extremely limited.

- *Rate of extraction*. A fast rate of extraction of the reef horizon is of paramount importance, not only for the sake of making it cost effective but also to destress other reefs below, allowing them to come into production timeously.
- *Optimize available ventilation*. The ventilation available to the area will be limited and it is therefore important that the mining method must be able to optimize the use thereof.
- *Simplicity*. The method used must be fairly simple, not requiring extraordinary skills or equipment.
- *Practicality*. To the best knowledge of the team the proposed layout must be practical. (If use is to be made of long hole drilling then what is a practical limit for straight holes?) The equipment must not be expected to perform outside its normal operating parameters. It is still only a stoping layout and the major issues are usually will the equipment be able to practically operate in the defined environment (what is the maximum incline for trackless equipment); and how the layout will cope with variations in geology, grade and rock structure. At all times the various proposals are tested to ensure that they do not contravene the untouchables.

Development of mechanized resue mining process

The best way to describe the mining method is to consider a strike gully approximately 60 metres long that will be advanced on an apparent dip in an up dip direction. The strike gully or resue face is sized to take mechanized equipment and will always include the reef to be mined and a section of footwall waste. Working from an existing excavation, which encompasses the reef and extends into the footwall, the reef in the sidewall of the gully is drilled, blasted into the gully and cleaned. The waste in the sidewall of the gully is drilled, charged up and blasted simultaneously. With the use of millisecond delays, the waste is initiated first and thrown into the gully creating backfill. After the waste has moved, the reef is popped down. The next step is to load the reef by LHD. The area blasted then becomes the new working surface for the process to be repeated. Thus, the image should be of a relatively large strike gully which is moving sideways, in steps, in an up dip direction. On the down dip side of this gully the stope is packed with blasted backfill. Access to the centre of the resue face is by means of an opening, which is left in the backfill as the face advances. This access will be on an apparent or true dip suitable for travel by trackless equipment.

The cross-section of the stope excavation is a function of the size of the trackless equipment and the channel width of the reef to be mined.

To achieve a well packed blasted backfill a sufficient volume of waste rock has to be blasted which will fill the volume that was previously occupied by waste and reef. It is normal to assume that broken rock in gold mines has a bulking factor of 1,6 and a density of 1,65 tons per cubic metre, however, this is a figure used as a skip factor. Experience in blasted backfill in the old COMRO days showed that the density of blasted fill was as high as 1,9 tons per cubic metre. In this exercise it was assumed that the blasted fill would have a density of 1,8 tons per cubic metre or a bulking factor of 1,5. Thus, the volume of solid waste rock, which must be broken to fill the equivalent empty space of waste rock and the empty space where the reef was, is twice the solid volume of the reef excavation. Knowing the thickness of the reef, the width of the excavation and the broken rock bulking factor it is now possible to calculate the dimensions of the excavation.

Preliminary assessment of the drilling and blasting parameters to be used for mechanized resue mining in deep level mining

A collaborative effort by personnel from Target Gold Mine and Sandvik Tamrock has resulted in the design of a method of resue mining which can be totally mechanized. African Explosive Limited, AEL, have joined this team to promote the concept that a good blast design matched to the geometry and rock conditions with sound underground mining practices will benefit this method.

This document lays out the foundations for a blast design based on information provided at several meetings.

Explosives and initiating systems

To compliment the mechanization the chosen explosives and initiating systems are:

- UBST
  - Underground Bulk Systems Technology, pumping systems to allow remote loading and avoid problems associated with storage and transport of explosives.
- Electronic detonators
  - Accurate and flexible timing, with a single unit inventory.

The total panel length is estimated to be 60 m, with 30 m blasted per mining cycle. The free face will be created with a wedge cut, with the waste fired in millisecond intervals to maximize throw. The reef is to be dropped into the void created by the removal of the waste, to encourage large fragmentation using long time intervals. The blasting will be completed in one operation.

The requirements for breaking the reef are:

- Coarse fragmentation to minimize gold losses

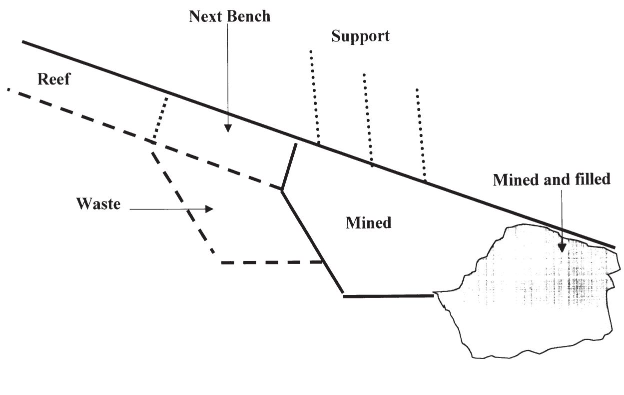

Figure 1—Section showing mining sequence for resue mining

- Minimal throw to drop the broken ore into the 'gully' to reduce contamination
- 2.0 m face advance per round.

The use of a single row of holes is suggested to meet these requirements with a +43 mm diameter hole on a burden of 0.97 m. Control over the direction of throw requires the holes to be drilled at 90°.

A requirement is coarse fragmentation to reduce dilution and contamination. The powder factor has been reduced to allow the burden to be opened up to match the waste parameters and optimize utilization of the drill rig.

Depending upon the limitation of the drilling equipment these holes should be stood off as close as possible to the final hangingwall position. A starting point would be 0.2 m with the option to review the overreach and adjust accordingly.

To minimize the handling of waste:

- Suitable fragmentation, to minimize volume of voids within the broken waste
- Maximize the throw to compact ore into the previous 'gully' and minimize waste handling.

The use of a square pattern is suggested to meet these requirements a +43 mm diameter hole on a marked burden of 0.97 m. We need to ensure that the waste is thrown and compacted, thus control over the direction of throw requires the holes to be drilled at 50°.

Observations when blasting with Shocktube indicate a throw of 7 to 15 m. The broken rock remaining on the footwall, i.e. contamination, is estimated to be between 5 to 10%. To aid movement of the broken waste we are looking for finer fragmentation. (Comparison of fuse and igniter cord vs. Shocktube, underground fragmentation project.) Early results from trials on electronic detonators indicate a potential to further control fragmentation using accurate timing and sequential firing. The timing will be in milliseconds to increase throw and compaction of the broken ore.

When using fuse and igniter cord underground, in the narrow reef environment, observations indicate the movement of the broken ore to be between 3 to 6 m.

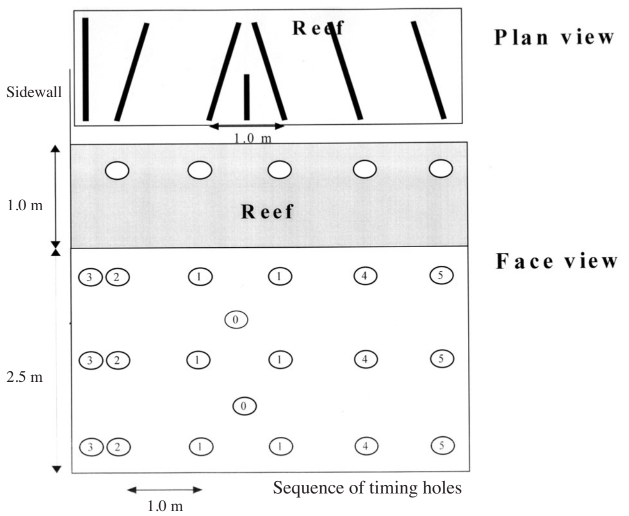

Figure 2—Drilling of the cut holes

The area at the top end of the panel is estimated to have between 5 to 20% contamination depending upon drilling accuracy and out of sequence shots.

The electronic detonators is the only initation system that will ensure sequential firing and the desired result, separation of the waste from the reef. Coupled with the accurate drilling the contamination is expected to be less than 5%. We will be using second timing intervals, to move the broken ore into the 'gully', based on observations of the muckpile profile.

The cut is the most critical element, if this does not break clean then no free face will be generated for the remainder of the blast. This will hamper the movement of the waste rock and make drilling of the next blast extremely difficult.

A wedge cut will complement the drilling and is shown in the following diagram. The centre holes drilled at 90° will be charged to 30% of the normal hole charge and a depth of 50% of the other holes.

Support requirement and specification

The specified support is 1.8 metre long split sets spaced at 1,5 m by 1,5 m.

Equipment selection

The appropriate equipment is now selected. The focus is not on a particular manufacturer but rather on the functional specification for the equipment and how it impacts on the proposed stoping layout. The equipment is now fitted into the stope excavations i.e. the stope excavations are sized to accommodate the equipment.

The mining method, in order to be as flexible as possible, should be capable of operating in as wide a reef geometry as possible. Accepting a 1.6 m minimum height of drilling and

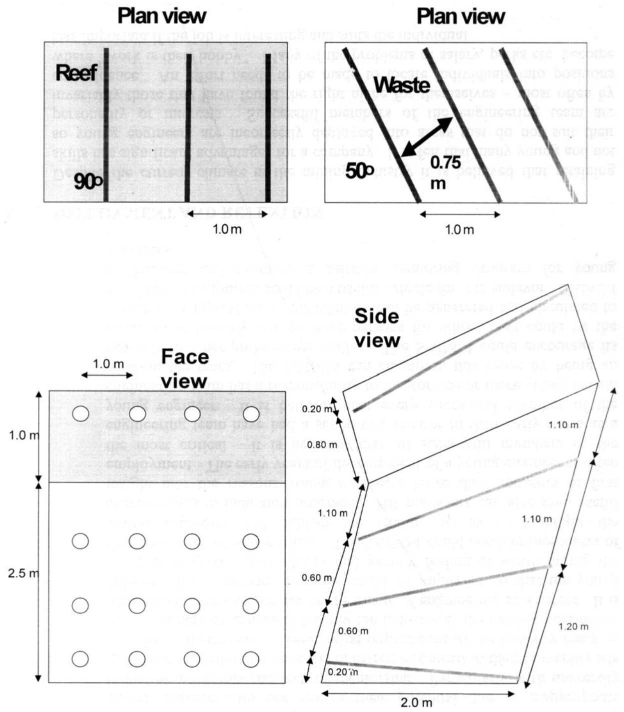

Figure 3—Drilling of reef and waste holes

loading equipment, and a highwall of less than 3.0 m for safety reasons.

A matched fleet of equipment was determined to be as follows:

- ➤ One small single boom face drill rig to be used to drill both the reef and the waste
- ➤ One 3,5t LHD to be used to clean the reef and tram it to the tips
- ➤ One utility vehicle used mainly for transport of explosives, charging up and transport and installation of services
- ➤ One roof bolter the same carrier size as the drill rig.

Definition of mining district

Based on the expected performance of the individual pieces of equipment an equipment fleet is identified. Every effort is made to optimize the production from the fleet. Rock handling capability is matched to rock breaking rate. Support is integrated into the mining cycle. It is at this stage that the stoping system and technology requirements are defined.

Based on the drilling, charging, supporting and rock handling requirements specified in the earlier part of this report the duration of the various activities required to advance a 30 m faces were estimated to be as follows.

	Hours
Drill reef and waste	5,25
Charge	4,00
Clean reef	4,00
Support	5,00

The total cycle of activities consists of drilling and blasting the reef advance of 2,0 metres in one operation, cleaning and levelling, and supporting the new excavation for the cycle to repeat itself. The face supplied by one access way will be approximately 60 metres long and the cycle is arranged so that on a shift either drilling, cleaning or bolting is taking place on a panel. Within a period of 3 shifts the 60 metre face is advanced 2,0 metres. The corresponding performance figures below are based on a particular mining operation and calculated on the assumption of a 26-day month.

Number of crews	3
Crew size	8
Face advance per month	27,7 m
Square metres per month	1662
Square metres per man per month	69
Tons per month (including 15% dilution)	5256
Tons per man per month	219

Stoping costs

The capital and operating costs of the equipment are now determined based on experience and historical data from similar environments. Dilution and manning levels are often major issues at this stage. These cost and performance figures are compared to conventional mining costs. *The new system has got to be more cost effective than the old system.*

Based on the equipment fleet selected, the operating costs are estimated to be:

	Rand/ton
Drilling	18-62
Charging	14–37
LHD cleaning	13–31
Roof bolting	13–26
Services	8–26
Labour	51–27
Contingency	11–90
Total Stoping Cost	**130–99**

Conclusions

It is accepted that resue mining has a number of advantages to offer, particularly in mining narrow reefs. However, there are also a number of disadvantages with the intermittent nature of the operation and gold loss probably being the most important. The challenge is to take a basically good idea and determine if, through the application of appropriate technology, the advantages can be maximized and the disadvantages minimized.

A novel method of Mechanized Resue Mining has been proposed. The important factors, or untouchables, were defined and a practical stoping layout developed. Appropriate equipment was selected and the area that could be mined by a specified fleet was defined. The performance of the system has been estimated as has the operating cost. Most of the development is on reef and a good quality backfill is installed. The stoping cost of R130 per ton is very attractive.

The next step is to validate the calculations and assumptions through a trial in a representative part of a mine.

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International Standard (I.S.) and the *Journal*

One of our readers, P.J. Lloyd has e-mailed a comment on an article in the May/June, 1999, issue of the *Journal*. The article 'Acid mine drainage in Virginian Coal Mines' was used, in keeping with editorial policy, to keep our readers in touch with world-wide opinion and technology.

I had thought the Journal's *policy was SI and only SI. In the Transactions Section in the May-June issue I find:*

p 167 Kcal/kg; p 170 gr/cm2, Gs, kg/cm^2*; p 175 miles; p 176 ACFM, gallon, mg/L; p 177 mg/L, foot, gallons per minute, psi, feet, hp, gal/min, ft., inch; p 178 miles.*

What kind of gallons? What kind of tons, for that matter? What is 'Gs'? And all of this in a section we proudly announce as 'refereed and edited according to Internationally accepted standards'.

Can we please have consistent units to the aforesaid international standard?

Editor's reply to P.J. Lloyd

It is necessary to reply to this letter so as to provide guidance to referees and authors.

No instruction has ever been issued to referees or proof readers to the effect that exclusive adherence to the SI unit system should be a criterion for assessing the value or international acceptability of a paper in the Transaction section of the *Journal*. The criterion to be used relate, for example, to the novelty and value of the contribution to the mining and metallurgical community and the quality and validity of experimental results and conclusions.

To a large extent, we leave to the authors, the selection of appropriate units. We do not accept that international standards require exclusive and slavish use of the SI unit system. If an author, for example, wishes to refer to the price of gold in 'dollars per ounce' (a well understood term throughout the world) we would neither reject the paper on this account nor slavishly convert such expressions to SI units. Similarly a paper of, say, American origin using American units can represent a valuable contribution without going through a conversion exercise.

However, we accept that any ambiguity in the use of units should be avoided. If the context of the paper does not make it clear what units are being used, we will request the author to make suitable modifications.

The bulk of Philip Lloyd's criticism is directed at a paper written by an American author and very obviously relating to American practice in handling Acid Mine Drainage in Virginian coal mines. It is presented under the heading 'Green Pages' and clearly was not a paper in the Transaction section. We are surprised that Philip Lloyd, with his MIT chemical engineering background, finds it difficult to appreciate that gallons and tons, in this context, are in American units.

We thank Philip Lloyd and would appreciate more comments from our readers on articles in the *Journal*—even if its only 'tilting at windmills'.

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Founded in 1894

OBITUARY

		Date of Election	Date Deceased
M.A.H. Harris	Member	17 June 1968	10 December 1999
J.L. Van Eyssen	Retired Fellow	25 January 1950	5 November 1999