An economic model for gold and platinum mining using selective blast mining

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Synopsis

Selective Blast Mining (SBM) makes use of milli-second sequential blasting technology to separate the valuable reef material from waste rock in the blasting operation at stope faces in the mining of tabular ore bodies such as exist in the gold and platinum mines in South Africa. The waste rock is cast blast to the back of the stoping area, and the reef material is fractured and deposited close to the stope face. Only the reef material is transported to the surface. A simplified economic model is presented to evaluate the economic benefits of SBM. Although premature, in that adequate experimental demonstration of the validity of the model has not yet been possible, the model is presented since it indicates substantial benefits in the case of narrow reef gold mines. It is suggested that many marginal mines should seriously consider SBM. Benefits can also be realised in platinum mines, but a lot more experimental work is necessary to establish cost parameters more reliably. Some practical experience in the use of SBM in the mining of the UG2 reef is presented.

Introduction

This paper may be considered premature in that it is not yet possible to quote sufficient results to demonstrate the validity of the models proposed. However, many marginal mines and shafts which might benefit from Selective Blast Mining (SBM), are either closing or considering closure. The authors believe that sufficient test work has been done to show that SBM does work in practice and that decision makers should be aware of the potential and give the methods a trial, since the benefits appear to be substantial.

The concept of Selective Blast Mining (SBM) was put forward by Bock and Robinson in August 1994 and reported at a symposium in 1996[1]. The features of SBM and its range of applicability can be summarized as follows:

- It is applicable to the underground mining of tabular bodies (reef formations) using stope face advance by blasting.
- The valuable constituents should be concentrated in a 'reef' demarcated from the waste rock by identifiable boundaries.
- The reef width should be less than the stoping width, usually but not necessarily by a factor of approximately 2.

A blasting sequence takes place in two consecutive steps spaced ± $^{1}/_{2}$ sec. apart. The waste rock above (or below) the reef is blasted separately from the reef and deposited in the stope area well away from the stope face. Immediately after the waste blast, the reef is blasted with the minimum energy to dislodge the material and deposits it separately from the waste in a position close to the stope face where it can be readily removed from the stoping area.

Although a crude form of SBM, known as Resue Mining, was practised some 30 years ago, blasting of the waste and the reef took place in separate blasts with a time interval between them so as to enable the waste material to be cleared by hand to make space for the second blast of the reef.

The ability to undertake SBM in one blast sequence has come about by the advent of sequential milli-second blasting technology in a reliable and cost-effective system. Such technology is required to conduct controlled 'cast blasting' of the waste material in the sense that the material is to be cast a considerable distance away from the stope face (5–20 metres) leaving an open space for the reef blast. This latter blast merely fractures the rock without significant spatial displacement.

An essential feature of SBM is the ability to create a split plane between the reef and the waste rock.

The basic features of SBM are illustrated in Figure 1, showing a vertical section through a stoping operation with the waste rock deposited at the back of the stope area and the valuable reef material on the floor close to the

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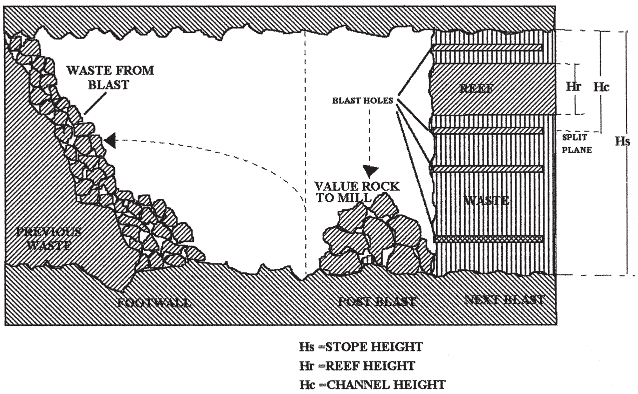

Figure 1—Selective blast mining. Section through stope

Figure 2—Waste rock thrown against the back of the stope

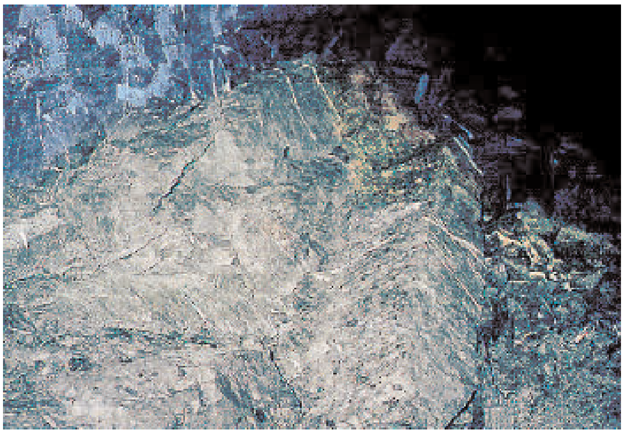

Figure 3—The reef dropped at the stope face

stope face. SBM can be used for many different mining patterns involving different configurations of stope panels, gulleys, pillars and haulage ways, and for different dip angles. In Figure 2 and Figure 3 photographs are shown of the waste pile at the back of the stope and the reef material at the stope face respectively. These were taken at a West Rand shaft from which data were obtained for the model.

The objective of this paper is to assess realistically the potential economic advantages of SBM and to direct attention to the important parameters requiring further investigation.

It is not the intention of this paper to discuss in detail the merits of different milli-second blasting systems, nor the electronic techniques involved. However, in the costing model, which is the main feature of this paper, input cost data have been used for one proprietary blasting system which is commercially available and which has been used sufficiently in practice to demonstrate the reliability of performance and pricing. Limited tests were conducted at Western Deep Levels and at a mine on the West Rand from which cost data for the gold model were obtained. Extensive tests are being carried out at No. 6 shaft at Impala Platinum Mines and the following section gives some observations made regarding these tests.

Observations on the use of SBM at Impala Platinum No. 6 shaft

The pyroxenite/pegmatoidal rock type is easier and faster to drill than the quartzites of the gold mines but it is more difficult to break. The burdens between holes are thus much closer on the platinum-bearing reefs than on the gold mines and this impacts on the igniter cord and connector spacings. The chances of out-of-sequence firing of shot holes thus increases because the variations in the burning speeds of igniter cord and capped fuses is of such a nature that it is virtually impossible to eradicate.

During the recent past, Impala has made tremendous strides on the productivity front by means of the introduction of the Crew Captain and small team concept (or panel miner as it is known on the gold mines). Daily blasts per panel are common and the subsequent stage of maximizing advance per blast became the next critical success factor. When analysing the face advance rates achieved, it soon becomes obvious that out-of-sequence firing is a larger problem than what is commonly accepted. There are many instances of misfires, bumps in the stoping face, etc., resulting from poor or incorrect timing due to wrong interconnector spacing or excessive variations in the burning speeds of the explosives accessories.

The resultant poor face shape makes it difficult to clean and of course impacts on the subsequent blasts, the situation getting worse and worse until the advancing stope face is stopped, a reference line inserted and the stoping face pulled back into a straight line.

Theoretically, if all is done correctly as per code of practice or mine standard then this situation should not arise. Unfortunately, with narrow reef mining there is also a difficulty in ensuring the 100% hole accuracy required from the rock drill operators because of the small burdens, temporary support on face, machine and airleg having a combined length of 2.5m, and the driller having no assistant.

Various experiments were conducted in an effort to find a solution to obtaining perfect sequential firing. These included fuse and igniter cord imported from Chile and various electronic blasting systems.

During this experimentation, problems were encountered in one of the sites with the immediate hangingwall above the UG2 reef because of three leader chromitite layers being in very close proximity to the upper contact of the reef channel.

Falls of ground were common resulting in excess dilution and mining delays with the obvious negative impact on profitability. When an attempt was made to mine these layers together with the reef channel, it became unpayable. It was then proposed that resue mining be attempted, but in a single blast sequence i.e. SBM.

Although there was much scepticism regarding selective blast mining, the technical representatives were adamant that it could be done and the go-ahead was given to use the Zedet system for operating trials.

The initial blasts were highly successful in terms of sequential blasting but removed all support from the face area. Many schemes and various methods were tried in an attempt to ensure some means of support existed after the blast (between the advancing face and the pile of waste blasted into the back area). This included pre-stressed packs, pre-stressed elongates, etc.

Initially it was hoped that the blasted waste could act as a support medium in the back area, especially if sufficiently compacted by the blast. However, the solid pyroxenite hangingwall beam (±3 m thick) also has no convergence and thus this had to be discounted for use as a support medium. Eventually 12 mm rebars inserted into holes with cement grout capsules was the only viable means of support when combined with the normal mine standard grid pillar system. The resultant stope width of 1.6 m allowed for this possibility. Rock engineers did the necessary calculations and a standard was agreed upon.

The UG2 reef in this specific area has a reef width of 65 cm and a channel width of 76 cm (i.e. 11 cm overbreak is allowed for practical mining purposes). The leader chromitite layers have no natural parting except against the hangingwall pyroxenite which is ±1 m above the UG2 reef. When this collapses, it causes mining delays and dangerous situations.

The current technique has evolved to a point where it was accepted that 100% pure reef would not be obtained due to splitting on this unnatural parting. Nevertheless, because of the mining method the overall reef extraction in this mining area has improved. No mining delays (i.e. the largest direct cost factor of labour is used to maximum advantage) resulted ultimately in a reduction of the mining pay limit.

With the larger stoping width the hole burdens could be opened up to nearly double that previously required, face advance can be increased to 1.5 or 1.8 m/blast (less labour used per unit of production) and the economic benefits illustrated in Table II could be achieved if this is applied to a complete shaft, mining on the UG2 reef horizon.

The current technique has been applied to the specific area as explained but will not be suitable for the majority of the platinum reefs in the current format, i.e. blasting out of the uneconomic portion above the reef into the back area and then ‘popping up’ the reef. Rather, the reverse method of blasting the uneconomic portion out from below the reef and then just ‘dropping down’ the reef (as in the gold mining example) would be more suitable. This is the subject of the next phase of the experiment.

The possible impact of selective blasting on the South African mining industry cannot be overestimated. The possible use of machinery in these larger working widths for drilling and cleaning can also now be more readily applied.

Further advantages from the current limited use of the selective blasting technique has been the improvement of stope face ventilation due to increased velocities of the ventilating air without having the difficulty of installing ventilation curtaining on stope faces. The back areas are also safer since the area is ‘closed off’ by broken rock and does not require barricading to prevent access, etc..

One is essentially left with a choice if current infrastructure on shafts and milling plants is considered, i.e. either less tonnage at the higher grade can be mined with obvious savings in overheads and plant, or plant could be optimally utilized by increasing reef mined and the Net Present Value of the venture vastly improved.

The application of SBM and the relevance of the costing model could apply to several other mining activities such as narrow seam coal or chromite mining.

Cost advantages and disadvantages of SBM

These are listed below and form the basis of the cost models:

Grade improvement

SBM reduces the dilution of the valuable constituents in the reef by valueless waste material. Thus the grade of the material to be hoisted to surface and delivered to the metallurgical recovery plant increases.

Reduced materials transport

After each blast a smaller quantity of reef material has to be moved away from each unit length of stope face. The waste material is thrown against the back of the stope and requires very little or no further handling.

Roof support, seismicity and rockburst control

With SBM the specific explosive energy used is reduced. Measurements of ground vibration have shown that the peak vibration levels with SBM are no more than those with a single shot. In addition, the low frequency rumble that normally occurs after single shots is totally absent with SBM. Of particular importance in many applications, is the fact that damage to the hangingwall can be minimized, creating much safer working conditions in the active area of the stope. The waste material can act as a spatially continuous roof support behind the working stope faces. This feature should make a contribution towards avoiding roof collapse in worked-out areas. There is a possibility that the roof support in the form of unmined pillars, wood-packs and mechanical or hydraulic props can be done away with in the worked-out areas of the mine, but more information on strength of broken rock support is required. The possibility of using a cement-like filler in the voids to provide much greater strength has been suggested.

Reduced losses and improved mine call factors

The effect of the blasting action on the loss of valuable constituents in the reef is believed to be much less as compared with conventional blasting. This reduces and possibly avoids completely the losses of small particles of gold, which are liberated at the fracture surfaces, and are dispersed by the high velocity explosion gasses into remote areas of the mine.

Saving in ventilation costs

The costs for ventilation and cooling are less for SBM because the backfill reduces the volume of space for the circulation of air and reduces the surface of rock that acts as heat transfer areas.

Working conditions

In many cases stoping heights can be increased significantly without excessive cost penalties so that working conditions, particularly as regards machine drilling, are much improved. The use of low height vehicles for material handling becomes possible.

Drilling accuracy

SBM requires greater accuracy in the drilling of blasting holes. Mechanical drilling methods can be used. This requires more working space and slightly increased costs.

Personnel training

Different procedures for drilling, charging and connecting detonators are required. Although, in many respects SBM is simpler than pyrotechnic methods, it requires retraining and re-certification of personnel and operating procedures.

Metallurgical processing

Some revisions of milling and metallurgical processing techniques may be necessary, for example, in the replacement of autogenous by semi-autogenous milling. However, SBM could possibly give rise to advantages with an improvement in metallurgical efficiency and costs.

Cost of initiation systems

The cost per hole blasted is increased by the cost of initiation systems and electric detonators.

Cost model

There are several different ways of calculating and expressing the cost components of an SBM operation. For example, they can be in terms of cost per tonnage milled, the cost per cubic metre of stope face or the cost per square metre of stope mined.

The preferred method in this paper is to relate all costs to the final quantities of gold or Platinum Group Elements (PGEs) produced. Thus, for comparison purposes, costs for the different categories of the mining operation will be expressed in terms of rands per gram of gold or PGEs finally produced. The comparison is made specifically for one shaft of a mine extracting from one ore horizon. Although this implies some arbitrary assumptions in allocating overheads and capital costs, this limitation avoids the advantages of SBM being submerged in the total mine operations when other shafts are not using SBM.

An arbitrary milling capacity of 100 000 t/month/shaft is used for illustrative purposes.

Grade, reef width, channel width and stoping height

The relationship between the grade, the reef width, the channel width and stoping height is the most important consideration in the cost model for SBM. Conventionally, the 'grade values' are expressed in terms of units such as *cm.gm/ton* or *metre.gm/ton*. In conventional mining, where all the material blasted from the stope face was sent to the recovery plant, the concentration of precious metals could be calculated by dividing the grade value by the stoping height. In the case of SBM, the corresponding value would be the grade value divided by the 'channel width'. This latter term is defined as the width of that portion of the stope face blasted selectively and separately from the waste material and which is sent to the surface for metallurgical treatment, as shown in Figure 1.

In terms of the cost model, the gold content of the material within the channel width can be expressed as:

$Gc = V/Hc$

and

$Hc/Hs = R$ the 'selective mining ratio'

where

G is the grade of material in the channel section in grams/ton

V is the grade value in metre.grams/ton

Hs is the stoping height in metres.

Hc is the channel width in metres.

The gold is almost invariably contained in a conglomerate band with a distinctive appearance. In some cases, the gold is concentrated in pencil-thin sections of this conglomerate band; in others, it is fairly evenly distributed over the conglomerate. Very rarely indeed is the gold present outside the conglomerate. In SBM, it is important to recover the whole conglomerate band plus a small portion of the waste above and below the conglomerate to ensure that no gold is discarded in the waste material.

In the case of the Merensky and UG2 reefs mined for platinum, the distinction between the PGE concentrated in a specific reef structure and the waste material is not as clearly defined and this aspect is discussed later.

Optimal values for the channel width, Hc

When rock is broken, the particles, when randomly packed, occupy a greater space than the unbroken rock due to the creation of a number of voids. This expansion in volume is approximately 40%, but could be as low as 30% for certain particle size distributions and forced packing. If the open space created behind the stope face is to be just filled by the waste then the volume taken to the surface should be equal to the voids in the waste material. Thus, for uniform face advance, the ideal value of Hc is

$$Hc = Fv.(Hs\text{-}Hc)$$
$$Hs = Hc(I+Fv)/Fv$$

where Fv is the fraction of voids in the broken and packed waste rock.

If the conglomerates or reefs containing the gold or PGEs are wider than, say, 20 cms, the optimal ratio can be maintained by increasing stoping height (Hs) to, say, 125 cms. If the slope height is increased, the face advance can be increased proportionately by drilling deeper holes. As a first approximation the face advance would be 0.9 Hs. In practice, it is not invariably an exact ratio. The conglomerate bands and the reefs are never uniform in width, nor completely straight, nor horizontal, and the blasting pattern must, and can, follow the contour of these bands. Also, the choice of whether the bands are at the top of the stope or at the bottom depends on the nature of the rock strata. If the rock above the conglomerate is of a low strength material such as clay or shale it is of benefit to have the reef width at the top of the stope. This is to ensure that the violent high intensity cast blast can take place in the footwall and the reef blast (which must be very much less severe so that the valuable material merely drops to the floor) does minimal damage to the roof. All these considerations lead to the conclusion that each section of the mine must decide by experimentation on the optimal choice of channel and stoping widths, provided that neither too much nor too little waste is left underground.

A selective mining ratio (R) can be defined as:

$R = Hc/Hs.$

Stoping costs

The costs involved in stoping operations such as drilling, explosives, detonators, initiation systems, roof support development and an appropriate proportion of labour, development ventilation, management and engineering overheads are conveniently grouped together as stoping costs. Conventionally, these are calculated in terms of costs per square metre of stope area or per metre of panel per blast. For this model, all these units should be converted into the costs per ton of rock blasted using equations such as

$Cb = Cbx.Fa.Hs.Ss.$

Where Cbx is the stoping cost in rands per metre panel width per blast, Cb is the stope blasting cost in rands per total ton blasted, Fa is the face advance in metres and Ss is the average specific gravity of the rock. The equivalent stoping cost *per ton milled*, Cs will be

$Cs = Cb.(Hs/Hc).(Sc/Sw)$

where Sc and Sw are the specific gravities of the channel and waste material respectively. Within the limits of the accuracy of this model, these can be considered to be equal so that

$Cs = Cb/R.$

Costs of transport, hoisting, milling, extraction and refining

Conventionally all these costs, plus appropriate overheads, are readily available from the mine data and are expressed in the appropriate units of a cost per ton milled and can be designated as

$Cr = \Sigma\, Cn.$

where Cn are the individual component costs as indicated above.

Precious metal losses

In many gold mines and particularly where fine gold is associated with carbon, there is a discrepancy between the gold that should have been produced from underground and that actually delivered to the mill. This discrepancy is referred to as the 'mine call factor' which in extreme cases can be as low as 0.75, i.e. only a 75% recovery. There is little doubt that a significant proportion of this missing gold arises from two well-established effects.

- On blasting, fractures in the conglomerate occur between the quartz pebbles to liberate the matrix containing the gold usually in a finely divided form.
- Immediately after fracture, the gasses formed from the explosives are expelled at a supersonic velocity many orders of magnitude higher than the Stokes settling velocity of the gold particles (particularly if they attached to the light-weight carbon particles). The gasses carry the gold away from the collection area in the stope. After the blast, gold can also be lost by the action of water or by falling into crevices in the stope floor.

In SBM the second blast can be of a very low energy, to merely fracture the rock without any displacement away from the stope face so that gold losses can be kept to a minimum—if not eliminated. An accurate evaluation of reduced gold losses can only be made after many months of operation and careful measurement of mine call factors before and after the introduction of SBM. However, the losses are theoretically highly significant cost-wise and, in the models provided, some illustrated values are used to show the importance and potential benefits that might be achieved. If Mf is the mine call factor (expressed as the fraction of gold sent to the mill as compared to the theoretical amount estimated to be in the stope face), then the grade of the material sent to the mill will be reduced according to the equation

$Gm = Gc.Mf.$

Once the ore has reached the mill, the recovery of gold in the metallurgical processing can, in most cases, be considered to be 100%. This is, however, not the case in certain refractory ores, nor for the PGEs. In these cases, the model allows for a metallurgical recovery factor, Mr, so that

$Gr = Gm.Mr$

where Gr is the grade of the mill feed in terms of recoverable gold or PGEs.

Capital cost considerations

The allocation of capital costs in any mining activity is a complex matter involving, for example, taxation formulae and shareholder criteria. It is not the intention in this paper to go into such complexity, but some of the more substantial benefits of SBM arise by way of an increased rate of gold production or a decrease in the paylimits and an increase in reserves. Strictly speaking these can only be evaluated on a Return On Investment analysis. To avoid undue complexity, evaluation is done on a comparative basis between conventional and SBM methods for illustrative purposes.

It is assumed and proposed that a 'capital allocation' can be made on the basis of

- the total gold or PGE reserves estimated at any given cost structure and gold or PGE price,
- the rate at which the gold is to be mined, and
- the capital cost required to establish the mine shaft.

This capital allocation can be expressed as a cost per gram of gold or PGEs.

If this capital allocation is designated by Q in the base case (conventional mining) and if the rate of gold production is increased by a factor X using SBM then a simplified estimate of the benefits will be a reduced capital allocation of

Q/X *i.e.*

Benefits = $Q.(X-1)/X.$

Table I

Illustrative cost model for selective blast mining: Gold

Cost parameters	Symbol	Units	Source	Convtl	Sbm
GENERAL DATA					
Dollar price of gold	Au	$/oz	In data	300	300
Rand/dollar exchange rate	RE	R/$	In data	9	4 9
Rand price for gold	Au/g	R/g	Au*RE/3	47 27	47 2
Tonnage milled per shaft per month	TM	Ktons	In data	00	100
Mine call factor	M		In data	0 8	0 9
Metallurgical call factor	M		In data		
Average ore value for shaft	V	in g/t	In data	5	5
STOPING CONDITIONS AND COSTS					
Reef height	H	meters	In data	0 3	0 3
Channel height	Hc	meters	In data		0.4
Void fraction in waste rock	Fv_		In data	0.45	0.45
Stoping height	Hs	meters	Hc(1-Fv)/Fv		29
Face advance (hole depth)	Fa	meters	Hc*0.9		6
Panel length	Pl	meters	In data	30	30
Specific gravity of rock	SG		In data	2 7	2 7
No of holes drilled per meter panel length	N		In data	3	4
No of holes initiated per meter panel length	Ni		In data	3	3
Explosive cost per hole	Ec	rands	In data	25	25
Initiation costs per hole	Ic	rands	In data	2 4	9
Drilling cost per hole	Dc	rands	In data	6	6
Stoping costs in R/blast / meter panel					
Drilling	Ds	R/b/m	Dc*Nh	8	24
Initiation	Is		Ic*Ni	7 2	27
Explosives	Es		Ec*Ni	3 75	3 75
Stope labour	Ls		In data	8	8
Stope supervision	Sp			5	5
Roof support	Rs			23	10
Stope development	Sd			90	90
Total stoping costs in R/blast/m panel	Cbx		SUM Ds to Sd	6 45	74 25
NON STOPING COSTS (Transp & Recov.)					
Scraping		R/t milled	In data		
U/G tramming				3 5	3.5
Plant tramming					
Tipping and hoisting				4	4
Milling extraction and refining				8	18
Labour				8	8
Supervision				5	5
Development, shaft and tunnels				5	5
Support costs				2	?
Mine management				0	0
Engineering and finance				0	0
Head office and marketing				5	5
Total non stoping costs	Cm		SUMns costs	2 5	72 5
Ratio tons blasted to tons milled	R		Hs/Hc		3 22
Stoping costs per ton milled	Cs		Cbx R/Hs/Fa/SG	65	39
OVERALL COSTS AND PROFITS					
Total cost per ton milled	r		Cm+Cs	32	2 2
Gold content per ton milled	Gm		V*MF*M/Hc	2 4	6.75
Gold [illegible] and sold [illegible] ton milled				2 4	
Total cost per gram of gold sold	Cg		Ct/G	55 2	3 3
NET PROFIT PER GRAM OF GOLD	Pg	r/g	Au/g-Cg	-7 9	5 9
NET PROFIT PER TON MILLED	Pt	r		9	07
PAYLIMIT	Vl	in g/t		**3 50**	**1 99**
TOTAL SALES PER MONTH		Rmillion/m		**1.3**	**31.9**
MON	Pm	R million			
Gold produced per month Kgrams				240	675
[illegible]nefits	Q1	R million/m	Q /X		
Capital allocation -Reserve benefits	Q2	Rmillion/m	Q2/Y	2 4	1.36
Net extra [illegible] from reduced capital allocation		Rmillion/m			**2.58**

Table II

Theoretical cost model for PGEs in UG2 reefs

Cost parameters	Symbol	Units	Source	Convtl	SBM
GENERAL DATA					
Dollar price of PGE's	PGE's	$/oz	n data	263	263
Rand/dollar exchange rate	RE	R/$	n data	4 9	4 9
Rand price for PGE's	PGE's,gm	R/g	PGE*RE/3	4 4	4 4
Tonnage milled per shaft per month	TM	Ktons	n data	[illegible]	[illegible]
Mine call factor	Mf		n data		0 9
Metallurgical call factor	Mr		n data	0 8	0 85
Average ore value for shaft	V	m/g/t	n data	5	5.1
STOPING CONDITIONS AND COSTS					
Reef height	H	meters	n data	0 65	0 65
Channel height	Hc	meters	n data	0 9	0 7
Void fraction in waste rock	Fv		n data	0 4	0 5
Stoping height	Hs	meters	n data	0 9	6
Face advance (hole depth)	Fa	meters	n data	0 35	1 2
Panel length	F	meters	n data	3	30
Specific gravity of waste rock	Sw		n data	3 2	3 2
Specific gravity of channel	Sc		n data	4 04	4 04
Specifis gravity of stope (average)	Ss		n data	3 93	85
Stoping costs in R/sq meter stope floor area					
Drill steel	Ds	R/5/m2		4 73	4 7
Explosives accessories	Is			3 85	34 4
Explosives	Es			09	13.67
Small stores	Ls			21 03	2 08
Roof support	Sp			8	7 82
Labour	s			97 5	57
Total stoping costs in R/sq meter floor area	Cpy			52.5	138 5
NON STOPING COSTS (Transport & Recovery)					
Total non stoping costs	Cm	R/t milled	n data	25	125
Ratio tons blasted to tons milled	R		sSs/HcSc		1 16
Stoping costs per ton blasted	Cb	R/t blasted	s/Ss	4	22 5
Stoping costs per ton milled	Cs	R/t milled	b*R	43	49 0
OVERALL COSTS AND PROFITS					
Total cost per ton milled	Ct	R/t milled	s+Cm	8 1	174 0
Pge s Sold per ton milled	Gr	g/t	V*Mf*Mr/Hc	4 53	5 57
Total cost per gram PGE's sold	Cg	R/g	Ct/G	37	31 2
Net profit per gram PGE's	Pp	R/g		4 4	10 2
Net profit per ton milled	Pt	R/t	G*Pp	9	57 0
Pay limit	V			4 56	3 84
Total sales per month		million	G*PGL/g*TM	3 8	23 1
per month	Pm	R million	P**TM	97	5 70

If as a result of SBM the pay limit is reduced to a lower value by a factor Y, then the benefits can be estimated by

$Q/f(Y)$.

Where $f(Y)$ is a function representing the increase in reserves as a result of the reduced pay limit, this function will vary from mine to mine and from shaft to shaft. For illustrative purposes a simple function can be assumed

$f(Y) = 1/Y$.

That is, for example, if the pay limit is reduced to half of the conventional value then the reserves would be double. The benefits would be a reduced capital allocation to

$Q.Y$ i.e.

Benefits = Q(1-Y)

Illustrative example of the cost model

A typical example for the case of a gold mine, is presented in detail in Table I. The figures used were obtained from a shaft on the West Rand where trials on SBM were being conducted before it was closed for economic viability reasons.

In this application of the model, two main components are identified.

➤ Non-stoping costs—these are costs involved in the handling of one ton of ore, which are independent on whether conventional mining or SBM is used. They include the costs of transporting material to the surface once it is moved from the stope face, the cost of milling and of extracting and refining the gold. The subcategories are illustrative and can be varied according to each mine's preferences.

- Cost factors that are critically dependent on the parameters chosen for selective blasting. These are referred to as stoping costs and are first indicated in terms of rands/blast/metre of stope panel. These are then converted to the cost in terms of rands/gram of gold using the formulae given previously.

In the example shown in Table I, it is assumed that the mine operates to maintain its hoisting and milling plants fully utilized.

Conclusions from the cost analysis for gold.

A dramatic change in the profitability is indicated when using SBM.

- The income per month improves from a loss of R1.9 million to a profit of R10.7 million.
- The 'pay-limit' decreases from a break-even value of 3.5 metre.grams/ton to 1.99 metre.grams/ton.
- The value of gold produced per month increases from 240 to 675 kg valued at R11.3 million and R31.9 million respectively.
- The impact on the capital cost allocation is a further profit benefit of R2.58 million (based on an arbitrary, but conservative allocation of R10 per gram of gold).

The reason for this improvement is clearly indicated by the model. Although the stoping costs increase by 132% for SBM the non-stoping costs remain constant and the total cost increases by only 61%. The gold content however increases by 181%.

It is of interest to calculate at what gold price the shaft would remain profitable if using SBM. *At a grade value of 3.0 metre.grams/ton the break-even gold price would be 171 $/oz.*

The model can be used to run sensitivities on the variables. By far the most sensitive parameter is the selective mining ratio, i.e. the channel height, to stope height, ratio. For those mines with a reef width greater than in this example, the channel height and stoping height could be increased. There is an advantage in increasing stope height (to say 1.5 metres) in that low-height vehicles could be used at the stope face to mechanize drilling, scraping and material transport of backfill.

The data supplied by the mine indicate a high 'stope development' cost, and this is typical for marginal mines. The decreased paylimit by using SBM could reduce this cost significantly. Another sensitive factor is the cost of roof support and this will be of particular importance in deep mines. The extent to which waste rock generated by SBM could act as roof support on its own account has still to be determined experimentally. A method for measuring the strength of such rock *in situ* has been devised and will be tested. Similarly, the extent to which gold losses can be reduced and the mine call factor improved is also to be determined.

Special cost considerations for modelling PGE mining

There are several important differences between gold and PGE mining in relation to SBM.

- Generally the PGE horizons are at much shallower depths than gold, so that roof supports and rock bursts are not as critical.
- The PGEs are not confined to only one specific reef segment but may occur in several reef segments adjacent to each other. The PGE values may even diffuse into surrounding waste rock. Typical PGE distributions in the Merensky reef are shown in Figure 4 and in the UG2 reefs in Figure 5.
- The metallurgical recovery of the PGEs is significantly less than in the case of gold, and can be affected by impurities present and by the grade of the mill feed, and the content of impurities such as chromite.

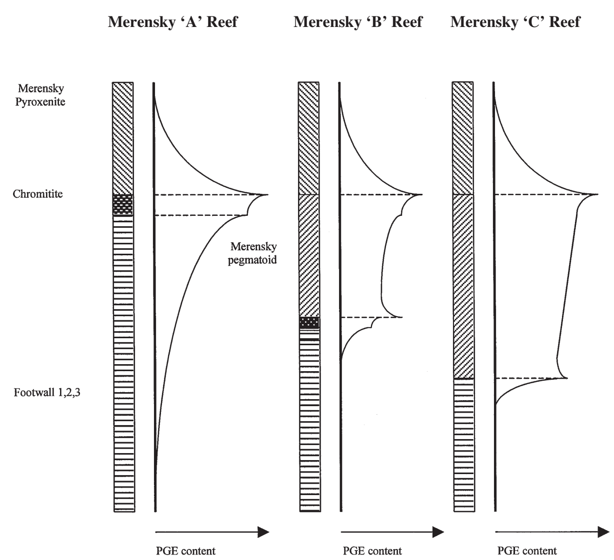

Figure 4—Typical PGE value distributions for Merensky 'A', 'B' and 'C' reefs

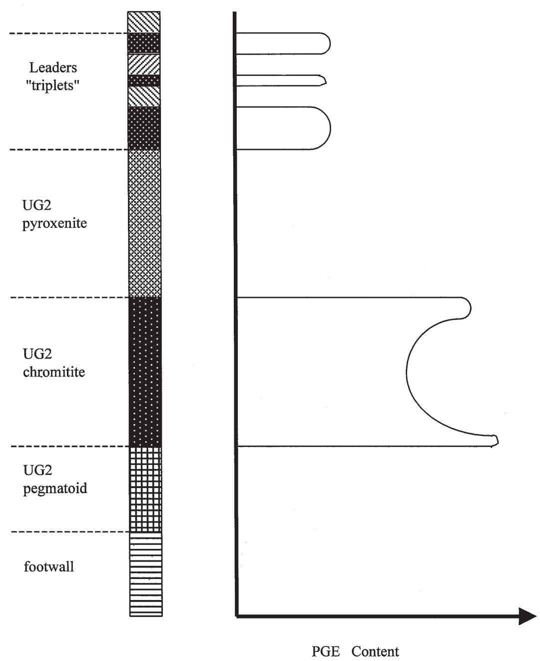

Figure 5—PGE value distribution for UG2 reefs

Therefore, the benefits to be derived from SBM are offset by the disadvantage that a certain quantity of PGEs will be left underground if one reduces the channel width to the same levels as might apply for gold.

In the case of the UG2 reefs, the PGE concentration boundaries are reasonably sharp, but the so-called 'triplets' contain PGEs and some of these will be included in the waste material.

In the case of the Merensky reef, the concentration profile is highly variable and the selection of a channel width to avoid undue losses becomes a difficult and complex task for the mining geologist. A great deal more empirical work is necessary before the benefits of SBM can be adequately modelled.

Despite these complexities, there can still be advantages in SBM, specifically for the UG2 reefs but also possibly for the Merensky reefs, depending on the spatial distribution of the platinum values. By increasing the stope height to values of up to 1.50 metres at relatively small incremental cost, a channel width of the order of 0.5–0.6 metres (depending on the void volume *Vr*) can be selectively brought to the surface. In many major areas of the mine, this could include a high proportion of the PGE values depending on the geology. For example, in the UG2 reef at No. 6 shaft, the channel width could include not only the UG2 reef but also the bottom chromitite triplet (see Figure 5). Consideration must also be given to the selection of the split plane to eliminate deleterious constituents being fed to the recovery plant and also to maintaining the most supportive roof structure.

Exactly the same cost model can be used as in the case of gold. The only considerations to be recognised are that:

- The 'Mine Call Factor', *Mf*, referred to in the gold model, can be considered as a PGE 'recovery' factor to take into account the PGEs left in the waste by selecting the channel width within the PGE-bearing zones.
  Mf for PGE mining can only be determined by measurements in mining operations with detailed studies by the mine geologist. Nevertheless, the model is useful in carrying out sensitivity analyses to predict where economic advantages are evident.
- The 'Metallurgical Recovery Factor' (*Mr*) can be considered as a variable to represent the extent to which metallurgical improvements can arise from a higher grade and a more selective selection of material sent to the mill. It is impossible to model such relationships on any theoretical basis. But standard statistical methods, as are already used in the plants to establish correlation factors between recovery, grades and input variables, can provide the data to optimize the selection of the main variables for SBM.

In Table II, the cost model for the UG2 reef is shown using illustrative values that appear reasonable in trials at Impala Platinum Mine. However, great difficulty has been experienced in sampling the waste rock when thrown to the back of the stope so that to date no reliable figures can be quoted for the losses of PGEs .

Conclusions from the cost analysis for PGEs

In the case of platinum mining in the UG2 reef, the benefits are not as dramatic as in the case of gold, but nevertheless opportunities exist for increasing profitability and improving pay limits. The illustrative numbers used are, however, somewhat arbitrary and the advantages of SBM can swing from positive to negative by assuming greater losses of PGEs in the waste rock by virtue of the PGE content in the triplets which are discarded in the waste. In the base case it is assumed that a 10% loss occurs. If this loss is increased to say 25% then SBM results in decreased profits. If the loss can be reduced to 5% the advantages of SBM increase dramatically. This relationship between the choice of the channel height in relation to the losses in the waste is the critical parameter requiring investigation.

Acknowledgement

Thanks must be expressed to the consulting engineers of Impala Platinum for permission to publish the observations made at No. 6 shaft.

Nomenclature

Symbol

V	The grade value of a mineable tabular body expressed in terms of centimetre grams/ton (cg/t)
V1	The paylimit or cost break-even value
G	The gold content of material in grams/ton
Gc	The average gold content of the channel section mined, grams/ton
Gs	The average gold content of the whole stope g/ton *(V/Hs)*
Gm	The average gold content of the material milled g/ton
Gr	The grams of gold recovered and sold per ton of rock milled
Hs	The stope height in metres
Hc	The 'channel height', i.e. the height of the segment selectively blasted from the waste rock and sent to the surface for gold recovery
Hr	The reef height, i.e. the width of the conglomerate band containing gold, in metres
Fv	Void fraction in the broken and cast waste rock
Fa	Stope face advance per blast in metres
Sc	Average specific gravity of material in the channel segment
Ss	Average specific gravity of material in the stope
R	Selective Mining Ratio, the ratio of the mass of total rock blasted to the mass of rock sent to the mill
Cb	Total stoping and blasting costs in rand/ton of rock blasted
Cbx	Total stoping and blasting costs in alternative units, e.g. cost per metre of panel width per blast
Cs	Total stoping and blasting costs expressed in terms of rands per ton of rock milled
Cm	Material costs, i.e. aggregate of transport, hoisting, crushing, milling recovery and refining costs per ton of ore milled (R/t)
Cr	Total recovery costs, i.e. stoping and material costs (*Cs*+*Cr*) in rands/ton milled
Mf	The Mine Call Factor. The gold content of the

material sent for milling expressed as a fraction of the total gold content in the stope face as measured by the surveyors, sampling and assay. 1–*Mf* is the fraction of gold lost in underground operations.

Mr The Metallurgical Recovery Factor—the fraction of gold recovered and sold in relation to gold in the mill feed

Q Capital Cost Allocation. The cost allocation (above total operating costs) allocated by mine policies in relation to reserves expressed as rands/gram of gold recovered

X The increase in the gold production rate per month as a result of SBM expressed as a fraction of conventional rate

Y The decrease in pay limit in terms of total operating costs as a result of SBM expressed as a fraction of existing pay limit.

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