The influence of economics on the design of mine shaft systems

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SYNOPSIS

The purpose of this paper is to review some of the more important economic criteria that must be taken into account when planning the number, type, size, and distribution of the shafts, or shaft systems,required to exploit a block of payable ore.

The postulated philosophy governing the selection of design parameters is that of optimization of the financial return per rand of venture capital employed for the project as a whole. While this criterion may not necessarily satisfy the investment objective under all circumstances (as, for instance, where risk limitation is important), it does serve to place the relative economic advantages and disadvantages of specific design features, which may sometimes be in opposition, into correct perspective with respect to the overall economics of the project under the prevailing economic climate at the time of the investigation.

SINOPSIS

Die doel van hierdie verhandeling is om sommige van die meer belangrike ekonomiese maatstawwe te hersien wat in aanmerking geneem moet word by die beplanning van skagte, of skagsisteme, benodig vir die ondersoek van 'n area lonende erts.

Die veronderstellende filosofie wat die keuse van ontwerp parameters bepaal is een van optimalisering van die finansiële inkomste per Rand van die ondernemings kapitaal aangewend vir die projek as 'n geheel. Alhoewel hierdie maatstaf nie noodwendig die beleggings objektief onder alle omstandighede bevredig nie (soos, byvoorbeeld, waar risiko limitasies belangrik is), plaas dit wel die relatiewe ekonomiese voordele en nadele van spesifieke ontwerp eienaardighede, wat soms in mededinging is, in die regte perspektief wat betref die algehele ekonomie van die projek onder heersende ekonomiese klimaat by tye van die ondersoek.

INTRODUCTION

The early planning of the shafts, or system of shafts, that will ultimately serve a mine is a complex undertaking in which cognizance must be taken both of the technical and of the economic requirements of the project as a whole.

For the purpose of this paper it is presumed that the geology of the area to be mined has been thoroughly investigated, and that the average grade and tonnage comprising the potential ore reserve has been satisfactorily established. Furthermore, it is presumed that a preliminary economic exercise has been conducted to establish a first estimate of the most suitable size of the mine in terms of tonnes to be mined and hoisted per month. What remains, therefore, is to design the system of shafts that will best serve this specific size and conformation of mine. Its duties will include the hoisting of waste rock and ore; the handling of men and materials; the provision of downcast and upcast ventilation ways; access for services such as power, water, compressed air, dieseline, cement grout; and rising mains for mine drainage water. Therefore, it must be equipped with a system of adequately supported guides to cater for conveyances of a size that will provide the required hoisting capacity for rock, men, and materials, as well as accommodate normal mine equipment without undue dismantling. Finally, these services must be made available to all major operating levels in the mine by means of intermediate shaft stations, ore- and waste-pass systems, collecting levels and loading stations, spillage-handling arrangements, and intermediate pump stations, cable pockets, etc.

The importance of thoroughly exploring all possible alternative ways of providing this complex of services cannot be over emphasized. The need for technical excellence is self-evident; the economic stakes may be broadly summarized as follows:

(i) The construction cost of establishing an operational shaft system for a deep mine will probably absorb a major portion of the capital funds required to bring the mine to production. For example, Roodt and Upton quote the cost of shaft sinking and equipment as R20,4 million out of a total of R68,9 million required to bring Vaal Reefs South to the initial production stage. It is clear that an outlay of this magnitude cannot be contemplated many times in the life of a mine!

(ii) The high capital cost of shaft construction must be met before any compensating revenue can be derived from it.

(iii) In the case of a new mine, it is the sum total of all pre-production expenditure that must be met before a revenue flow is generated.

(iv) The shaft system, once constructed, cannot readily be modified either with regard to depth or capacity. It will therefore constitute a relatively rigid constraint on production volume for the remaining life of the mine.

BASIS OF SELECTION

If a fixed potential ore reserve is available for exploitation, it follows that the size of the mine in terms of tonnes mined per annum will be a function of the desired life span of the mine.

The capital expenditure required to bring the mine to full production is in turn a function of the size of the mine. A large initial capital expenditure will give big returns for a short period of time, whereas a

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smaller initial investment will produce lower returns for a longer period of time. The best solution will lie somewhere between the practical extremes. With such very large sums of money at stake, it follows that all plans and designs that may materially affect the future flow of expenditure and revenue should be subjected to a critical test of economic optimization.

Economic optimization infers a process of selection or rejection whereby a clearly defined objective may be attained with maximum economic benefit. It presupposes that:

(i) all parameters significantly affecting either expenditure or revenue can be clearly identified,

(ii) the 'cost function' or 'revenue function' of each parameter with respect to its unit of variability can be positively established, and

(iii) the effect of varying any parameter can be measured in terms of the objective sought.

It is essential to define the objective very clearly for the purpose of selecting design parameters in the planning stage of a new mine. It is proposed that the ultimate objective should be to exploit the available ore reserves in a manner that will give the best return on the venture capital, calculated in terms of the 'Present Value Ratio' or 'P.V. Ratio' as defined below:

$$\text{P.V. Ratio} = \frac{\text{Present value of cash flow generated}}{\text{Present value of capital funds invested,}}$$

where capital funds invested are the amounts that qualify as 'capital expenditure' for the purpose of calculating taxation;

cash flow generated is the balance remaining from year to year of the revenue from sales of mineral, less working costs, taxation, and lease consideration;

the rate of discount should ideally be the cost of capital of the investment funds concerned. In practice, the current bank overdraft rate will give a satisfactory first estimate if the financing arrangements for the venture have not been finalized at the time concerned.

The advent of the digital computer has made available a very powerful tool for the analysis of multi-variate problems of this nature because the input data can be varied at will to test the economic effects of any combination of physical design parameters comprising a mining programme.

The first step in the optimization exercise is to feed into the computer those input data which represent the best current estimate of the variables concerned. This will establish a measure of the reference starting point in terms of the P.V. Ratio. A typical computer print-out is submitted as Appendix 1. All figures in the lines designated *DATA* may be varied as required to test alternative designs and programmes.

Each condition it is desired to test for optimization should be analysed independently in order to establish its incremental cost behaviour; the input data can then be updated by adding or subtracting the marginal difference in costs to the reference data, and the effect of the new plan on the P.V. Ratio will determine whether it will be accepted or rejected. Successive iteration of this process will ultimately 'home in' on the optimum combination of design parameters. This approach has two major advantages to commend it.

(i) No single parameter is considered in isolation; the final indicator (P.V. Ratio) is a function of all the variables combined under the exact circumstances of taxation, lease, and rates of interest ruling for the mine as a whole. For example, a project may be uneconomic if the tax benefit from its proposed capital investment is deferred for, say, six years because the mine is already carrying a very high unredeemed capital expenditure. Six years later when the mine has fully redeemed its capital expenditure, the same project may become attractive.

(ii) Because the optimization procedure is a comparison between alternative solutions, it follows that only the marginal change in costs will increase or decrease the P.V. Ratio. The optimization exercise may, therefore, be carried out by the mere addition or deduction of the marginal change in costs to or from the reference data.

The last step must always be to make a completely new estimate of the overall cost for the derived best solution in order to give a final financial appraisal of the worth of the mine as an investment. It is presumed for the purpose of this paper that the mine offers an investment opportunity better than any alternative investment for which the funds may otherwise be used.

TIMING OF CASH FLOW

In a high-risk venture demanding a commensurately high discount rate to attract venture capital, the timing of the resultant cash flow is of prime importance. A particularly critical time is the period when large sums of venture capital have been committed to pre-production construction work. As soon as production commences, the situation is relieved of much of its anxiety and urgency—the flow of revenue is the first concrete evidence of the economic potential of the mine, and it provides the means of repaying high interest loans and/or of self-financing future capital expenditure, especially during the very attractive tax-free period. The best solution will vary with the circumstances. The practical examples that follow illustrate the importance of 'timing' in some specific situations.

Opening Up a New Deep-level Mine

Regulation 6.1.1 of the Mines and Works Act No. 27 of 1956, as amended, reads as follows:

'In connection with every mine there shall be provided shafts or outlets to surface such that except as permitted in regulation 6.3.1, any person employed underground in such a mine shall have available to him not less than two separate and independent shafts or outlets affording means of egress from underground to surface, provided that it shall not be necessary for

such shafts or outlets to be situated on the same mine.'

Humanitarian reasons apart, this regulation is largely responsible for the introduction of the twin-shaft system, in which two shafts are sunk concurrently at a distance apart of 80 to 100 metres so that no restraints will be placed on early production from stoping.

From considerations of ventilation efficiency and ease of access for men and materials, it would, in the long run, be more advantageous to locate the two shafts some distance apart. In such a case it would be necessary to excavate a connection between the shafts, under the severe constraints imposed by the mining regulations, before even commencing to open up access for stoping. The resultant delay in the flow of revenue makes such a programme economically unattractive.

In opening up a new mine where speed is of the essence, the twin-shaft system lends itself to a further benefit. One of the exceptions to regulation 6.1.1 is regulation 6.3.1 (b), which reads:

'One shaft or outlet to the surface shall be deemed to be sufficient means of egress from underground to surface for persons employed . . . in any shaft or winze from surface in the course of sinking . . .'

In the application of this regulation, shaft sinking is usually taken to include all work on ancillary excavations conducted from the shaft bottom during sinking. In deep shafts, work within the horizontal limits of the shaft pillar falls into this category, subject to any limitations that may be imposed by the Inspector of Mines.

Considerable benefit in terms of the overall time required to bring a mine into production can be derived by careful phasing of the construction programme for each of the twin shafts. A feature of a twin-shaft system is that normally one of the shafts will be virtually free of large intermediate stations and openings from the top to the bottom, i.e., it will serve for hoisting rock from the bottom level or as an upcast ventilation shaft. Sinking in the main shaft will be successively stopped for extending each intermediate station to the ore-pass position so that subsequent establishment of rock-handling facilities on all levels can be completed as soon as possible after the shaft is equipped; however, sinking in the other shaft is not subject to these delays and, on bottoming, an immediate start can be made from this shaft on establishing the shaft bottom service excavations for the main shaft, viz, main station connections, belt levels, silos, lower ore and waste passes, preliminary settlers and pump stations, etc. The main shaft may lag behind by as much as a year or more, so that by the time it bottoms, is equipped, and commissioned, the lower levels will be more or less established from the other shaft.

Under these circumstances high-speed development can then start on successive levels very soon after completion of the main shaft, as the ore- and waste-pass systems are pushed through from the bottom upwards, and concentrated stoping in the vicinity of the shaft system can follow not long afterwards.

Extension of an Existing Mining Area

Many of the benefits of a twin-shaft system fall away if it is possible to provide a connection from an existing shaft on the property or from an adjoining property. If the phasing of operations is carefully planned, it is possible to complete most of the underground service excavations while sinking from surface proceeds at a pace that allows it to be integrated into the daily work of the mine. In the ultimate, shafts have been excavated from both ends simultaneously, i.e., raising from the bottom upwards between levels while sinking proceeds from the surface downwards.

Under these circumstances a single shaft can normally meet the immediate requirements of hoisting. The need to provide, in addition, for an upcast airway has led to the development of the double-duty shaft having the upcast/downcast sections separated by a longitudinal concrete brattice wall. The brattice wall is built up from the bottom of the shaft prior to equipping, using pre-cast, pre-stressed concrete panels capable of withstanding an air pressure difference of the order of 75 cm of water gauge (7,35 kPa).

A large divided shaft of this nature is normally less expensive than a twin-shaft system comprising two smaller shafts. Some very large shafts of this type have been constructed on the South African goldfields, notably the President Steyn No. 4 Shaft and the Elsburg Main Shaft.

Outcrop workings

Where the orebody is situated on or near surface and does not lend itself to opencast working, or where it dips from surface to depth, the situation is again overshadowed by the economic advantages of gaining early revenue for minimum initial capital outlay.

Under these circumstances it is economically advantageous to draw the early mill tonnage from areas close to the surface through a number of small inclined or vertical shafts or winzes (Impala Platinum Mine[2] and Winkelhaak Mine[3]).

Inclined shafts or winzes equipped with conveyor belts are standard practice in coal mines, where they handle large tonnages at extremely low capital and working costs. To date, this type of shaft has not seen much application on metalliferous mines, but technical improvements in the transportation of hard rock by conveyor belts indicates a big potential for wider application in future.

In the long term, the exploitation of the deep levels of such a mine will demand a more centralized arrangement of shaft facilities as operations extend beyond the range of the smaller shafts. Such extensions, however, would be fully planned and executed as a phased take-over on the lines discussed previously.

CONSTRUCTION COST/SIZE RELATIONSHIP

One of the major snags encountered in an economic analysis of this type is the difficulty of finding accurate and fully representative cost data from which to construct a model of the 'cost function' of the variable parameters.

Many efforts have been made in the past to determine the relationship between the diameter and the cost per linear foot of sinking a

circular shaft. Costs have been derived from historic data, which have generally suffered from one or more of the following defects:

(i) inadequate and/or unreliable cost records, and

(ii) inconsistent physical conditions, viz, ground conditions, water and gas accumulations, crew expertise, ancillary excavations, etc.

The additional costs resulting from variables of this nature are usually of such a magnitude that they completely blanket the true basic cost of shaft sinking. As a result, attempts to establish the cost/size relationship for shaft sinking have been imprecise and open to contention[4, 5].

However, there is an alternative to the use of historic data. In recent years, mining companies have frequently used the services of specialist shaft-sinking contractors for shaft construction, often from the turning of the first sod to the ultimate commissioning of the shaft as a going concern. Because the contracts have been awarded against a firm bid, the contractors have given considerable thought to the techniques of pre-estimating costs from first principles to meet the technical specifications of the proposed shaft. As a result, a considerable bank of basic activity and commodity cost values has been accumulated.

The derivation of cost/size relationships from basic estimates of this type has the great advantage that hypothetical physical conditions can be standardized to give a true comparison between different types and/or sizes of shaft.

If we bear in mind that the object of this exercise is to find the incremental cost of an incremental change in shaft diameter, then it follows that only those costs which will vary with diameter are of significance. Therefore, miscellaneous fixed overheads, which frequently confuse the issue, can be entirely neglected in such an analysis.

In order to illustrate this approach, a series of basic estimates has been made of the major direct costs of sinking a range of ten circular shafts, increasing in diameter in steps of 2 feet (0,61 metres) from 18 feet (5,49 metres) to 36 feet (10,98 metres). It is assumed that all the shafts considered will be sunk to a depth of 6500 feet (approximately 2000 metres) on a high-speed multi-shift basis, using a multi-deck stage and mechanical cactus grab for cleaning. This method of shaft sinking is not practicable in shafts of less than 18 feet in diameter — in fact, the constricted working space in an 18-foot shaft causes many in-

TABLE I
Estimated cost per linear foot of shaft sunk (Rands)

Shaft diameter (feet)	18	20	22	24	26	28	30	32	34	36
Consumables	71	82	93	105	117	132	145	160	177	193
European labour	29	29	31	33	38	46	50	56	61	67
Bantu labour	23	26	29	31	36	44	52	61	72	83
Sinking equipment	30	31	35	40	45	59	66	74	82	89
	153	168	188	209	236	281	313	351	392	432

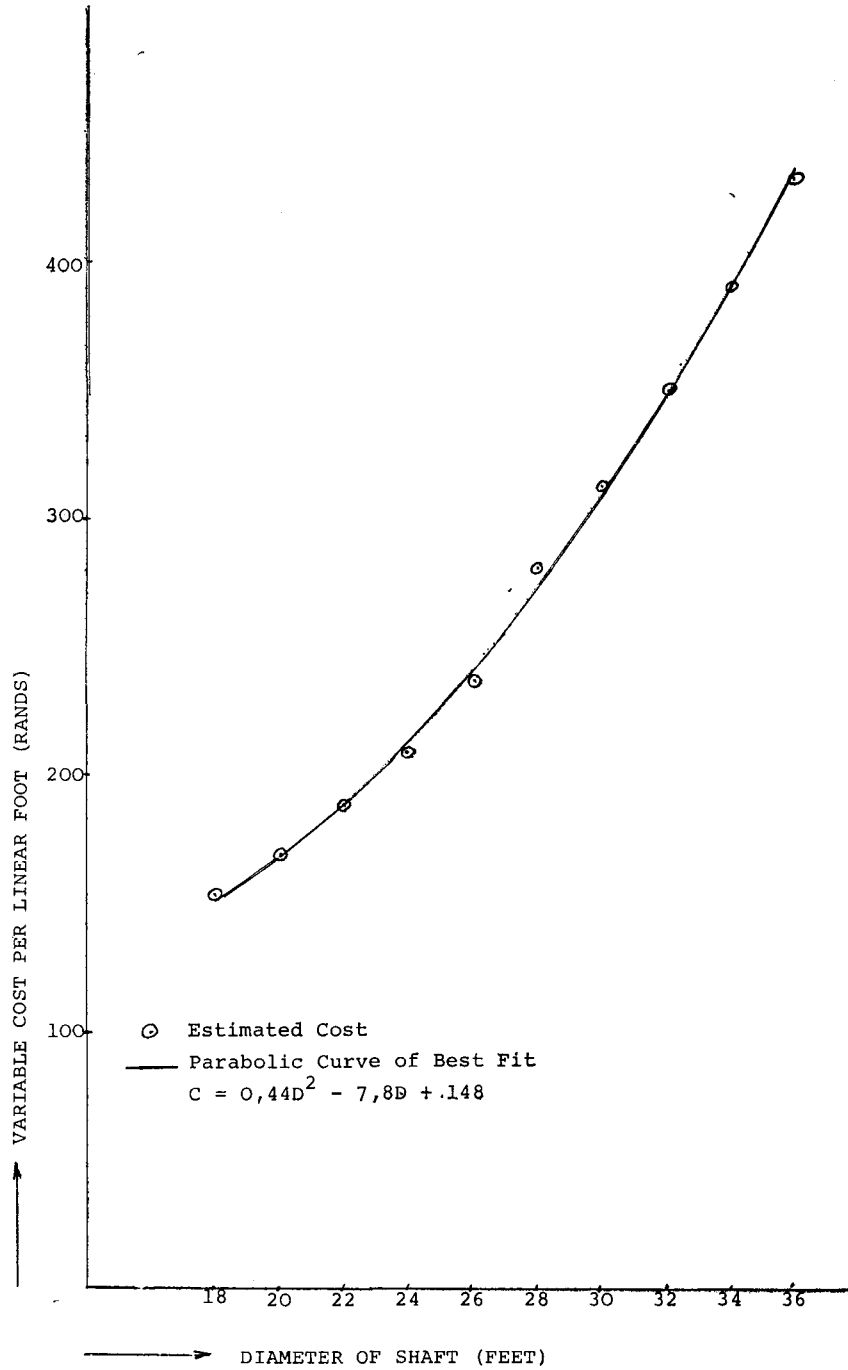
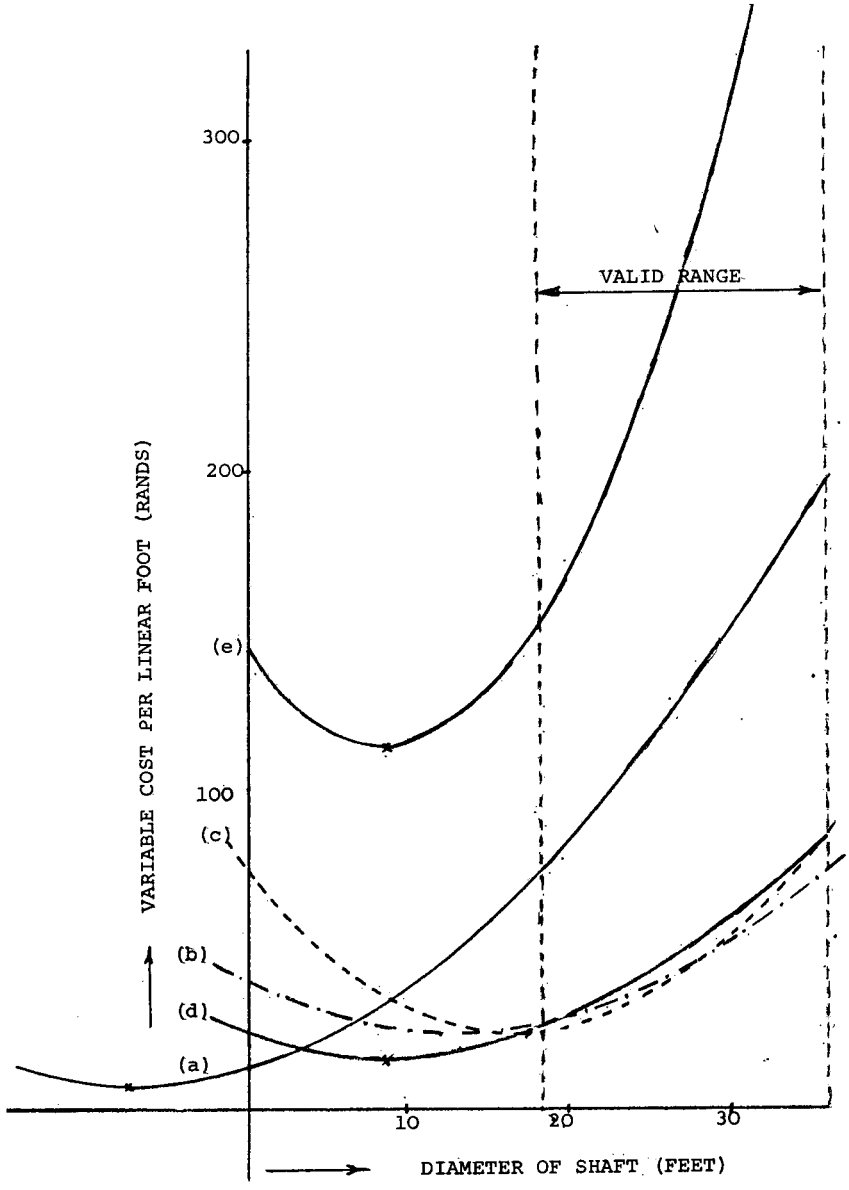


FIG. 1



(a) Consumables : $C = 0{,}098D^2 + 1{,}50D + 12{,}7$
(b) European Labour : $C = 0{,}081D^2 - 2{,}07D + 38{,}4$
(c) Bantu Labour : $C = 0{,}164D^2 - 5{,}54D + 70{,}3$
(d) Sinking Equipment : $C = 0{,}092D^2 - 1{,}40D + 22{,}9$
(e) Total : $C = 0{,}439D^2 - 7{,}81D + 148$

FIG. 2

efficiencies and necessitates expensive modifications in the design of mechanical equipment. Consequently, no conclusions should be extended beyond the discontinuity at 18 feet diameter.

Estimates of the direct cost of European labour, Bantu labour, consumables, and sinking equipment are submitted as Appendices 2, 3, 4, and 5 respectively. The cost per linear foot of shaft sinking is summarized under the same headings in Table I.

A first examination of these results shows that the cost per linear foot increases at an ever-increasing rate as the shaft diameter increases. The cost function is therefore not linear. Using the method of curve fitting by least squares, a second-order polynomial (quadratic) function was found to fit the estimated data very satisfactorily. In Fig. 1 the estimated points may be compared with the smooth curve of best fit defined by the expression,

$$C=0{,}44D^2-7{,}8D+148$$

Note: A minor discontinuity exists between shaft diameters of 26 feet and 28 feet. This represents the change from 2-kibble to 4-kibble hoisting; it is typical of a stepwise increase in units of mechanical equipment.

Parabolic curves of best fit have been computed in a similar manner for each of the major headings in Table I. They are shown in Fig. 2 extended for analysis purposes below the practical minimum diameter of 18 feet. Consider the practical logic of each of these curves.

(a) Consumables ($C=0{,}098D^2+1{,}50D+12{,}7$). Examination of the cost items included under this heading (Appendix 4) will show that some of them vary with shaft diameter as a linear function (concrete for shaft lining), and some vary as a square function (proportional to tonnage). These relationships suggest intuitively that the overall cost will behave as a quadratic function.

Note: The curve does not intersect the origin of the axes because breaking becomes more and more constrained as the shaft diameter decreases. This is borne out by the rule-of-thumb formula often used as a first estimate of the number of holes in a round, viz, $N=0{,}25A+20$, where N=number of holes and A=excavated area of shaft.

(b) European labour ($C=0{,}081D^2-2{,}07D+38{,}4$). In mechanized shaft sinking, the European labour strength is made up to meet a complex mixture of supervision and direct-operating functions. As shafts become smaller, an irreducible minimum strength is required to perform the various duties. The flattening of the curve towards the lower end of the range has the effect of locating the theoretical minimum point of the parabola in the positive quadrant; this, in turn, explains the negative coefficient of the linear term in the cost function.

(c) Bantu labour ($C=0{,}164D^2-5{,}54D+70{,}3$). Except for the blast-hole drilling labour, which is allocated in proportion to shaft area, the remaining Bantu strength is required to perform specific functional duties in the shaft. The manpower required for these duties again approaches a minimum towards the lower end of the range of shafts considered.

(d) Sinking equipment ($C=0{,}092D^2-1{,}40D+22{,}9$). Minimum

TABLE II
Increase in cost per foot for a 1-foot increase in diameter

Shaft diameter (feet)	18	20	22	24	26	28	30	32	34	36
$\frac{dC}{dD}=0{,}88D-7{,}8$ (Rands)	8,0	9,8	11,6	13,3	15,1	16,8	18,6	20,4	22,1	23,9

equipment requirements and the relatively higher cost of smaller equipment both tend to cause a flattening of the curve in the lower range. However, the flattening is less pronounced than in the case of labour costs.

If one is considering an incremental enlargement in the planned diameter of a shaft, it is important to know the additional costs involved. The rate of change at any point on the curve in Fig. 1 is given by the first derivative of the function of the curve

$$\frac{dC}{dD} = 0{,}88D - 7{,}8$$

The values in Table II assume that the curve approximates a straight line for an increment of 1 foot in diameter.

It is interesting to note the relatively small increase in cost for 1 foot increase in diameter when compared with the overall cost of bringing a new mine to production. If there is any doubt concerning the optimum size of a shaft in the design stage, it is often false economy in the long run to attempt to reduce capital expenditure by shaving tolerances. The additional marginal cost of allowing for a 'comfortable' design may be a small premium to pay for making certain that the shaft will not prove to be an expensive bottleneck for the life of the mine.

If there is a real possibility that an increase in production may be required at some time in the future, it may pay to excavate the shaft to cater for peak production but to delay installing additional hoisting facilities until such time as they are required. The additional immediate cost of shaft construction plus the deferred cost of hoisting facilities must be compared with the deferred cost of a complete additional shaft entity to make up the equivalent tonnage. Steed[6] comments further on the economics of this problem.

Note: The cost estimates used in this analysis do not necessarily correspond to today's ruling price and wage levels; they are submitted purely as an example of the principle involved. No attempt has been made to include estimates for shaft equipping nor for permanent equipment and surface installations, as these vary so much in design that a general comparison would be specious. However, the practical problem generally involves consideration of increasing or decreasing the diameter of a proposed shaft by the order of one or two feet. Under these circumstances, Table II will give a first approximation of the associated incremental costs.

VENTILATION COST/SHAFT SIZE RELATIONSHIP

Hoisting capacity and the quantity of circulating air required for mine ventilation purposes are the two major requirements that dictate the cross-sectional area of a shaft. In deep, hot mines it is normally the ventilation requirement that determines the minimum shaft area permissible[6]. The economics of this situation is discussed in the following sections.

Determination of ventilation air quantity

A firm knowledge of the air quantity required for ventilation purposes is a primary prerequisite before the planning and design of a shaft system can be commenced. This subject is not pertinent to this paper. Suffice it to say that it depends on the depth of the orebody, the virgin-rock temperature gradient, the area to be served by the shaft, the mining method, and the rate of mining in terms of tonnes per month. Grave and Stroh[7] give an excellent exposition of the extent of the problem and of the current techniques for the prediction of the future requirements for any set of conditions. In particular, attention is drawn to the following extract from this paper:

'Therefore the establishment of whole shaft systems can be avoided and removed from the estimates by making the existing shaft system, which would otherwise become inadequate, serve, with the aid of refrigeration, areas far beyond those originally intended.'

Lambrechts[8] and Steed[6] further discuss the economics of replacing a portion of the fresh air requirement of a hot, deep mine by refrigeration, and Van Schalkwyk and Viljoen[9] describe a practical situation in which the ventilating air requirement for a section of a mine was reduced to less than half by the introduction of appropriate refrigeration capacity.

In the case of planning a new shaft, the same principles apply, although the situation may differ materially. The balance between refrigeration and air quantity is discussed in more detail in the following sections.

Determination of shaft size

An incremental decrease in the planned shaft cross-sectional area will affect the cash flow for the life of the mine in the following ways:

— reduction in shaft-sinking costs over the period of sinking,
— increase in capital cost of circulating fans,
— increase in capital cost of refrigeration,
— increase in annual fan operating cost, and
— increase in refrigeration-plant operating cost.

The economics of making the change can be tested, as explained previously, by its resultant effect on the P.V. Ratio for the mine as a whole. The following pertinent points emerge.

(i) The capital cost of shaft sinking is heavily weighted as it occurs in the early pre-production stage. Although some compensating tax benefit will be derived from an increase in capital expenditure, the effect of incremental capital redemption may only be felt in possibly six or seven years' time, and it will therefore be materially discounted at today's rates of interest.

(ii) The increase in the capital cost of fans is relatively small and it is deferred until shaft sinking is complete. In the early production stage, fresh-air demands will be low so that full power may be deferred even further.

(iii) The provision of refrigeration may be deferred until the designed air flow can no longer

maintain acceptable underground working conditions. If at this time the mine has fully redeemed all capital expenditure, then an immediate tax advantage will accrue to offset the capital cost of installation.

(iv) The bulk of the increase in working costs over the life of the mine is heavily discounted, especially at today's high rates of interest.

On balance, it appears that very material long-term cost reductions are required to justify any immediate expenditure on increasing the shaft size over and above that required for hoisting purposes. It has already become impracticable to expect to maintain acceptable environmental conditions in ultra-deep, hot mines by circulating fresh air alone[7]. A certain base load of air is necessary to supply oxygen for maintaining life and to dispel air-borne contaminants from the mine; this volume of air may serve adequately in the early life of the mine during the build-up period to full production, but, as mining progresses, any subsequent deficit in cooling power must be supplied by refrigeration as and when it is required. The optimum shaft size will be a function of the balance between fresh-air volume and refrigeration, and it can be determined only by an overall economic optimization exercise as described earlier in this report.

*Design to reduce K**

The cost of power to overcome the internal resistance to the flow of air through a shaft will live with the mine for its full life span. Consequently, all those design parameters that affect the value of K should be very carefully considered when designing the internal conformation of a shaft. Generally, the additional expense incurred in the construction stage is negligible compared with the benefits that will accrue for the rest of the life of the mine. Current literature covers this subject in great detail. Some of the more important parameters are listed below:

—Arrangement of shaft steelwork
—Spacing of buntons
—Streamlining of bunton cross-section
—Alternative methods of guiding conveyances: stub buntons, rope guides
—Smooth concrete lining
—Widening of shaft at station elevations
—Bank bypass (independent ventilation inlet)
—Long narrow conveyances
—Scientifically designed ventilation inlets to, or outlets from, shaft (splitter blades)
—Fairings to streamline conveyances
—Evasée design.

*K is the air resistance co-efficient in Atkinson's formula $P=\frac{KSV^2}{A}$

CENTRALIZED VERSUS DECENTRALIZED HOISTING

How many shafts should be provided to serve the needs of a mining property over the span of its productive life? The economics of this problem involves a comparison of centralized hoisting (with underground transport of rock, men, materials, and services,) with multiple-shaft hoisting combined with surface transport facilities. While a full discussion of this subject is not the province of this paper, it is necessary to identify the problem areas in broad principle insofar as they affect the selection of shaft facilities for any particular mining proposition.

Balance of size of shaft and size of mine

The optimum life span of a proposed mine is dictated by economic factors that take into account the predicted net cash flow and the risks involved. At discount rates of 15 per cent and more, the present value of income deferred more than, say, 20 years is insignificant (the P.V. of R1 deferred 20 years at 15 per cent discount rate is 6,1 cents). Hence for planning purposes it is proposed to consider, as a first estimate, a tonnage call that will exhaust the potential ore reserves within a period of 20 years.

For a deep-level mine it is economically desirable that a shaft complex should be designed to serve an area that will have a life span equivalent to that of the mine as a whole. A small shaft will thus serve a comparatively small area, necessitating a multiple-shaft system to meet the gross monthly tonnage call of the mine as a whole; a very large shaft may be capable of meeting the overall tonnage call, but it will then be required to serve the entire area of the mine for its life span.

In both these cases the ratio of the ore-reserve tonnage served by the shaft(s) to the capacity of the shaft(s) is identical. What is not the same is the cost of the shaft facility per ton of ore reserve served.

The cost of sinking one high-capacity shaft is significantly less than the cost of sinking two or more smaller shafts with equivalent combined capacity. In addition, one central shaft will avoid duplication of ancillary facilities such as shaft stations, rock-pass systems, surface structures, road and rail links, etc. In contrast to this saving in capital cost per unit of ore reserves for a central shaft system, the phasing of two or more smaller shafts to effect a stepwise build-up to full production has definite advantages such as reducing the initial venture capitalization required, and allowing for a more controlled growth of manpower and management organization. The final decision will depend on the testing of many combinations to arrive at an acceptable compromise between economic optimization, risk propensity, and availability of resources.

There has been a progressive increase in the size and hoisting capacity of deep vertical shafts sunk on the South African goldfields in recent years. This trend has not been the result of any particular technological innovation; it has resulted from a logical extension in the application of previously known and proved techniques as a direct response to economic stimulus. Today it is possible to construct and operate a self-contained shaft system capable of providing all essential services for a gold mine producing 400 000 tonnes of rock per month. In other words, one vertical shaft-system is capable of sustaining a production rate that will exhaust some 80 million tonnes of ore in 20 years. Few gold mines in South Africa exceed this potential.

Underground mine layout

A large, high-production shaft can operate effectively only if it is adequately served by a sophisticated

APPENDIX I

D.C.F. for a new Gold Mine

INITIAL LOAN BALANCE (R1 000)	0.	TAX FORMULA	y=60.0−480.0/X
INITIAL FUNDS AVAILABLE (R1 000)	35000.	LEASE FORMULA	Y=15.0−120.0/X
ISSUED SHARES (1 000)	35000.	LOAN LEVY (GOLD)	5.0%
SHARE PURCHASE PRICE (R)	1.000	TAX SURCHARGE (GOLD)	5.0%
PURCHASE PRICE (R1 000)	2000.	CONSOLIDATED COMPANY TAX RATE	41%
UNREDEEMED CAPEX B/F (R1 000)	2500.	INTEREST RATE ON LOAN CAPITAL	10.0%

CALCULATION REFERENCE	ITEM	1973	1974	1975	1976	1977	1978	1979	198
	Production and Revenue								
(1) DATA	Tonnes milled (1000)	0.	0.	0.	0.	1200.	1800.	2400.	2400
(2) DATA	Mill head grade (Au—g/t)	0.00	0.00	0.00	0.00	16.00	17.00	17.00	17.0
(3) DATA	(U—Kg/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.0
(4) DATA	Sales realisation (Au—R/gr)	1.260	1.310	1.362	1.416	1.473	1.531	1.592	1.65
(5) DATA	(U—R/Kg)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
(6) 1*2*4*.97	Revenue (Au)	0.	0.	0.	0.	27433.	45443.	63005.	65538
(7) 1*3*5*.85	(U)	0.	0.	0.	0.	0.	0.	0.	0
(8) DATA	Other mining revenue	0.	0.	5.	5.	5.	10.	10.	10
(9) 6+7+8	Gross mining revenue	0.	0.	5.	5.	27438.	45453.	63015.	65548
	Cost and Profit								
(10) DATA	Working cost per tonne milled (R)	10.00	10.50	11.02	11.76	12.15	12.76	13.40	14.0
(11) 1 * 10	Gross working cost	0.	0.	0.	0.	14580.	22968.	32160.	33768
(12) DATA	Lump sum adjustment	0.	0.	0.	0.	0.	0.	0.	0
(13) 58*rate/100	Post prod int paid on loan	0.	0.	0.	0.	2541.	3688.	2477.	69
(14) 9—(11+12+13)	Gross mining income	0.	0.	5.	5.	10317.	18797.	28378.	31711
	Lease Consideration								
(15) 24prev	Unredeemed capex B/F	2500.	11708.	28251.	37467.	64825.	76300.	64198.	40120
(16) DATA	Current capex	9208.	16468.	9101.	27157.	21793.	6695.	4300.	2000
(17)	Pre-prod int paid on loan	0.	75.	120.	205.	0.	0.	0.	0
(18) 15+16+17	Capex ranking for redemption	11708.	28251.	37472.	64830.	86618.	82995.	68498.	42120
(19) 27prev	Capital allowance B/F	0.	403.	1583.	3623.	6835.	11734.	17200.	22203
(20) (15+19)*.06	Capital allowance on 15+19	150.	727.	1790.	2465.	4300.	5282.	4884.	3739
(21) 16*.0275	Capital allowance on 16	253.	453.	250.	747.	599.	184.	118.	55
(22) 19+20+21	Capital allowance for redemption	403.	1583.	3623.	6835.	11734.	17200.	22203.	25997
(23) Lesser of 14 or 18	Capital redeemed	0.	0.	5.	5.	10317.	18797.	28378.	31711
(24) 18—23	Unredeemed capex C/F	11708.	28251.	37467.	64825.	76300.	64198.	40120.	10490
(25) 14—23	Balance of income after 23	0.	0.	0.	0.	0.	0.	0.	0
(26) Lesser of 25 or 22	Capital allowance redeemed	0.	0.	0.	0.	0.	0.	0.	0
(27) 22—26	Unredeemed capital allowance C/F	403.	1583.	3623.	6835.	11734.	17200.	22203.	25997
(28) 25—26 (Y)	Balance of income after 26	0.	0.	0.	0.	0.	0.	0.	0
(29) Y=al—(bl*9/(25—17)*	Rate of lease payment (%)	0.	0.	0.	0.	0.	0.	0.	0
(30) 28 * 29/100	Amount of lease payment	0.	0.	0.	0.	0.	0.	0.	0
(31) 30 * 1.0125	Lease plus 1.25% surcharge	0.	0.	0.	0.	0.	0.	0.	0
	Taxation								
(32) 37prev	Unredeemed capex B/F	2500.	12246.	30376.	42361.	74111.	92315.	87844.	70951
(33) 32 * .08	Capital allowance on 32	200.	980.	2430.	3389.	5929.	7385.	7027.	5676
(34) (16+17)*.0367	Capital allowance on 16+17	338.	607.	338.	1004.	800.	246.	158.	73
(35) 32+33+34+16+17	Capex ranking for redemption	12246.	30376.	42366.	74116.	102632.	106641.	99329.	78700
(36) Lesser of 14 or 35	Capex redeemed	0.	0.	5.	5.	10317.	18797.	28378.	31711
(37) 35—36	Unredeemed capex C/F	12246.	30376.	42361.	74111.	92315.	87844.	70951.	46989
(38) 14—31—36	Taxable profit	0.	0.	0.	0.	0.	0.	0.	0
(39) at—(bt*9/(38*100)	Rate of taxation (%)	0.	0.	0.	0.	0.	0.	0.	0
(40) 38*39/100	Formula tax	0.	0.	0.	0.	0.	0.	0.	0
(41) 40*tax surcharge	Tax surcharge	0.	0.	0.	0.	0.	0.	0.	0
(42) 53*int rate	Interest on reserve funds	2700.	1200.	400.	0.	0.	0.	0.	0
(43) DATA	Other non-mining income	0.	0.	0.	5.	5.	7.	7.	7
(44) (42+43)*cotax/100	Tax on non-mining income	243.	108.	36.	.	.	1.	1.	1
(45) (40+41)*aulev/100	Loan levy (gold)	0.	0.	0.	0.	0.	0.	0.	0
(46)	Loan levy repayment	0.	0.	0.	0.	0.	0.	0.	0
(47) 40+41+44+45—46	Net payment to state	243.	108.	36.	.	.	1.	1.	1
	Source and Application of Funds								
(48) 14—31—47	Net funds generated	—243.	—108.	—31.	5.	10317.	18797.	28377.	31711
(49) 53prev	Reserves B/F	35000.	27000.	12000.	4000.	0.	0.	0.	0
(50) DATA	Capital recoupment (after tax)	0.	0.	0.	0.	0.	0.	0.	0
(51) 48+49+50+42+43	Gross funds available for year	37457.	28092.	12369.	4010.	10322.	18804.	28384.	31718
(52) DATA	Acquisition cost instalments	2000.	0.	0.	0.	0.	0.	0.	0
(53) DATA	To reserves	27000.	12000.	4000.	0.	0.	0.	0.	0
(54) 52+53+16+17	Gross funds required for year	38208.	28543.	13221.	27362.	21793.	6695.	4300.	2000
(55)	Loan to be raised	751.	451.	852.	23353.	11471.	0.	0.	0
(56)	Loan repayment	0.	0.	0.	0.	0.	12109.	24084.	685
(57)	Dividend payment	0.	0.	0.	0.	0.	0.	0.	29032
(58)	Accumulated loan C/F	751.	1202.	2054.	25407.	36878.	24770.	685.	0
	N.P.V. and Pay Back Period								
(59) 51—49	Net cash inflow	2457.	1092.	369.	10.	10322.	18804.	28384.	31718
(60)	Disc factor for year N	.9091	.8264	.7513	.6830	.6209	.5645	.5132	.466
(61) 60 *59	P.V. of cash inflow	2234.	902.	277.	7.	6409.	10614.	14566.	14797
(62) 54—53	Net cash outflow	11208.	16543.	9221.	27362.	21793.	6695.	4300.	2000
(63)	Disc factor for year N—1	1.0000	.9091	.8264	.7513	.6830	.6209	.5645	.513
(64) 62 * 63	P.V. of cash outflow	11208.	15039.	7621.	20558.	14885.	4157.	2427.	1026
(65)	Net present value of project	38054.		Pay back period		9.0 years			
	Calculation of yield								
(66)	P.V. factor at 40% disc rate	.7143	.5102	.3644	.2603	.1859	.1328	.0949	.067
(67)	P.V. factor at 30% disc rate	.7692	.5917	.4552	.3501	.2693	.2072	.1594	.122
(68)	P.V. factor at 20% disc rate	.8333	.6944	.5787	.4823	.4019	.3349	.2791	.232
(69)	P.V. factor at 10% disc rate	.9091	.8264	.7513	.6830	.6209	.5645	.5132	.466
(70)	P.V. factor at 0% disc rate	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.000
(71)	P.V. of one share at 40% disc rate	.159							
(72)	P.V. of one share at 30% disc rate	.334							
(73)	P.V. of one share at 20% disc rate	.793							
(74)	P.V. of one share at 10% disc rate	2.270							
(75)	P.V. of one share at 0% disc rate	8.842		Yield on share purchase price 18.6					

1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997
2400.	2400.	2400.	2400.	2400.	2400.	2400.	2400.	2400.	2400.	2400.	2400.	2400.	1800.	1800.	1200.	1200.
17.00	17.00	16.00	16.00	16.00	16 00	16.00	16.00	16.00	16.00	16 00	16 00	17.00	17.00	17.00	17.00	17.00
0 00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0 00	0 00	0 00	0 00	0 00	0 00	0.00	0.00	0 00
1.722	1.790	1.861	1.935	2 012	2.092	2.175	2.262	2 446	2.446	2.544	2.646	2.751	2.861	2.975	3.094	3.218
0 000	0 000	0.000	0 000	0.000	0.000	0.000	0.000	0 000	0.000	0.000	0.000	0.000	0.000	0.000	0 000	0.000
68150.	70841.	69318.	72075.	74943.	77923.	81014.	84225.	87607.	91109.	94759.	98558.	108874.	84920.	88304.	61224.	63678.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.	10.
68160.	70851.	69328.	72085.	74953.	77933.	81024.	84265.	87617.	91119.	94769.	98568.	108884.	84930.	88314.	61234.	63688.
14.77	15 51	16.28	17.10	17.96	18.86	19.80	20 79	21.83	22.93	24.07	25.27	26.53	27.86	29.25	30.71	32.25
35448.	37224.	39072.	41040.	43104.	45264.	47520.	49896.	52392.	55032.	57768.	60648.	63672.	50148.	52650.	36852.	38700.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
32712.	33627.	30256.	31045.	31849.	32669.	33504.	34369.	35225.	36087.	37001.	37920.	45212.	34782.	35664.	24382.	24988.
10409.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
12409.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.
25997.	7933.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
2184.	476.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.
28236.	8464.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.
12409.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
20303.	31627.	28256.	29045.	29849.	30669.	31504.	32369.	33225.	34087.	35001.	35920.	43212.	32782.	33664.	22382.	22988.
20303.	8464.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.	55.
7933.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
0.	23163.	28201.	28990.	29794.	30614.	31449.	32314.	33170.	34032.	34946.	35865.	43157.	32727.	33609.	22327.	22933.
0.	12.	12.	12.	11.	11.	11.	11.	11.	11.	11.	11.	11.	11.	11.	11.	11.
0.	2780.	3384.	3479.	3277.	3368.	3459.	3555.	3649.	3743.	3844.	3945.	4747.	3600.	3697.	2456.	2523.
0.	2814.	3426.	3522.	3318.	3410.	3503.	3599.	3694.	3790.	3892.	3994.	4807.	3645.	3743.	2487.	2554.
46989.	20110.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
3759.	1609.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.	73.
52821.	23792.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.
32712.	23792.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.	2073.
20110.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
0.	7021.	24757.	25449.	26457.	27186.	27928.	28697.	29457.	30223.	31035.	31852.	38332.	29064.	29847.	19822.	20360.
0.	12.	47.	46.	46.	46.	46.	46.	46.	46.	45.	45.	46.	46.	46.	45.	45.
0.	812.	11526.	11809.	12277.	12571.	12868.	13173.	13469.	13760.	14072.	14380.	17773.	13362.	13669.	8954.	9159.
0.	41.	576.	590.	614.	629.	643.	659.	673.	688.	704.	719.	889.	668.	683.	448.	458.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.	7.
1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.	1.
0.	43.	605.	620.	645.	660.	676.	692.	707.	722.	739.	755.	933.	701.	718.	470.	481.
0.	0.	0.	0.	0.	0.	53.	756.	775.	806.	825.	844.	864.	884.	903.	923.	944.
1.	896.	12708.	13021.	13536.	13860.	14134.	13768.	14075.	14365.	14690.	15010.	18730.	13848.	14168.	8949.	9155.
32711.	29917.	14122.	14502.	14995.	15399.	15868.	17002.	17456.	17931.	18418.	18915.	21675.	17289.	17753.	12947.	13279.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
100.	30.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	4000.
32818.	29954.	14129.	14509.	15002.	15406.	15875.	17009.	17463.	17938.	18425.	18922.	21682.	17296.	17760.	12954.	17286.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
30818.	27954.	12129.	12509.	13002.	13406.	13875.	15009.	15463.	15938.	16425.	16922.	19682.	15296.	15760.	10954.	15286.
0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.	0.
32818.	29954.	14129.	14509.	15002.	15406.	15875.	17009.	17463.	17938.	18425.	18922.	21682.	17296.	17760.	12954.	17286.
.4241	.3855	.3505	.3186	.2897	.2633	.2394	.2176	.1978	.1799	.1635	.1486	.1351	.1228	.1117	.1015	.0923
13918.	11549.	4952.	4623.	4346.	4057.	3800.	3702.	3455.	3226.	3013.	2813.	2930.	2125.	1983.	1315.	1595.
2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.	2000.
.4665	.4241	.3855	.3505	.3186	.2897	.2633	.2394	.2176	.1978	.1799	.1635	.1486	.1351	.1228	.1117	.1015
933.	848.	771.	701.	637.	579.	527.	479.	435.	396.	360.	327.	297.	270.	246.	223.	203.
P.V. ratio at 10.0%=1.4469																
.0484	.0346	.0247	.0176	.0126	.0090	.0064	.0046	.0033	.0023	.0017	.0012	.0009	.0006	.0004	.0003	.0002
.0943	.0725	.0558	.0429	.0330	.0254	.0195	.0150	.0116	.0089	.0068	.0053	.0040	.0031	.0024	.0018	.0014
.1938	.1615	.1346	.1122	.0935	.0779	.0649	.0541	.0451	.0376	.0313	.0261	.0217	.0181	.0151	.0126	.0105
.4241	.3855	.3505	.3186	.2897	.2633	.2394	.2176	.1978	.1799	.1635	.1486	.1351	.1228	.1117	.1015	.0923
1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000	1.0000

system of underground haulages and shaft excavations, and by a suitable surface layout designed to handle the concentration of rock, men, and material involved. The entire system, from stope to mill, must be considered as an integrated whole, designed primarily as a means of moving traffic efficiently from dispersed source to concentrated destination (rock), or from concentrated source to dispersed destination (materials). In all cases it is in the shaft that maximum concentration occurs. Although the shaft may be designed to handle this concentration, it is necessary to consider the practical problems associated with the communications network extending from the shaft.

General underground layout

In principle, one can visualize a number of major footwall haulages extending from the shaft to serve self-contained production-manager's sections, each having a centralized rock- and materials-handling system connecting with the appropriate footwall haulage at main interchange stations; new sections would be developed on a pre-planned basis to replace exhausted areas as and where required so that development expenditure would be spread over the life of the mine.

Rock transport

The main footwall haulages should preferably be perfectly straight, evenly graded towards the shaft, and of a size that will accommodate large equipment travelling at high speed. With matched loading and discharging arrangements, a single haulage could easily handle 100 000 tonnes or more per month over a distance of, say, 5 kilometres. Alternatively, the transportation of large tonnages between two fixed points is a situation ideally suited to a major conveyor-belt transportation system.

Materials transport

Considerable scope exists for the application of advanced bulk-materials handling methods using specially designed containers and rolling stock that can be integrated into the rock-transport system.

Transport of men

The most serious problem is that of absorbing productive shift time in travelling. Some relaxation of the existing regulations governing the speed of traction of man-haulages underground would be called for as the haulages extend outwards. Under the ideal conditions envisaged, why should the speed in a mine be limited more than that on an underground railway system in a city? Mere distance from the shaft would be no problem in such circumstances.

Ventilation

In deep, hot mines the provision of acceptable ventilation conditions so far from the downcast shaft will demand adequate fresh-air and return airways, coupled with suitable refrigeration and chilled-water cooling arrangements[6].

Technically, there appears to be no reason why a very large mine cannot be served by one central shaft system; economically, the bias appears also to be in this direction. In addition to the direct saving in shaft sinking and ancillary construction costs already described, it is probable that overall transportation costs will be significantly reduced. The cost of highly concentrated underground transport on the one hand must be compared on the other hand with more dispersed underground transport, sometimes in the wrong direction, combined with surface transport under conditions not always ideal, especially insofar as surface topography is concerned.

The final selection of the combination of size and distribution of shafts must take into account all dependent costs and savings; the best economic combination will be the one that gives the highest return on venture capital for the project as a whole (P.V. Ratio).

CONCLUSION

An effort has been made to review in broad principle the influence exerted by economic considerations on the selection of the size and distribution of shafts for a mining property. Emphasis has been placed on the philosophy of optimization for the mining prospect as a whole, and the value of a standard computer programme to test for optimization of incremental changes with respect to reference data is discussed.

The cost behaviour of some of the more important parameters is examined, and some practical examples illustrate the fact that many factors not directly related to the technical design problem itself often govern the final selection. These examples are by no means exhaustive, but it is hoped that they will highlight the thought that technical design and planning of the constituent parts cannot be divorced from the economics of the whole. In fact, the final objective of a venture is economic optimization, and good design should be clearly directed towards its achievement.

It must be repeated that the selection process is an iterative procedure that may end with more than one acceptable compromise solution. In the final analysis, the decision will rest with one man who will use his own judgment to discriminate between long- and short-term advantages, taking into account the risks involved and the resources at his disposal. To do this he must have concise information, presented to him in the form of an economic appraisement of each possible choice. A computer print-out of the type illustrated in Appendix 1 is well suited for this purpose.

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REFERENCES

1. Rood, D. M. and Upton, J. H. G. 'The planning, layout and sinking of Vaal Reefs South'. *Association of Mine Managers of South Africa Papers and Discussions*, 1968-1969, pp. 177-208.
2. Bovell, R. C. 'Bafekeng Mine—A general description of the first three years of operations'. *Association of Mine Managers of South Africa, AMM Circular No.* 5/71.
3. Townsend, C. H. 'The sinking and equipping of small diameter shafts at Winkelhaak Mines, Ltd.' *Association of Mine Managers of South Africa Papers and Discussions*, 1960-1961, pp. 47-67.
4. Barenbrug, A. W. T. 'The economic aspect of reducing shaft resistance.' *Symposium on Mine Shaft Design and Its Effect on Air Flow.* The Associated Scientific and Technical Societies of South Africa, Johannesburg, November 1963.
5. Jamieson, D. M., Pearse, M. P. and Plumstead, E. R. A. 'The evolution

of shaft design and sinking techniques in South Africa.' *Transactions of the Seventh Commonwealth Mining and Metallurgical Congress*, 1961.

6. STEED, V. O. 'An analysis of the economics of shaft systems, ventilation and refrigeration in an ultra-deep mine.' *Association of Mine Managers of South Africa Papers and Discussions*, 1960-61, pp. 711-722.

7. GRAVE, D. F. H. and STROH, R. M. 'The planning of ventilation and refrigeration requirements in deep mines.' *J. S.Afr. Inst. Min. Metall.*, January 1972, pp. 171-184.

8. LAMBRECHTS, J. de V. 'Ventilation planning for Western Deep Levels, Ltd.' *Association of Mine Managers of South Africa Papers and Discussions*, 1958-1959, pp. 1017-1031.

9. VAN SCHALKWYK, A. J. H. and VILJOEN, W. W. J. 'Refrigeration at President Brand Gold Mine.' *Association of Mine Managers of South Africa Papers and Discussions*, 1968-69, pp. 743-760.

APPENDIX 2

ESTIMATED EUROPEAN LABOUR COSTS (RANDS)

DESIGNATION	Cost per month	DIAMETER OF SHAFT MEASURED INSIDE LINING (FEET)																			
		18		20		22		24		26		28		30		32		34		36	
		No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost
Master Sinker	1000	1	1000	1	1000	1	1000	1	1000	1	1000	1	1000	1	1000	1	1000	1	1000	1	1000
Shaft Foreman	1000	1	1000	1	1000	2	2000	2	2000	2	2000	3	3000	3	3000	3	3000	3	3000	3	3000
Sinker	1100	3	3300	3	3300	3	3300	3	3300	3	3300	3	3300	3	3300	3	3300	3	3300	3	3300
Sinker's Helper	900	—		—		—		—		3	2700	3	2700	3	2700	3	2700	3	2700	3	2700
Ring Charge Hand	950	1	950	1	950	1	950	1	950	1	950	1	950	1	950	1	950	1	950	1	950
Stage Hand	900	3	2700	3	2700	3	2700	4	3600	4	3600	5	4500	5	4500	6	5400	6	5400	6	5400
Grab Driver	900	3	2700	3	2700	3	2700	3	2700	3	2700	3	2700	3	2700	3	2700	3	2700	3	2700
Fitter	700	3	2100	3	2100	3	2100	3	2100	3	2100	4	2800	4	2800	4	2800	4	2800	4	2800
Electrician	700	2	1400	2	1400	2	1400	2	1400	3	2100	3	2100	3	2100	3	2100	3	2100	3	2100
Boilermaker	700	—		—		1	700	1	700	2	1400	2	1400	3	2100	2	2100	3	2100	3	2100
Rigger	700	—		—		—		—		—		1	700	1	700	1	700	1	700	1	700
Winding Engine Driver	600	3	1800	3	1800	3	1800	3	1800	3	1800	6	3600	6	3600	6	3600	6	3600	6	3600
Banksman	500	3	1500	3	1500	3	1500	3	1500	3	1500	3	1500	3	1500	3	1500	3	1500	3	1500
Surface Handyman	450	3	1350	3	1350	3	1350	3	1350	3	1350	3	1350	3	1350	3	1350	3	1350	3	1350
Shaft Clerk	350	1	350	1	350	1	350	1	350	1	350	1	350	1	350	1	350	1	350	1	350
TOTALS		27	20150	27	20150	29	21850	30	22750	35	26850	42	31950	43	32650	44	33550	44	33550	44	33550
AVERAGE SINKING RATE			700		700		700		700		700		700		650		600		550		500
COST PER LINEAR FOOT			28,9		28,9		31,2		32,5		38,3		45,6		50,2		55,9		61,0		67,1

APPENDIX 3

ESTIMATED BANTU LABOUR COSTS (RANDS)

DESIGNATION	Cost per month	DIAMETER OF SHAFT MEASURED INSIDE LINING (FEET)																			
		18		20		22		24		26		28		30		32		34		36	
		No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost	No.	Cost
Supervision Boss Boy	120	1	120	1	120	1	120	1	120	1	120	1	120	1	120	1	120	1	120	1	120
Machine Boss Boy	115	3	345	3	345	3	345	3	345	3	345	3	345	6	690	6	690	6	690	6	690
Stage Boss Boy	107	—		—		—		—		3	321	3	321	3	321	3	321	3	321	3	321
Timber Boss Boy	107	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321
Lashing Boss Boy	107	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321
Surface Boss Boy	107	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321	3	321
Machine	103	36	3708	42	4326	51	5253	57	5871	66	6798	75	7725	81	8343	96	9888	108	11124	117	12051
Spanner	91	36	3276	42	3822	51	4641	57	5186	66	6006	75	6825	81	7371	96	8736	108	9828	117	10647
Stage	91	6	546	6	546	6	546	6	546	6	546	12	1092	12	1092	12	1092	12	1092	12	1092
Timber	91	18	1638	18	1638	18	1638	18	1638	18	1638	36	3276	36	3276	36	3276	36	3276	36	3276
Lash	79	12	948	12	948	12	948	12	948	15	1185	18	1422	18	1422	24	1896	24	1896	24	1896
Lazy Chain	38	2	76	2	76	2	76	2	76	2	76	4	152	4	152	4	152	4	152	4	152
Batching	61	12	732	12	732	12	732	12	732	18	1098	18	1098	18	1098	24	1464	24	1464	24	1464
Bank	46	12	552	12	552	12	552	12	552	18	828	18	828	18	828	24	1104	24	1104	24	1104
W. E. Driver	41	3	123	3	123	3	123	3	123	3	123	6	246	6	246	6	246	6	246	6	246
Fitter	68	10	680	10	680	10	680	15	1020	15	1020	18	1224	18	1224	20	1360	20	1360	20	1360
Electrician	68	4	272	4	272	4	272	4	272	4	272	6	408	6	408	6	408	6	408	6	408
Boilermaker	68	—		—		3	204	3	204	6	408	6	408	9	612	9	612	9	612	9	612
Rigger	68	—		—		—		—		—		6	408	6	408	6	408	6	408	6	408
Surface Handyman	46	6	276	6	276	6	276	6	276	6	276	6	276	6	276	6	276	6	276	6	276
First Aid/Tea	46	3	138	3	138	3	138	3	138	3	138	3	138	3	138	3	138	3	138	3	138
Induna	114	1	114	1	114	1	114	1	114	1	114	1	114	1	114	1	114	1	114	1	114
Clerk	84	3	252	3	252	4	336	4	336	4	336	5	420	5	420	5	420	6	504	6	504
Call-Out Driver	91	3	273	3	273	3	273	3	273	3	273	3	273	3	273	3	273	3	273	3	273
Waste Disposal Driver	91	3	273	6	546	6	546	6	546	6	546	9	819	9	819	9	819	12	1092	12	1092
Waste Disposal Lab.	46	6	276	12	552	12	552	12	552	12	552	18	828	18	828	18	828	24	1104	24	1104
Police	41	3	123	3	123	3	123	6	246	6	246	6	246	6	246	9	369	9	369	9	369
Fan/Comp/Dump Att.	68	6	408	6	408	6	408	6	408	6	408	6	408	6	408	6	408	6	408	6	408
Lamp Room	41	3	123	3	123	3	123	6	236	6	236	6	236	9	369	9	369	9	369	9	369
TOTALS		201	16231	222	17948	244	19982	267	21722	306	24868	377	30619	398	33465	451	36750	483	39711	503	41457
AVERAGE SINKING RATE			700		700		700		700		700		700		650		600		550		500
COST PER LINEAR FOOT			23,2		25,6		28,5		31,0		35,5		43,7		51,5		61,3		72,2		82,9

APPENDIX 4

ESTIMATED COST OF MAJOR CONSUMABLES (RANDS)

DESIGNATION	DIAMETER OF SHAFT MEASURED INSIDE LINING (FEET)									
	18	20	22	24	26	28	30	32	34	36
Diameter of Shaft Broken	21	23	25	27	29	31	33	35	37	39
Area of Shaft Broken	346	415	491	572	660	760	859	964	1080	1200
Tons Broken per Foot	28,8	34,6	40,9	47,7	55,0	63,3	71,6	80,3	90,0	100,0
Average Length of Round Broken	6	6	6	6	6	6	6	6	6	6
Average Length of Holes Drilled	7	7	7	7	7	7	7	7	7	7
Number of Holes per Round	84	99	115	133	153	175	197	222	248	272
Number of 8 in. Sticks Expl/Hole	10	10	10	10	10	10	10	10	10	10
Number of Sticks/Round	840	990	1150	1330	1530	1750	1970	2220	2480	2720
Cost of Expl/Round at 4,7c/Stick	39,5	46,5	54,1	62,5	71,9	82,3	92,6	104,3	116,6	127,8
Cost of Det. Sets/Round at 20c/Det.	16,8	19,8	23,0	36,6	30,6	35,0	39,4	44,4	49,6	54,4
COST PER TON BROKEN (Cents)										
Explosives	22,9	22,4	22,0	21,8	21,8	21,7	21,6	21,6	21,6	21,3
Detonator Sets	9,7	9,5	9,4	9,3	9,2	9,2	9,2	9,2	9,2	9,1
Drill Steel	13,2	13,2	13,2	13,2	13,2	13,2	13,2	13,2	13,2	13,2
Machine Maintenance and Smalls	11,0	11,0	11,0	11,0	11,0	11,0	11,0	11,0	11,0	11,0
General Oils and Greases	5,5	5,5	5,5	5,5	5,5	5,5	5,5	5,5	5,5	5,5
Compressed Air	8,1	8,0	7,9	7,9	7,9	8,1	8,5	9,0	10,0	11,0
Grab/Stage Maintenance	16,0	16,0	16,0	16,0	16,0	16,0	16,0	16,0	16,0	16,0
Surface Plant Maintenance	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0
Power for Hoisting, Fans, etc.	22,0	22,0	22,0	22,0	22,0	22,0	22,0	22,0	22,0	22,0
Waste Rock Disposal	10,0	10,0	10,0	10,0	10,0	10,0	10,0	10,0	10,0	10,0
Miscellaneous	6,0	6,0	6,0	6,0	6,0	6,0	6,0	6,0	6,0	6,0
	127,4	126,6	126,4	125,7	125,6	125,7	126,0	126,5	127,5	128,1
COST PER LINEAR FOOT (Rands)	37	44	52	60	69	80	90	102	115	128
Mean Diameter of Lining	19,5	21,5	23,5	25,5	27,5	29,5	31,5	33,5	35,5	37,5
Mean Circumference of Lining	61,3	67,5	73,8	80,1	86,4	92,7	99,0	105,2	111,5	117,8
Cu. Yds. Concrete/Linear Foot	3,4	3,8	4,1	4,5	4,8	5,2	5,5	5,8	6,2	6,5
Cost/Linear Foot at R10/cu. Yard	34	38	41	45	48	52	55	58	62	65
TOTAL COST PER LINEAR FOOT	71	82	93	105	117	132	145	160	177	193

APPENDIX 5

SHAFT-SINKING EQUIPMENT AND TEMPORARY STEELWORK (RANDS)

DESIGNATION	DIAMETER OF SHAFT MEASURED INSIDE LINING (FEET)									
	18	20	22	24	26	28	30	32	34	36
Static Weight of Stage (tonnes)	40	45	53	61	70	80	90	110	100	120
No. of Falls of Rope	4	4	6	6	8	8	8	12	12	12
*Cost of Stage Ropes	28000	32000	39000	45000	50000	57000	64000	72000	79000	86000
*Cost of Stage at R450/tonne	18000	20000	24000	27000	32000	36000	41000	45000	50000	54000
*Cost of Lashing Unit + Grabs + Access.	40000	30000	30000	30000	30000	30000	40000	40000	40000	40000
No. of Rock Kibbles	2	2	2	2	2	4	4	4	4	4
Live Capacity of Rock Kibbles	5	6	7	8	9	10	12	12	12	12
*Cost of Kibble Ropes	7000	9000	10000	12000	13000	29000	36000	36000	36000	36000
*Cost of Kibbles, X-Heads, Attachments	5000	6000	7000	8000	9000	10000	12000	12000	12000	12000
No. of Service Kibbles					2				2	2
*Cost of Service Kibbles + Ropes, etc.					10000				10000	10000
*Cost of Temporary Sinking Steelwork	18000	20000	25000	30000	35000	50000	60000	80000	100000	120000
*Cost of Concrete Batching Equipment	15000	20000	25000	30000	35000	40000	48000	55000	65000	75000
*Cost of Ventilating Equipment	8000	8000	10000	10000	10000	12000	12000	16000	16000	16000
*Cost of Stage Hoist	30000	30000	30000	40000	40000	80000	80000	80000	80000	80000
*Misc. Pipes, Cables, Fittings, etc.	25000	26000	27000	28000	29000	37000	39000	46000	48000	50000
TOTALS	194000	201000	227000	260000	293000	381000	432000	482000	532000	579000
AVERAGE COST PER FOOT SUNK	30	31	35	40	45	59	66	74	82	89

*Note: Where appropriate, credit has been allowed for residual value of equipment