A review of some aspects of shaft design

by G. W. HOLL, Pr. Eng., Associate of the Mine Managers Association of South Africa (Fellow)*
and E. G. FAIRON, Pr. Eng., F.S.A.I.E.E., M.I. Cert. M.E.E., M.I.Mech. E., (Visitor) †

SYNOPSIS

The reasons for the choice of vertical shafts in preference to inclined shafts of comparable capacity are set out, and the conclusion that the former are superior in virtually all circumstances is substantiated.

The shapes of vertical shafts sunk during the past sixty years are recorded, and the shapes of those sunk during the past decade are analysed. From this analysis it is clear that circular and elliptical (and quasi-elliptical) shafts are the shafts of the future and that the rectangular shape will conceivably become obsolete.

The determination of the size of shaft is discussed and it is forecast that shafts of a diameter of 13 metres and over will probably be sunk without difficulty in the years to come.

The depth of the shaft is determined intrinsically by the Mines and Works Regulations applicable to factors of safety of winding ropes and the design of the winding system. The Blair winder is the most suitable winder for hoisting rock, men, and material from a depth of, say, 3000 metres on an economic basis. Tertiary shafts will, therefore, be unnecessary in future deep-level mining projects at the limit of depth now envisaged.

A few other factors relating to shaft design are described.

SINOPSIS

Die redes vir die keuse van vertikale skagte in voorkeur tot skuinskagte van vergelykende kapasiteit word uiteengesit en die gevolgtrekking dat die voormalige meerwaardig is in feitlike alle opsigte word bewys.

Die vorm van die vertikale skagte, wat oor die laaste sestig jare opgeteken is, asook die wat oor die afgelope dekade gesink is, word ontleed. Dit word duidelik afgelei van hierdie ontleding dat die sirkelvormige en eliptiese (en byna-eliptiese) skagte die vorms van skagte van die toekoms is en dat die reghoekige vorms desmoontlik uit gebruik sal gaan.

Die bepaling van die grootte van die skag word bespreek en dit word voorspel dat skagte van 'n deersnee van 13 meter en meer moontlik sonder moeite in die toekomstige jare gesink sal word.

Die diepte van die skag word bepaal deur ingewikkelde Myn en Werks Regulasies wat toepaslik is op faktore van veiligheid van wentoue asook die ontwerp van die hystoestelsisteem. Die Blair hysmasjien is mees geskik vir die die hys van rots, mense en materiaal vanaf 'n diepte van ongeveer 3000 meters op 'n ekonomiese basis. Tersiêre skagte sal desmoontlik onnodig word in die toekoms van diepvlak myn projekte tot op die diepte wat tans in sig is.

Sommige ander verwante faktore van skag ontwerp word ook beskrywe.

INTRODUCTION

In the past much has been written about shaft-sinking operations, almost invariably incorporating a description of the design features of the particular shaft as part and parcel of the general description of the works. Less literature is available on the basic work of designing shafts in general, though, as a general source of information, the Transactions of the Institute of Mining Engineers, London, those of the Mine Managers Association of South Africa, and those of this Institute may be consulted.

The authors of the present paper will attempt to treat the subject from the more basic aspects in the hope that some useful information will flow and that general discussion may bring out further amplification of the ideas mentioned in this paper.

Shafts have, in the past, generally been associated with mines, but more and more shafts are being sunk for purposes other than to produce ore from the earth's crust. These non-mine shafts serve as means of access to excavations used for the:

(i) diversion of river systems;
(ii) production of hydro-electric power;
(iii) provision of underground road and railway facilities;
(iv) underground storage of water, oil, and other commodities, e.g. atomic waste materials;
(v) foundation structures of high-rise buildings.

The excavations for high-rise buildings initially provide valuable information on the bearing characteristics of the soil and rock formations on which the building will be constructed and, finally, when filled with concrete, form the main foundation support of the high-rise building.

It is generally true to say that each shaft is tailor-made for the work envisaged during the life of the project it is to serve. In mining, the essential feature in determining the cross-section of the shaft is the quantity of air to be supplied to the workings, whereas, in civil-engineering projects, it is the size of the machinery and the equipment to be handled in the shaft that determines the shaft area.

In the Republic of South Africa a shaft is defined as:

"any tunnel having a cross-sectional dimension of twelve feet and over and

(a) having an inclination to the horizontal of fifteen degrees or over, or
(b) having an inclination to the horizontal of less than 15 degrees but more than 10 degrees, where the speed of traction may exceed four hundred feet per minute [122 metres per minute]".

This definition is designed to distinguish a shaft from a winze or inclined rope haulage, the regulations for which are much less stringent. For the purpose of this paper all tunnels having an inclination of more than 10 degrees from the horizontal will be treated as shafts.

* Managing Director, Shaft Sinkers (Pty.) Ltd, and Mining and Engineering Technical Services (Pty.) Ltd.
† General Manager, Mining & Engineering Technical Services (Pty) Ltd.

SPECIFIC TYPES OF SHAFT

For the purpose of this paper there are only two types of shaft:

(i) inclined shafts, and
(ii) vertical shafts.

In the past, a large number of inclined shafts were sunk for the exploitation of mines or sections of mines, but this type of shaft has now become almost obsolete as a major production unit. Mullins and Haggie[1] give the following information on the shafts serving 105 mines in the Union of South Africa at 31st December, 1959:

"Of these shafts 315 are vertical and 208 are inclined. There are no compound shafts but several of the inclined shafts vary in inclination along their length. Of these shafts, 395 are rectangular, 119 are circular, and 8 are elliptical in section."

From this statement it is clear that, at that date, no fewer than 40 per cent of the shafts were inclined, and the vast majority were rectangular.

Although the vertical shaft is generally favoured in mines under virtually all conditions, the authors are not aware of any literature that clearly sets out the reasons for this preference. The following argument is an attempt to define as accurately as possible why, for a comparable output of ore, men, and material, vertical shafts are superior to inclined shafts.

At present the use of the inclined shaft is limited to:

(i) a prospecting operation;
(ii) the provision of ingress to and exit from workings associated with high-production tabular-ore deposits, particularly for large items of machinery and for the conveyance of men and material by high-capacity buses and lorries; (the ore produced from these mines is nevertheless hoisted through vertical shafts);
(iii) coal mines, where the deposit is shallow and a conveyor belt or endless-rope system is used for the conveyance of coal; men invariably walk into the mine, and the material is minimal in relation to the mineral output.

In the first case, the inclined shafts are of small size and seldom consist of more than three small compartments. These shafts are placed on or close to the orebody and, at depth, are usually overmined almost immediately because of their extreme vulnerability to the rock stresses that develop as soon as stoping operations begin. Several such shafts would be required to cater for the output of the average mine because each unit caters for only a limited mining area.

In shafts used exclusively for the movement of men and materials, the inclination is limited to at most 14 degrees and, more usually, only 10 degrees or less because of the effect of steep gradients on the size of engine required for the buses conveying personnel and for the lorries transporting the materials required underground. As the depth of the workings increases, these inclines become extremely long, and certainly at depth would become uneconomical to maintain because of their size, which provides for two lanes of traffic.

Inclined shafts have a limited length, the longest known inclines being of the order of 2043 metres. These shafts therefore cater for only a limited depth, depending on the inclination of the orebody. At inclinations of below 25 degrees, the length of incline to reach the very limited depth of 1000 metres becomes virtually prohibitive, i.e. 2530 metres at 25 degrees and 4085 metres at 15 degrees. Finally, it is estimated that the cost of maintenance and power consumption in inclined shafts is at least 3½ cents per tonne hoisted more than that for vertical shafts of equal capacity.

To illustrate the actual relation between the effectiveness of inclined and vertical shafts, the following model was studied.

On the assumption of a rate of 67 000 tonnes hoisted per month through a shaft, and of a moderate vertical depth of mining area of 976 metres, a layout of the shaft and the necessary tunnel for the exploitation of orebodies of various inclinations was prepared. To make the comparison as accurate as possible, the layout for the inclined shaft incorporates tunnels into the footwall so that only two main hoisting points are established over their length as would be the case in a vertical shaft. Further, the inclined shaft is placed 90 metres, measured in a horizontal plane, from the orebody to permit adequate storage space on each station for materials that are to be transported to the workings by locomotives, as well as to permit the transport of ore and waste to the orepasses (see Fig. 1).

Using this model, a series of layouts was prepared for inclinations from the horizontal of the orebody from 15 degrees to 75 degrees in 10-degree steps. The length of the two types of shaft and the horizontal tunnels applicable to this model were thereafter measured. The results shown in Table I were obtained (see also Graph I).

The cost of sinking, lining, and equipping an inclined shaft having an output comparable with that of a vertical shaft is at best approximately identical. Therefore, on the basis of shaft length only, inclined shafts could not be considered as a serious competitor to vertical shafts.

The fundamental consideration, however, is the amount of horizontal development required to reach the orebody and also to provide for at most two loading points in the shaft system. The length of the ore-pass systems is then common to both shaft systems. In the first instance, a vertical shaft placed in such a

TABLE I
Comparison of lengths of shafts

Inclination of orebody degrees	Length of vertical shaft metres	Length of incline shaft metres	Percentage increase in length %
15	1098	4085	272,0
20	1098	3110	183,2
25	1098	2531	130,5
35	1098	1890	72,1
45	1098	1555	41,6
55	1098	1341	22,2
65	1098	1220	11,1
75	1098	1159	5,6

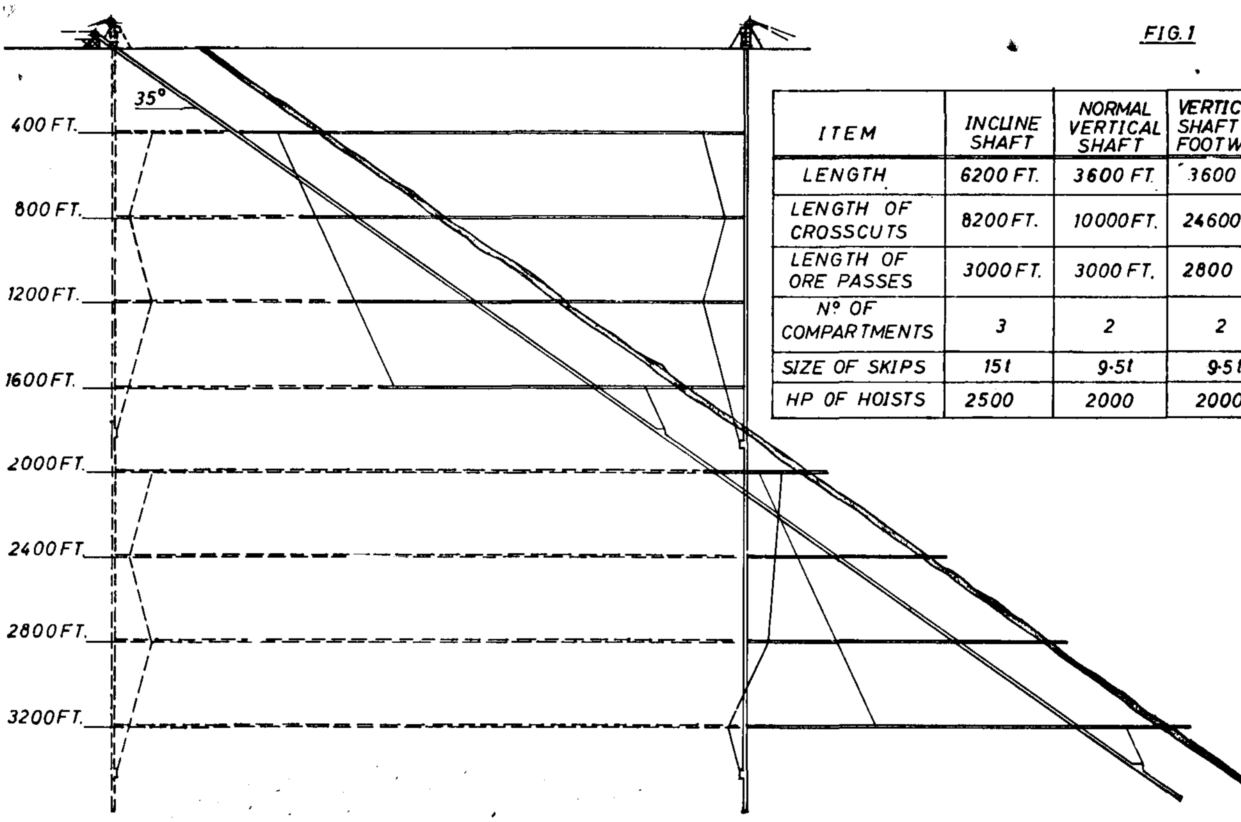


FIG. 1

ITEM	INCLINE SHAFT	NORMAL VERTICAL SHAFT	VERTICAL SHAFT IN FOOTWALL
LENGTH	6200 FT.	3600 FT.	3600 FT.
LENGTH OF CROSSCUTS	8200 FT.	10000 FT.	24600 FT.
LENGTH OF ORE PASSES	3000 FT.	3000 FT.	2800 FT.
Nº OF COMPARTMENTS	3	2	2
SIZE OF SKIPS	15 t	9·5 t	9·5 t
HP OF HOISTS	2500	2000	2000

position as to pierce the orebody, thereby requiring a shaft pillar (termed a conventional vertical shaft) is considered and, in the second case, a vertical shaft placed in the footwall of the orebody (termed a footwall vertical shaft). Table II and Graph II show the relationship between the horizontal tunnel lengths required for the conventional vertical shaft as compared with the inclined shaft, and Table III and Graph III show the comparison relating to the footwall vertical shaft.

There is again nothing in favour of inclined shafts under these circumstances, since the horizontal tunnels are at most 30 per cent greater at the flatter inclinations and the steeper inclinations favour the vertical shaft. The average 25 per cent increase in horizontal tunnel length is easily offset by the increased length of the inclined shaft required.

Under these circumstances, the advantage of the inclined shaft is apparently well established on the basis of the additional horizontal development required. However, for comparative outputs, the overall

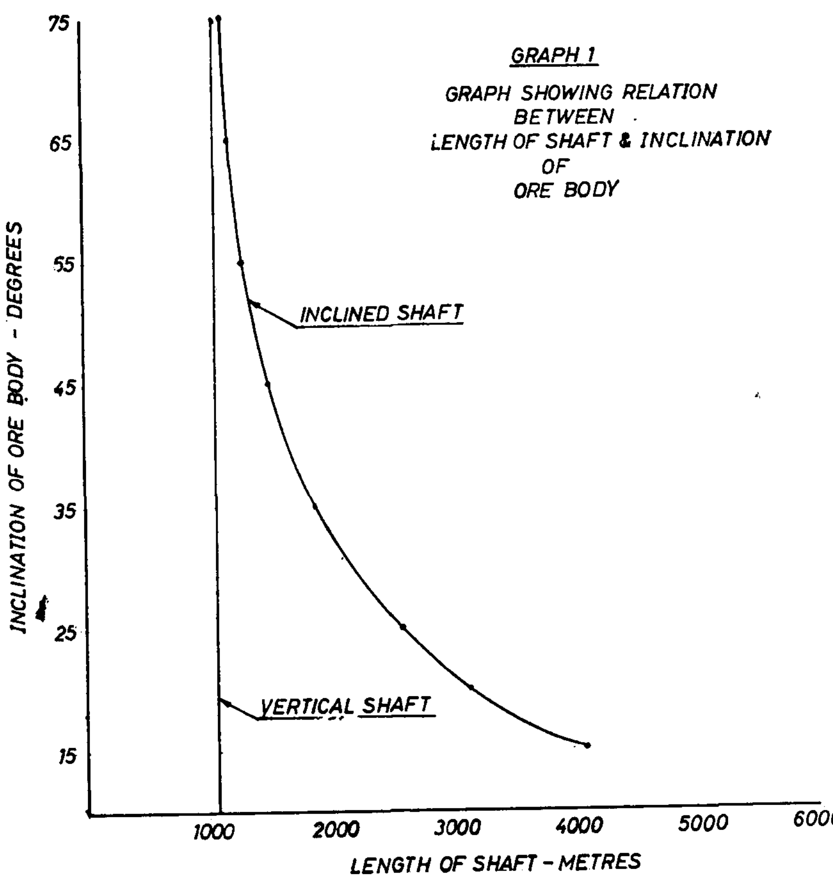


TABLE II
Comparison of horizontal tunnel lengths for inclined shaft versus conventional vertical shaft

Inclination of orebody degrees	Lengths of horizontal tunnel		Percentage increase over incline %
	Inclined shaft metres	Vertical shaft metres	
15	6066	7742	+27,6
20	4450	5791	+30,1
25	3672	4450	+22,7
35	2499	3048	+22,0
45	1707	2225	+30,3
55	1402	1646	+17,4
65	1219	1158	− 5,0
75	1219	853	−42,9

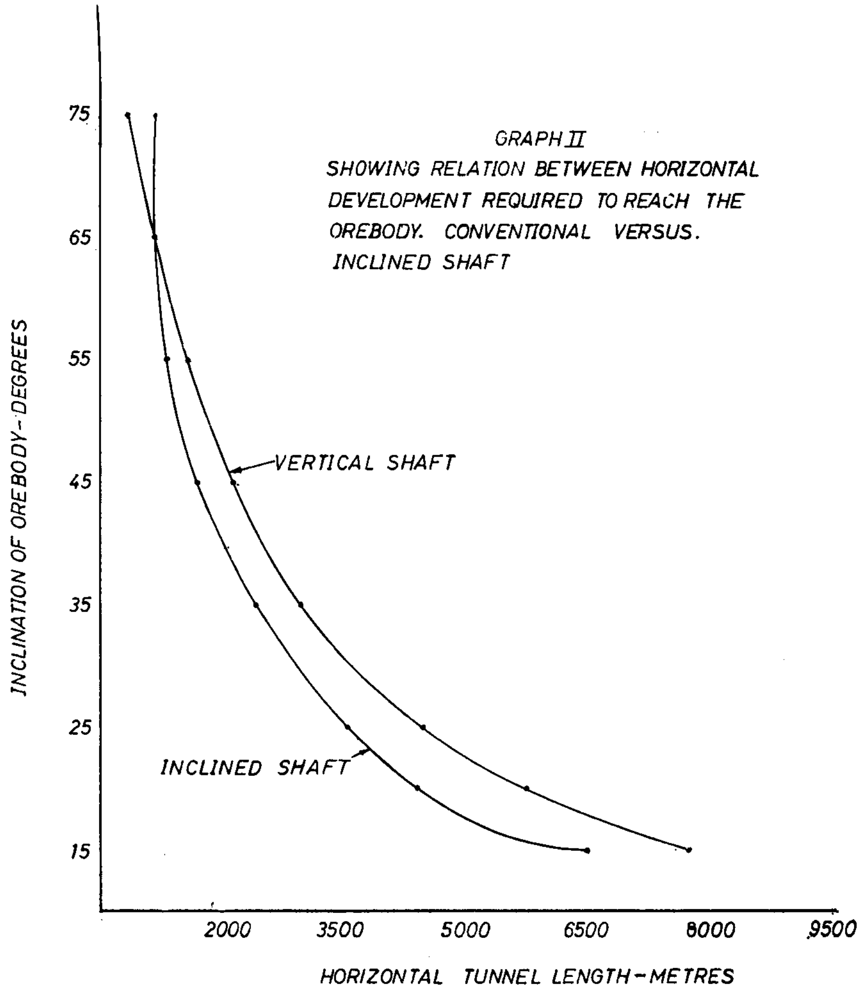


TABLE III
Comparison of horizontal tunnel lengths for inclined shaft versus footwall vertical shaft

Inclination of orebody degrees	Horizontal length of tunnel required		Percentage increase %
	Inclined shaft metres	Footwall vertical shaft metres	
15	6066	17 831	193,9
20	4450	13 411	201,4
25	3627	10 790	195,5
35	2499	7 437	197,6
45	1707	5 608	228,5
55	1402	4 328	208,7
65	1219	3 231	165,1
75	1219	2 316	90,0

costs of the shaft are approximately equal and, if it is accepted that the equipped inclined shaft is five times as expensive as the fully equipped development, then the more accurate comparison should be as shown in Table IV.

From Table IV it is obvious that the inclined shaft under these circumstances should be considered only for inclinations of 30 degrees upwards, where the design and operational difficulties are highest, and, even then, it has only a limited advantage of some 1711 metres of development, which, on the basis considered here, is equivalent to 342 metres of inclined-shaft length, or in monetary terms some R450 000.

However, the following additional factors require consideration.

(a) In comparison with the conventional vertical shaft, there is the question of the loss of the financial return on the profits, over the life of the project, from the ore locked up in the shaft pillar required for the vertical type of shaft, and the subsequent problem of extracting this ore at the end of the project's life.

This aspect could be looked at in two ways. Either the shaft pillar could be extracted at the beginning of the mining operations, when it is generally a comparatively simple operation, or, where this step is not advisable, e.g. in water-saturated formations, a monetary value could be assigned to this factor. For the depth envisaged in the selected model, the shaft pillar would not exceed a radius of 300 metres and would contain, over a stope width of 2 metres, not more than, say, 153 000 tonnes of ore. At a profit margin of R2 per tonne, the loss of interest on this profit over a 30-year life of the project would, at 7½ per cent per annum, amount to R2,7M.

(b) The cost of maintenance of the inclined shaft in comparison with that of the vertical shaft will offset the above advantage completely over the same period.

It is estimated that this difference amounts to approximately 3½ cents per tonne hoisted. The monetary value of this factor over 30 years is assessed at R2,9M at 7½ per cent per annum, and this amount offsets the first-mentioned consideration completely. In addition, this aspect completely answers the question of the additional development from the use of the footwall vertical shaft, when no shaft pillar is required, as compared with that of the inclined shaft.

(c) The rate of exploitation of the orebody should the project relate to a new mine.

   (i) The average speed of sinking, lining, station-cutting, and equipping an inclined shaft is extremely low. It is 22 metres per month as compared with 66 metres per month for the similarly equipped vertical shaft.

   (ii) The average hoisting speed is 530 as against 915 metres per minute — 1 to 1,7 in favour of the vertical shaft. The length of the wind is as illustrated in Table I.

Thus, Table V illustrates the estimated time of completion, the skip sizes, and the horsepower requirements of the two types of shaft for an output of 75 000 tonnes per month from an orebody dipping at 35 degrees.

There is therefore much to be said in favour of the vertical shaft if a new project is being considered.

Often, however, it could be desirable to complete half of the project in order to expedite the date of production, and thereafter to deepen the shafts as a secondary operation while production continues. This aspect is often cited as one where the inclined shaft has an advantage over the vertical shaft. In such a case, the comparison shown in Table VI results.

In this case it is clear that the vertical shaft could be sunk and the reef horizon reached in approximately half the period required for the inclined shaft during the first stage. The increased amount of development is accentuated in the second

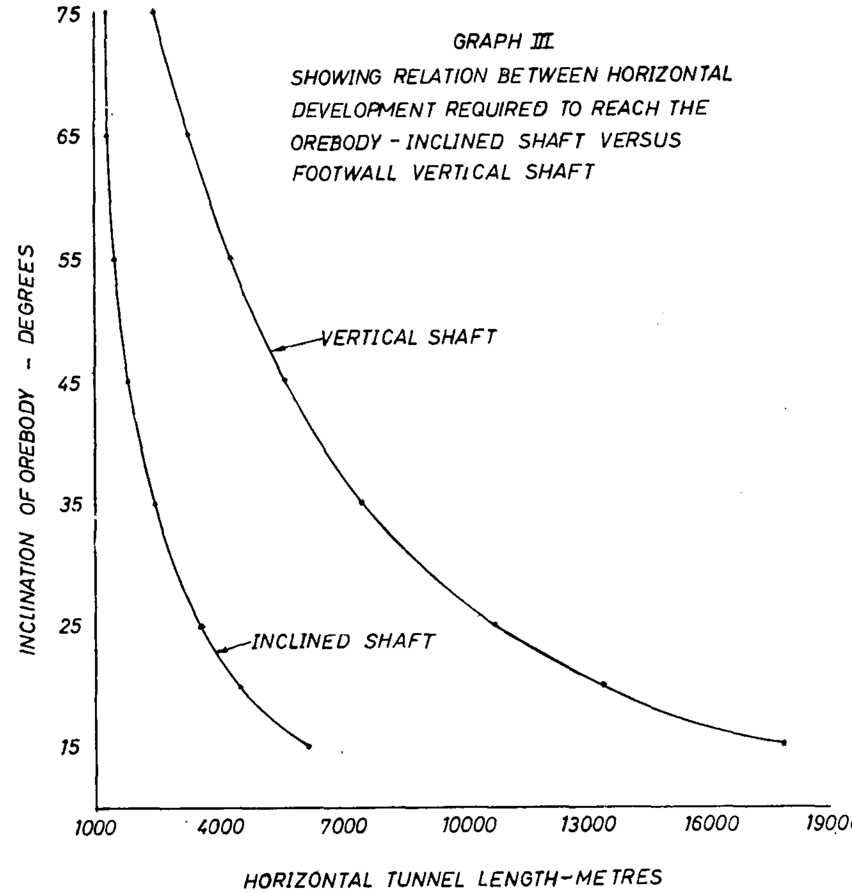


TABLE IV

Comparison of horizontal tunnel lengths (adjusted for inclined shaft length) for inclined shaft versus vertical shaft

Inclination of orebody degrees	Additional length of inclined shaft over that of vertical shaft metres	Value of this additional length of shaft expressed as development metres	Additional horizontal development metres	Net additional development for vertical shafts metres
15	2609	13 045	11 765	−1280
20	2012	10 060	8 961	−1099
25	1433	7 165	7 163	− 2
35	792	3 960	4 938	+ 978
45	457	2 285	3 901	+1616
55	243	1 215	2 926	+1711
65	122	610	2 012	+1402
75	61	305	1 097	+ 792

TABLE V

Conparison of lead times, skip size, and winders

OPERATION	Elapsed time, months	
	Inclined shaft months	Footwall vertical shaft months
1. Prepare to sink	3,0	6,0
2. Sinking time	70,7	16,7
3. Equipping	0,0	2,0
4. Development from stations	2,0	2,0
5. Further development to orepasses	1,5	0,5
6. Orepasses by raiseborer	3,0	0,4 concurrent
7. Development	4,0	34,0
TOTAL:	84,2	61,6
Size of skips	15,0 ton	9,5 ton
Horsepower of winder	2500	2000

TABLE VI
Comparison of completion times of shafts

Stage of project	ITEM	Inclined shaft metres	Footwall vertical shaft metres
First	Length of shaft	854	549
	Length of horizontal development	854	1738
Second	Length of shaft	854	549
	Length of horizontal development	854	3750
	∴ ESTIMATED COMPLETION TIME		

Operation first stage	Elapsed time, months	
	Inclined shaft	Footwall vertical shaft
1. Prepare to sink	3,0	6,0
2. Sinking	38,8	8,0
3. Equipping	0,0	1,0
4. Development on stations	2,0	2,0
5. Development for orepasses	1,5	0,5
6. Orepasses	3,0	0,4 concurrent
7. Development to reef	2,0	6,0
TOTAL	50,3	23,9

stage. There is no great difference in the problem of deepening vertical shafts as opposed to inclined shafts.

(d) For comparative outputs the horsepower demands of the winders required for the two types of shaft favour the vertical shaft. Design and operational difficulties are encountered on the steeper ranges of inclinations between 35 and 75 degrees.

Table VII indicates the comparison arrived at for shafts handling 75 000 tonnes per month, together with the men and material requirements for that rate of production.

With due consideration to all these factors, the general conclusion reached is that, as a major production unit, the vertical shaft is superior to the inclined shaft of comparable capacity under all circumstances.

SHAPE OF SHAFT

The recognized shape of the excavation for vertical shafts can be any of the following:

(a) rectangular, narrow, wide or square;
(b) circular;
(c) elliptical (or quasi-elliptical, i.e. distended circular).

Inclined shafts are excavated to the following shapes:

(a) rectangular with or without arched roof;
(b) elliptical (at great depth).

An analysis of the vertical shafts that were sunk or were being sunk over the period January 1961 to December 1971 is given in Table VIII.

For the purpose of comparison, Table IX was compiled from information supplied by Jamieson et al.[2]

The increased use of the circular and elliptical shapes is due to development of mechanical cleaning in the first instance and, finally, to the development of the principle of concurrent lining whilst sinking proceeds. The circular shape is also the most efficient from the ventilation aspect. Great credit is due to those who, through their tenacity of purpose, painstakingly developed the methods to make possible the results at present being achieved in vertical-shaft sinking operations.

It is clear, therefore, that during the past decade the rectangular shape was used only for shafts with an area of less than 30 square metres. For larger shafts, the circular or elliptical shapes were favoured.

The elliptical or distended shape is returning to favour because it has virtually all the advantages of the circular shape but provides for a better utilization of space and, when required, a lesser length of brattice wall section. In some instances, therefore, the elliptical shape could be preferred to the circular.

SIZE OF SHAFT

In the case of mines, the size of shaft used is generally determined by the volume of air required to ventilate the prospective mine. Only in the case of small shafts is the size determined by the equipment to be fitted into it and/or handled through it. On the other hand, in the case of access to underground works of a civil nature, the size of the shafts generally is determined by the size of the equipment to be handled through them.

Since, at the time the shaft or shafts for a new mine are being designed, the general layout of the workings is largely one of conjecture, the volume of air is based on the estimated quantity of rock to be broken in the mine.

TABLE VII
Comparison of hoisting details

Requirement	Inclination of orebody, degrees					
	20	35	45	60	75	90
Hoisting distance, metres	3029	1806	1465	1196	1073	1036
Persons per carriage	180	120	100	100	100	80
Capacity of skip, tonnes	25,5	15	12,5	10	9,75	9,5
Rope diameter, millimetres	47	47	47	47	47	46
R.M.S. horsepower of winder	3215	2500	2350	2160	2088	2000

TABLE VIII
Vertical shafts sunk 1961-1971
(based on information kindly supplied by the Government Engineer)

Type and function of shaft	Area of shaft in square metres				Total
	Very small up to 20	Small 20 to 35	Medium 35 to 50	Large over 50	
(a) RECTANGULAR SHAFTS					
Production	17	2	NIL	NIL	19
Service	9	NIL	NIL	NIL	9
Coal service	2	NIL	NIL	NIL	2
TOTAL RECTANGULAR	28	2	NIL	NIL	30
(b) CIRCULAR					
Production	12	5	22	8	47
Service	10	6	8	3	27
Sub-production	3	2	18	3	26
Sub-service	12	3	NIL	1	16
Coal service	25	15	5	NIL	45
TOTAL CIRCULAR	62	31	53	15	161
(c) ELLIPTICAL					
Sub-production	2	2	1	1	6
Sub-service	5	NIL	NIL	1	6
Coal service	1	NIL	NIL	NIL	1
Sub-shafts production	NIL	NIL	NIL	2	2
TOTAL ELLIPTICAL	8	2	1	4	15
GRAND TOTAL	98	35	54	19	206
Percentage	47,6	17,0	26,2	9,2	100,0

SUMMARY

	Percentage of each type			Total
	Rectangular	Circular	Elliptical	
Very small	28,57	63,27	8,16	100
Small	5,71	88,58	5,71	100
Medium	Nil	98,15	1,85	100
Large	Nil	78,95	21,05	100
All shapes, number	30	161	15	206
Percentage	14,56	78,16	7,2	100%

This state of affairs is no reflection on the skill of the geologist in interpolating the possible strike and dip of the deposit, nor the faulting, but is purely due to the fact that the borehole and reef intersections normally available to these specialists are not sufficient to permit even a reasonably accurate forecast. Because of this factor, the ventilation engineer cannot with any degree of accuracy calculate the volume of air required for the prospective mine.

Further, it has been established that the velocity of the air current in downcast shafts is generally limited to a figure of 500-600 metres per minute, whereas in the upcast shaft the limit approaches 1200 metres per minute.

The mining engineer in designing the size of the shafts has, therefore, perforce to allow for the best possible solution having regard to the information available from other mines. Fortunately, the Chamber of Mines of South Africa provides, through its Research Department, a wealth of information relating to this subject in its Annual Ventilation Reports.

The figures of interest in the 1970 report are as follows.

(i) The quantity of air provided per minute per tonne, broken into:
Average volume per mine = 0,162 cubic metres
Maximum = 0,40 cubic metres
Minimum = 0,052 cubic metres

(ii) The quantity of air provided underground averages 5,094 cubic metres per minute per person underground.

From the results of the mines where the average virgin rock temperature is below 37,7 °C and where ventilation conditions are known to be reasonably good, the average figure for the quantity of air per minute per tonne broken per month is 0,15 cubic metres.

An average of selected values for mines where the virgin rock temperature is over 37,7 °C shows that a figure of 0,2 cubic metres per minute per tonne broken is a more reasonable requirement.

The size of the shaft can be calculated from these two results, coupled with the knowledge that for comfort the velocity in the downcast shaft should be 500 metres per minute and not exceed 600 metres per minute. An allowance of 15 per cent is made for the obstruction caused by the shaft steelwork and other fixed equipment.

As stated previously, the velocity of the return air in the upcast shaft is generally limited to not more than 1200 metres per minute to ensure good fan efficiency in the shaft.

The following calculations determine the size of circular vertical shafts required for an output of

TABLE IX
Types of shafts sunk 1910-1971

Period	Type of Shaft			Totals
	Rectangular	Circular	Elliptical	
1910-1947				
Number	341	41	7	389
Percentage	87,7	10,5	1,8	100
1947-1960				
Number	51	72	5	128
Percentage	39,8	56,3	3,9	100
1961-1971				
Number	30	161	15	206
Percentage	14,6	78,2	7,2	100

	Using a factor of 0,15 cubic metres per minute per tonne mined per month	Using a factor of 0,20 cubic metres per minute per tonne mined per month
(a) Downcast Shafts Quantity, cubic metres per minute	67 000≥0,15=10,050	67 000≥0,20=13,400
(i) Area required with an air velocity of 600 m per minute	10,050/600=16,75 m^2	13,400/600=22,33 m^2
(ii) Area required with an air velocity of 500 m per minute	10,500/500=20,10 m^2	13,400/500=26,80 m^2
(iii) Adjustment of these areas by 15% for obstruction gives the shaft areas required		
(i) above	19,70 m^2	26,27
(ii) above	23,64 m^2	31,53
Diameter of shaft	. . . . .	
(i) above	5,00 m (=16,4 ft)	5,78 m (=19,0 ft)
(ii) above	5,50 m (=18,0 ft)	6,34 m (=20,8 ft)
(b) Upcast Shafts Shaft area required	10 050/1 200=8,375 m^2	13 400/1 200=11,167 m^2
Diameter of shaft	=3,27 m (=10,7 ft)	=3,76 m (=12,3 ft)

TABLE X
Diameter of downcast shafts, metres

Tonnes planned to be mined	Virgin rock temperatures							
	−37,8 °C Velocity of air current metres per minute				+37,8 °C Velocity of air current metres per minute			
	(al)* 600		(a2) 500		(a3) 600		(a4) 500	
67 000	5,00	(16,5)†	5,48	(18,0)	5,98	(19,5)	6,34	(20,75)
75 000	5,30	(17,5)	5,80	(19,0)	6,12	(20,0)	6,70	(22,0)
100 000	6,06	(20,0)	6,70	(22,0)	7,07	(23,0)	7,74	(25,5)
150 000	7,50	(24,5)	8,21	(27,0)	8,66	(28,5)	9,48	(31,0)
200 000	8,66	(28,5)	9,57	(31,5)	9,96	(32,75)	10,94	(36,0)
250 000	9,67	(31,75)	10,60	(34,75)	11,06	(36,0)	12,24	(40,0)
300 000	10,60	(34,75)	11,61	(38,0)	12,24	(40,0)	13,40	(44,0)

* (al), (a2), (a3) and (a4) refer to curves on Graph IV
† Figures in brackets denote nearest convenient conversion to feet.

TABLE XI
Diameter of upcast shafts, metres

Tonnes planned to be mined	Virgin rock temperatures			
	−37 °C 600 M (b1)*		+37,7 °C 500 M (b2)	
67 000	3,27	(11,0)†	3,76	(12,5)
75 000	3,45	(11,5)	3,99	(13,0)
100 000	3,99	(13,0)	4,60	(15,0)
150 000	4,90	(16,0)	5,64	(18,5)
200 000	5,64	(18,5)	6,54	(21,5)
250 000	6,30	(20,5)	7,28	(24,0)
300 000	6,90	(22,5)	7,98	(26,0)

* (b1) and (b2) refer to curves on Graph IV
† Figures in brackets denote nearest convenient conversion to feet.

67 000 tonnes broken per month.

From similar calculations, Table X and Graph IV have been compiled, from which it is easy to determine the range of diameters of the shafts that would be required for planned tonnages to be mined from 67 000 up to 300 000 tonnes per month.

Because of the large capital outlay required for shafts, it is conceivable that shafts of larger diameter than those extremes indicated here could well be sunk in the future, particularly when the large base-metal mines that are now open-cast have to mine underground in depth.

In a new mine it is generally accepted practice to sink twin vertical circular shafts in order to provide for the second outlet, as required by law, as soon as possible, and also to arrange for adequate ventilation from the inception of mining operations.

Where an additional shaft is required for the expansion of production or for the mining of the deeper levels, or where a new mine could establish a second outlet through a neighbouring mine, the bratticed shaft provides a more economical answer under certain conditions. In such a case the two areas required for downcast and upcast ventilation are calculated as before and, allowing for the area covered by the brattice wall, the sum of the total area required is derived and the overall diameter calculated.

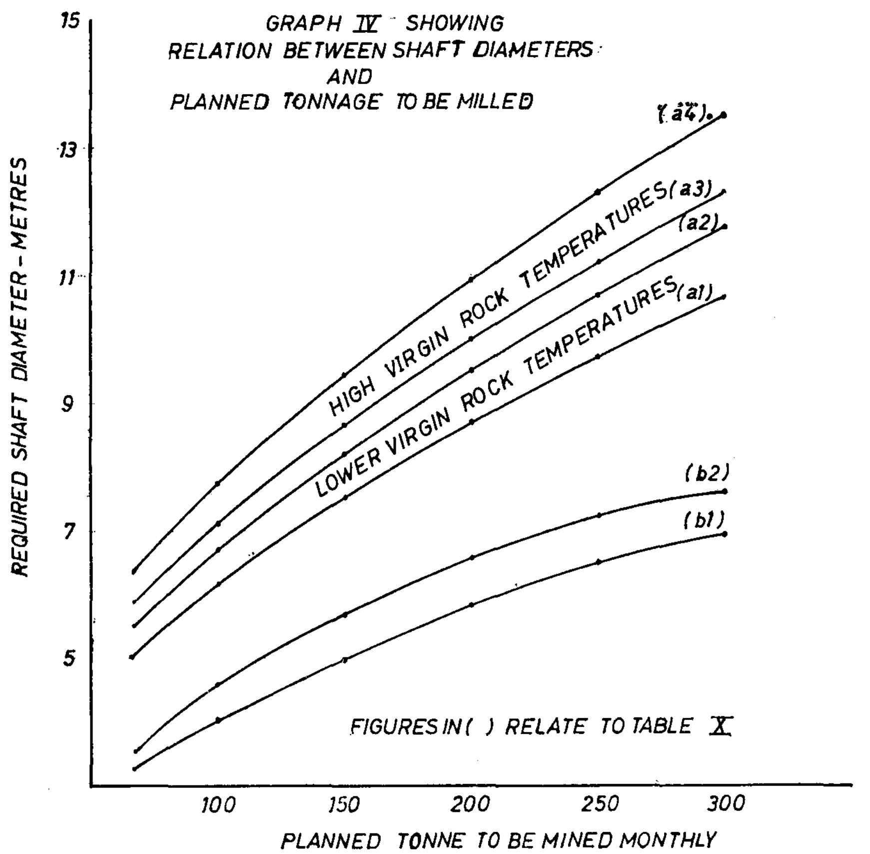
Examples of such dual purpose shafts are:

Vaal Reefs South Bratticed Section—Two circles 8,53 metres in diameter, with centres spaced 0,6 metre apart

Elsburg Gold Mining Company Limited—Circular, 10,21 metres in diameter

President Steyn Number 4 Shaft—Two circles 10,21 metres in diameter and a straight section 0,76 metre long joining these.

Although these large shafts were sunk through igneous and sedimentary rocks, as well as heavy water-bearing strata, no particular difficulties were encountered. In the case of another large downcast shaft some difficulties arose where a major steeply inclined fault was intersected. The stress induced in



of small sectional area but large capacity because of their extended depth. In this manner shaft sizes have been kept to a minimum, but the output has increased significantly. In one case study, a shaft of 7,93 metres diameter is handling an average of 227 423 tonnes per month regularly (with a record month of 239 000 tonnes), plus all the personnel and about 90 per cent of the material requirements. This shaft area obviously cannot handle the ventilation requirements for this rate of production, but ventilation is augmented through another shaft.

As an example of the use of shafts in civil-engineering projects, the proposed St. Gotthard Railway Tunnel may be cited.

A preliminary design has been proposed for three vertical shafts to be sunk enabling the construction of the new Gotthard Railway Tunnel between Erstfelt in Switzerland and the new Biasca station in Italy. This tunnel will be approximately 46 kilometres long, and bored 10 metres in diameter.

The shaft design was based on the

the rocks by the fault, together with the slickensiding, caused serious overbreak, but the shaft was completed successfully.

The high cost of shafts calls for a minimum number per mine. With such large-diameter shafts passing correspondingly large quantities of air for ventilation purposes, the areas on strike and dip served by such shafts are considerably larger than was thought possible a decade ago. In order to convey the volume of air to the workings, twin, or even treble, tunnels are necessary at times. Further, the use of refrigeration enables the re-use of air, and in this manner the area served by shafts is capable of a further extension or, conversely, the total quantity can be reduced and thereby the size of shaft. This latter alternative holds particularly in deep-level mines where the air has to be refrigerated, in any event, sooner or later, because of the rapid transfer of heat from rock to air in its passage through the workings at high virgin-rock temperatures.

The bottom discharge skip has made it possible to fit in conveyances

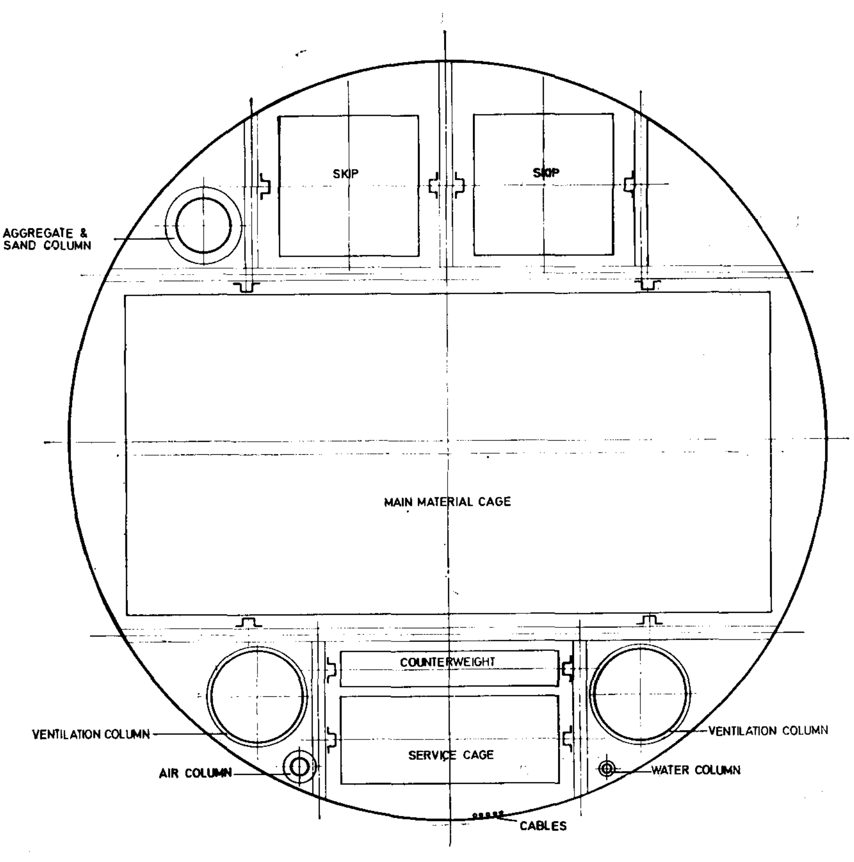


Fig. 2—Permanent configuration

information supplied. The controlling factors in the design were the considerable tonnages of rock produced by the two tunnel-boring machines in each shaft, which have to be hoisted daily, and the lowering and raising of the largest component of the tunnel-boring machines, which is the bearing. This item is 5 metres in diameter, 3 metres wide, and weighs 30 tonnes. In addition, the lowering of the daily supplies for the construction of the tunnel, consisting of heavy precast-concrete tunnel lining and draining segments, as well as rails, sleepers, pipes, machine spares, and the labour force, is to be catered for.

A service cage is provided to permit maintenance of temporary electrical cables, ventilation, aggregate, water, and air columns. Finally, on completion of the works, the shafts will be stripped and used for ventilation purposes and power supply. The service cage will be retained for maintenance of the power cables.

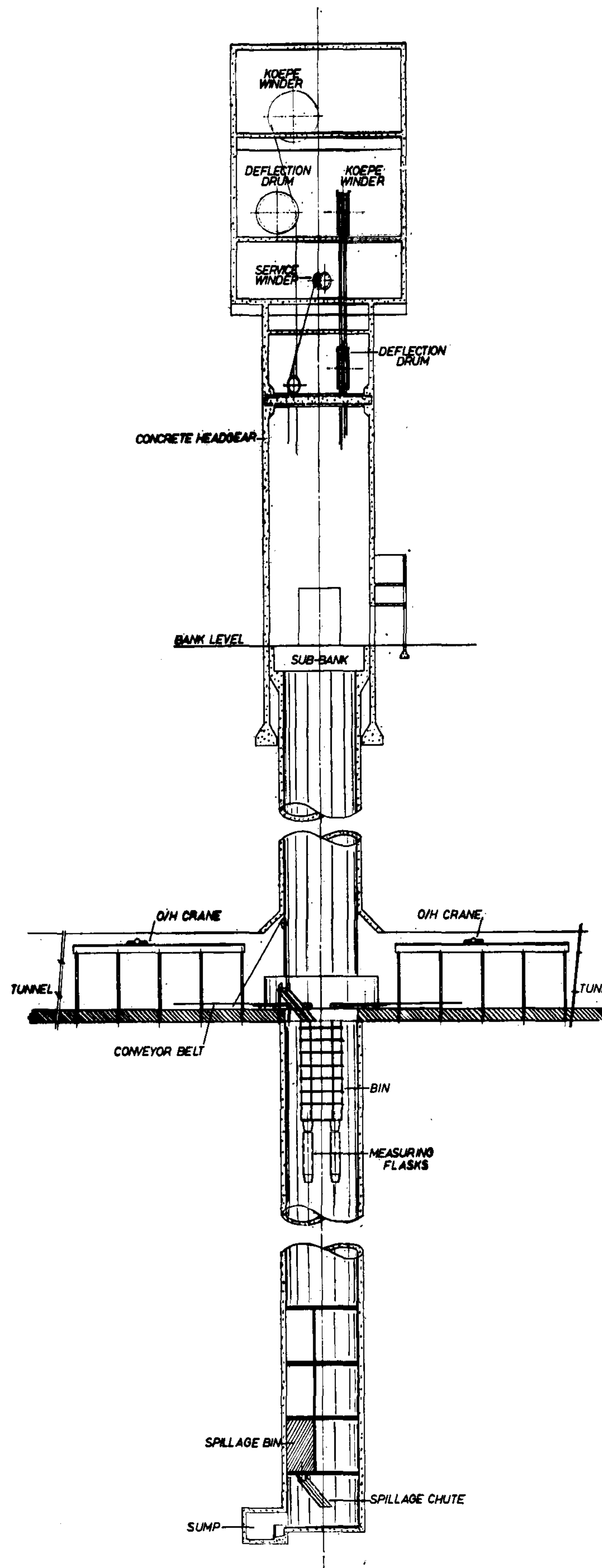
It was also necessary to allow for the extremely low site temperatures in winter, which could reach $-30\,°C$.

The proposed shaft configuration during the tunnel-construction period is shown in Fig. 2.

At the tunnel elevation, the rock will be delivered from either side of the shaft by means of a conveyor and delivered into a loading box; from this box the rock is to be discharged into two measuring flasks and hoisted direct to the surface. A suitable spillage arrangement is also to be provided. (See Figs. 2 and 3.)

A concrete headgear has been proposed with tower-mounted Koepe winders to cater for the adverse climatic conditions anticipated. The minimum size for this shaft is found to be 8,5 metres, largely because of the size of cage required to lower the main bearing of the boring machine and because of the Koepe winders.

Should it be possible to revert to normal double-drum winding with the possibility of slinging the bearing underneath the cage, the shaft diameter could be reduced to 7 metres, and this, because of the saving in shaft space, is what may finally be done.



PROPOSED SHAFT FOR GOTTHARD RAILWAY TUNNEL (DIAGRAMMATIC)

FIG. 3

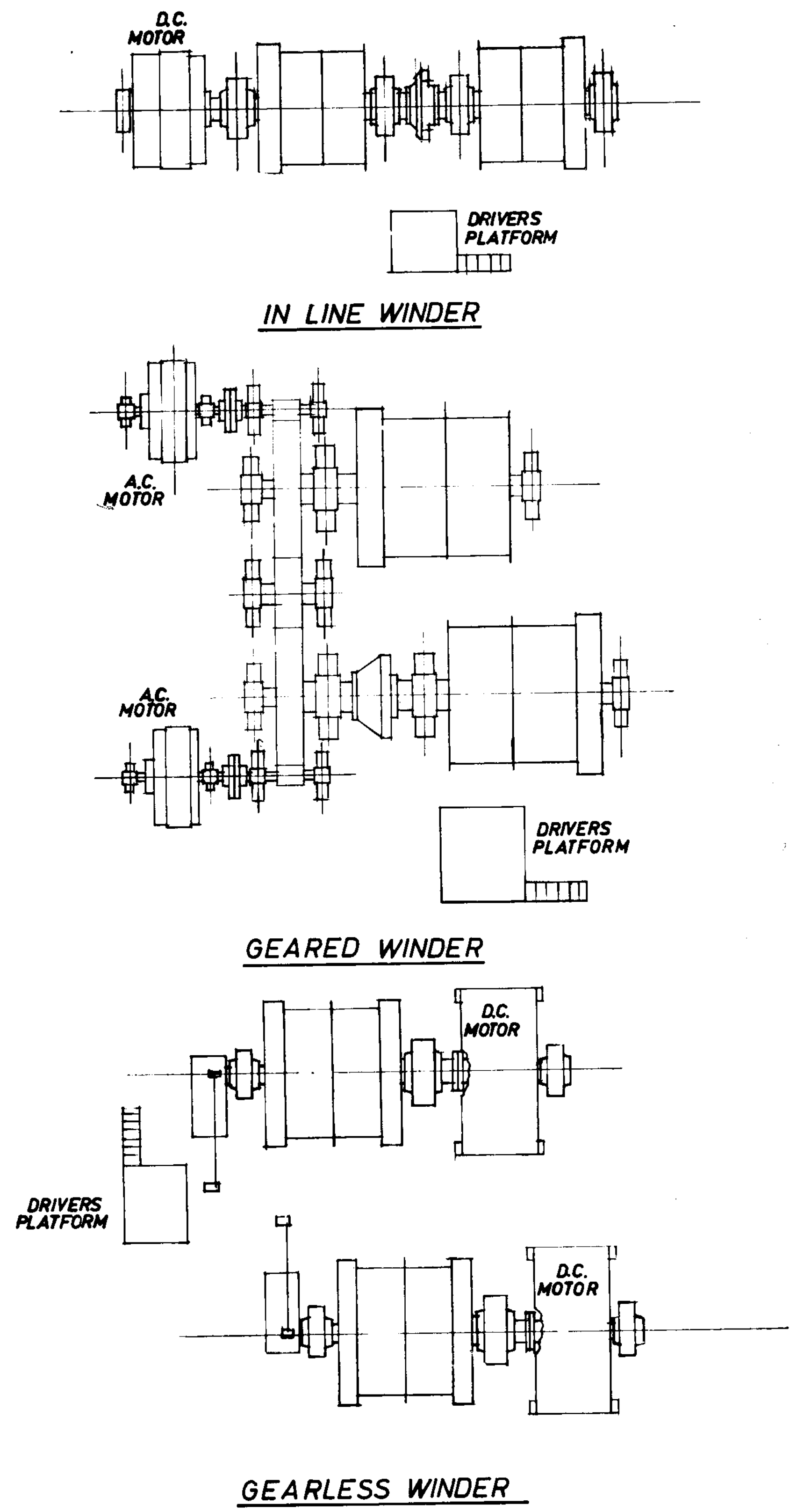


FIG. 4

THE DEPTH OF SHAFT

For many years the economic depth for hoisting has been accepted as about 1600 metres. From this point in depth, a sub-shaft was necessary to reach the next stage of similar depth, and finally even a tertiary shaft to mine to the limit in depth — at present estimated at some 3650 metres. This system necessitated the cutting of large underground hoist chambers, rope raises, and waste and ore transfers, which was both very expensive and time-consuming. As the demand for the hoisting of larger tonnages from greater depths increased, the double-drum winder with its single rope became uneconomic and it was necessary to provide a winding system using more than one rope.

At first the Koepe winder was considered a solution to the problem, mainly because it was the only known method of operating a hoisting system with multi-ropes to hoist heavy payloads. This system of hoisting has operated successfully but has certain disadvantages, such as the following.

(i) In a Koepe winder the stress pattern due to rope weight is reversed every winding cycle, and, with increased depth and rope weight, the rope life is reduced accordingly.

(ii) The installation of new or the changing of old ropes is a problem and requires an additional large winch.

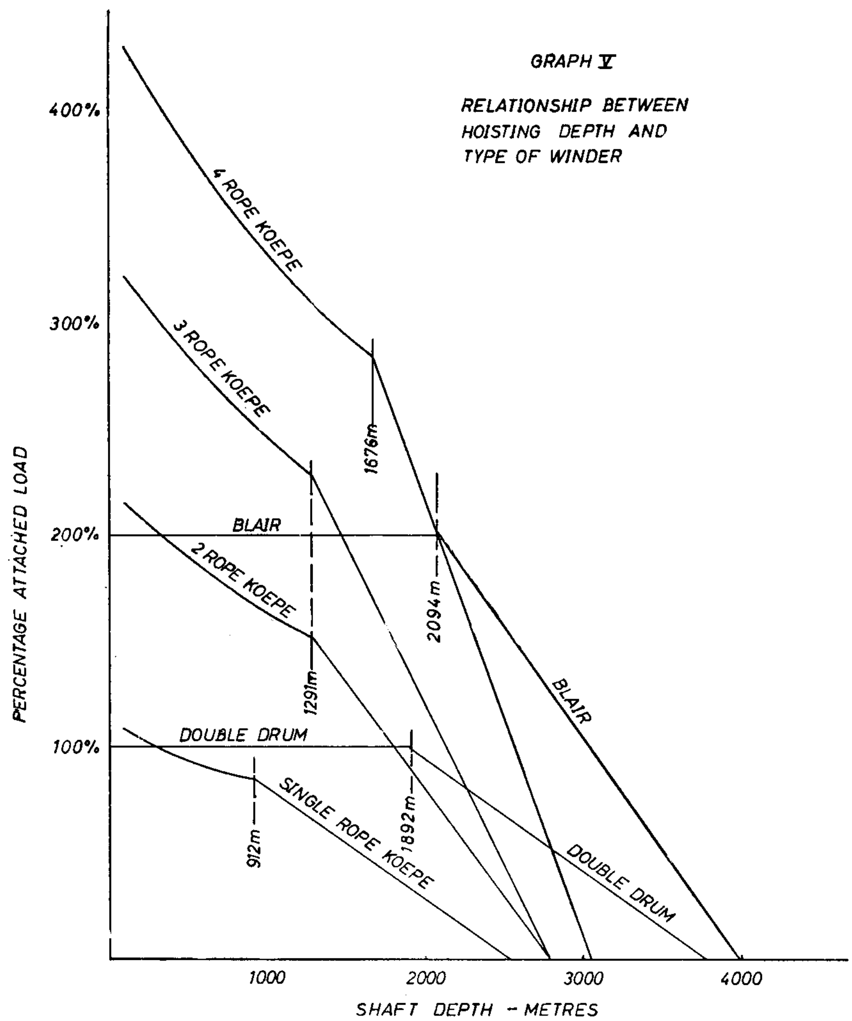
(iii) An adjustment has to be made at capel ends to compensate for rope stretch.

(iv) The cage must be made sufficiently large to cater for all the machinery and equipment to be lowered inside it because, owing to the balance ropes, it is virtually impossible for anything to be slung below the cage.

(v) Winding should take place from one loading point only.

(vi) The shaft has to be deeper to provide for the balance ropes and sheaves, if necessary, and the added cost of the sophisticated spillage arrangements.

Finally, the Blair twin-rope, drum-type winding system was developed as a solution to the problem of



winding from great depth. The attachment of two ropes to each conveyance means that the diameter of the ropes can be reduced, and, therefore, the diameter of the winder drums and sheaves. Considerable development has been carried out on this winder, and there are three variations in basic layout.

The In-Line Winder is the simplest and cheapest, but the relatively large spacing between the two sets of ropes can present design difficulties with fleeting angles when the winder is being positioned.

The Geared Winder does not present any positional problems. However, because the combination of the out-of-balance loads from both drums has to be transmitted through the gear drive, a gear design problem is created. Because generally only one clutch is provided, single-drum winding is not practicable in all cases.

The Gearless Winder, although more expensive, has no positional problems and requires no complicated gear train; there is no mechanical coupling of the two drum assemblies, and the out-of-balance load is transmitted electrically; declutching is simple and is performed electrically. An additional advantage of this scheme is that a failure on one drum assembly will not prevent single-drum hoisting to take place with a reduced tonnage output. As proposed by Tindall[3], the large direct-coupled, direct-current motors are more reliable than comparable high-speed alternating-current motors fitted to a gear train. (See Fig. 4.)

The regulations of the Mines and Works Act dictate the load to be attached to the winding ropes at different depths and consequently the winding system to be used. The rope life and tread wear on a Koepe winder operating at a depth below 1830 metres become serious considerations.

A theoretical comparison between a double-drum winder with a single rope, a Koepe winder using a single and multiple ropes, and a Blair twin-rope drum winder based on the present-day regulations is shown in Graph V. The indications are that the most suitable winder for any depth below 2094 metres is the Blair twin-rope drum winder. This winder has proved itself a most suitable installation for hoisting large tonnages from deep shafts.

SHAFT LININGS

In South Africa unreinforced-concrete linings of from 0,23-0,30 metre in thickness are generally accepted as sufficiently strong for virtually all rock formations. In certain cases where the rock formations are sufficiently strong, only concrete rings of 1 metre depth are used at, say, 15 feet intervals to support the steelwork as well as the shaft walls. Where, over short distances, the rock walls are suspect, the full lining is placed and thereafter the spaced rings are reverted to.

Under certain conditions, however, such as where salt water has to be penetrated, or where the shaft walls are very weak, different types of lining are required. As an example of this method the case of the Cleveland Potash Mine in North Yorkshire, United Kingdom, may be cited.

Two shafts are currently being sunk for Cleveland Potash Limited by the Mines Construction Consortium. These shafts, known as the rock and man shafts, are 91,5 metres apart and will be sunk to a final depth of 1133 metres, with a finished lined diameter of 5,5 metres. Between 633 metres and 913 metres, one of the major aquafiers in Great Britain, known as the Bunter Sandstone, with its saturated brine solution, representing a formidable obstacle to shaft sinking, is to be penetrated.

Conventional methods were used for the sinking of both shafts down to the Bunter Sandstone. The rock shaft is being sunk through the Bunter Sandstone inside a frozen

Photo A—Freeze chamber showing part of brine ring intake and return pipes (insulated) and some individual freeze pipes with ice formation.

ice wall with a temporary concrete lining and will be permanently lined with a double steel welded lining reinforced with high quality concrete. The steel-plate used for the shells of the lining varies in thickness from 16 to 48 mm to cater for the varying hydrostatic pressure. (See Photograph A, which depicts the freeze chamber that was constructed at a depth of 585 metres.)

The double steel lining will be installed in the shaft, on a concrete foundation, from a specially constructed stage equipped with automatic welding machines.

The man shaft is being sunk through the Bunter Sandstone and lined with cast-iron tubbing of varying thickness to cater for the hydraulic head. Two different grades of cast iron were required, and behind these is a back-fill to the shaft walls of high-quality concrete. Finally, cement back-grouting is applied between the tubbing, concrete, and shaft walls to seal off the remaining water. The groundwater is controlled by means of advanced and sophisticated methods of grouting techniques with chemicals and/or cement, which are required to meet the combination of high pressures of the brine solution and the low permeability of the Bunter Sandstone. (See Photograph B depicting the installed tubbing rings.)

Under certain conditions where soft water-bearing strata have to be penetrated and where ground movements are likely to occur when mining takes place, it has been found necessary to use a double steel-welded lining similar to that described previously and reinforced with the necessary thickness of high-quality concrete between the two shells. In addition to this, a lining of bitumen 0,13-0,15 metre in thickness is placed between the shaft walls and the outer steel lining. In this manner the ground movement may deform this softer lining before bringing full pressure to bear on the double steel-welded concrete-reinforced lining[4].

DEVELOPMENT IN THE UTILIZATION OF LASHING GEAR

The lashing gear for the previously mentioned Cleveland Potash Shafts had to be designed to fit onto the bottom deck of the stage of the 5,45-metre diameter shaft in which two 5,4 tonne kibbles 1,52 metres in diameter also operated; this left 1,25 metres clearance in which to fit a 20 h.p., 4,5 tonne capacity hoisting unit. The regulations in England also demanded that the hoisting unit itself is self-sustaining with its full load, independent of the braking system.

To comply with the above requirements, it was necessary to design a new lashing gear.

(a) The main driving motor was mounted vertically, driving a high-ratio worm gearbox on end and then through a train of gears driving the winch drum.
(b) The radial boom of the unit was made extensible, allowing the hoisting trolley to run out to any position selected for the particular shaft diameter being cleared. During construction, the excavated diameter of the shafts varied from 5,45 to 7,6 metres. The boom is moved in and out by twin hydraulic cylinders, and the action of the driver in returning his boom-extension lever to neutral automatically locks the boom in any of the desired positions.
(c) The grab unit was also designed to be used in the installation of the individual segments of cast-

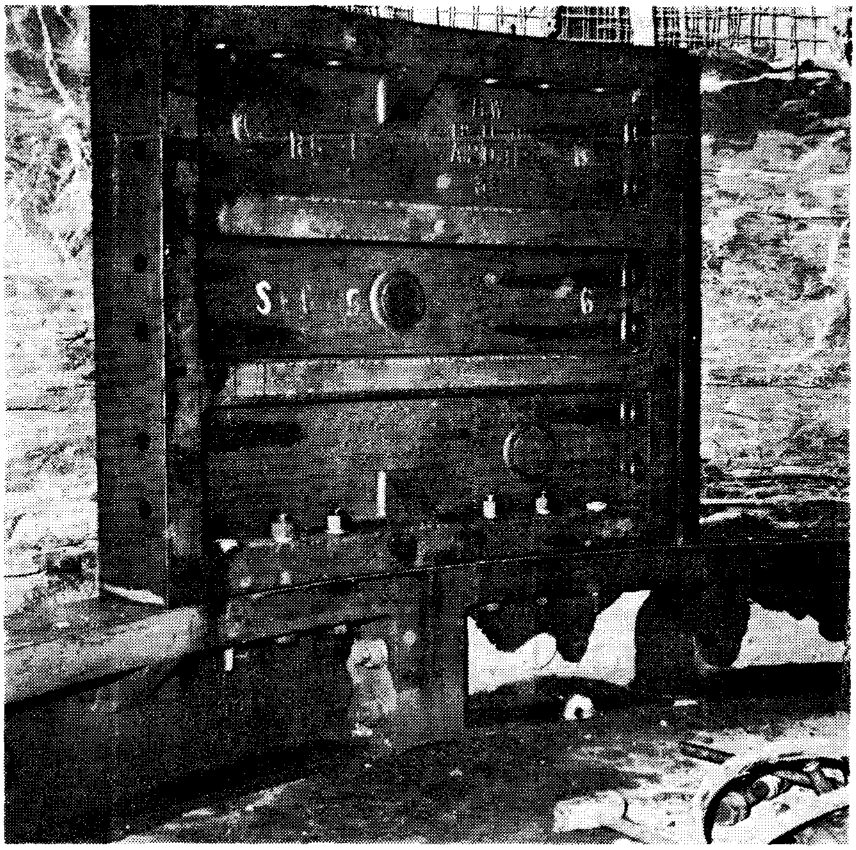


Photo B—Tubbing installation showing "tubbing platform" in foreground.

iron tubbing, each weighing approximately 6,25 tonnes. For this purpose it was necessary to have a more sensitive control than that required for normal shaft-lashing duties. In order to achieve this the following modifications were carried out.

The speed of the hoisting rope was reduced and controlled by adjusting the governor of the air motor driving the winch. A foot-operated pedal brake was installed in the driver's cab, the brake acting on the first motion shaft of the worm gearbox.

A complete new air-volume control valve was designed. This valve comprises three balanced-spool valves operated from a single control lever; the large-diameter valve controls the main supply of air to the hoist motor, which is linked to the driver's hand lever, and movement of the hand lever directly controls the volume of air being supplied to the motor and consequently its speed; also coupled to the hand lever are the two smaller auxiliary valves; one of these valves supplies pilot air to the reversing cylinder on the hoisting motor and the other to the main winch brake cylinder. The segments of cast-iron tubbing are lowered down the shaft suspended from the kibble rope. These are then transferred to the hoisting rope of the lashing gear and then finally positioned by means of the hydraulically controlled boom. (See Fig. 5.)

Extreme care must be exercised in placing the tubbing so that the lead sealing gaskets between adjoining sections of tubbing are not damaged.

SHAFT STEELWORK

Much work has been done to streamline the shaft steelwork so that the resistance to the airflow will be reduced. The flattened pipe section, 15 cm wide and spaced at 4, 6 or 8 metre intervals, is finding

TABLE XII Comparison of bunton sections

Type of bunton section	Unit cost percentage	Approximate shaft resistance
6″ Beam	100	100
6″ Mushroom I beam	120	64
6″ Covered I beam	150	45
6″ Aerofoil I beam	200	42
6″ Wide flattened pipe section	130	45
6″ Prismatic section	125	43
6″ Aerofoil section	+350	40
4″ Prismatic section	110	28
4″ Aerofoil section	300	25

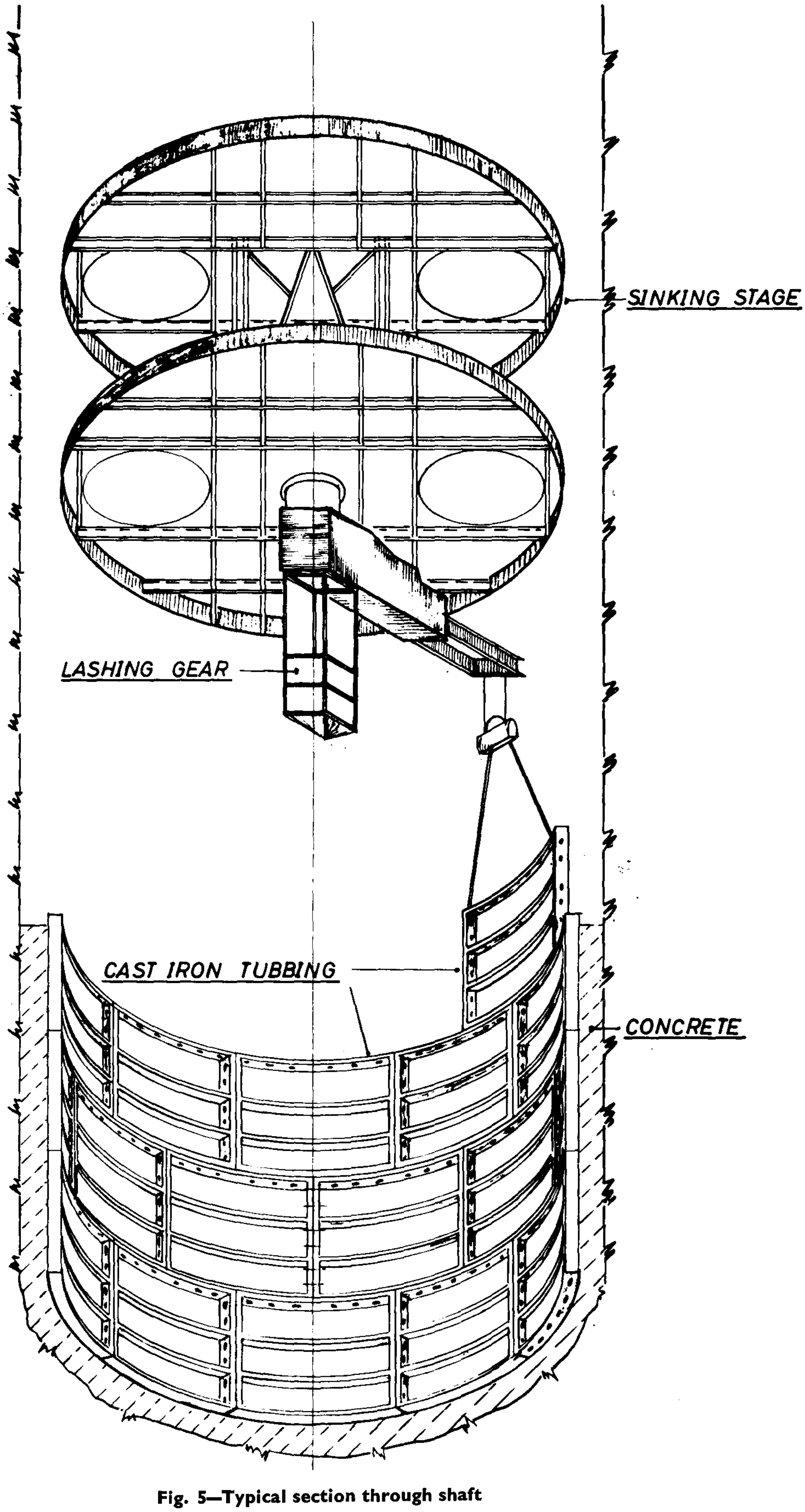


Fig. 5—Typical section through shaft

much favour at present.

Table XII, extracted from a paper by Van Wyk[5], shows the effect of various sections of shaft steelwork on resistance and also the related unit cost percentage.

CONCLUSION

It is not possible to deal with any but the most important aspects of shaft design within the scope of one paper without making it too lengthy. The authors realize that the aspects of fan-drift openings, ore-loading facilities, shaft spillage, and power, air, and water supplies, and various other matters have not been dealt with in this paper.

However, by utilization of the principles enumerated here, it is possible to design tailor-made shafts of minimum size, yet capable of supplying the ventilation requirements, having the desired output, and handling the labour and materials at a minimum capital outlay.

ACKNOWLEDGEMENTS

The authors wish to convey their appreciation to those South African mining and mechanical engineers who, through their determination and ingenuity, have contributed so much to the development of improved shaft design during the past three decades. They are also grateful to the Executive Committee of Shaft Sinkers (Proprietary) Limited and Mining and Engineering Technical Services (Proprietary) Limited for permission to publish this paper.

REFERENCES

1. MULLINS, A. R. and HAGGIE, I. S. Winding rope problems in South Africa. *Trans. 7th Commonwealth Min. Metall. Cong.*, Volume 111 (1961), 1235-6.
2. JAMIESON, D. M., PEARSE, M. P. and PLUMSTEAD, E. R. A. The evolution of shaft design and sinking technique in South Africa. *Trans. 7th Commonwealth Min. Metall. Cong.*, Volume 11 (1961), 591.
3. TINDALL, R. B. L. Some interesting South African developments in the design and practice of mine winders. *S.Afr. Instn. Mech. Engrs.*, July, 1968.
4. WEEHUIZEN, J. M. New shafts for the Dutch State Mines. *Trans. Instn. Min. Engrs. Symposium on Shaft Sinking and Tunnelling* (1959).
5. VAN WYK, C. F. B. A review of progress in the design of shaft equipment aimed at a reduction in shaft resistance. *Trans. S.Afr. Inst. Min. Metall., S.Afr. Instn. Mech. Engrs., Mine Ventilation Soc. S.Afr.—Symposium on mine shaft design and its effect on air flow*, (Nov. 1963).

AUTHORS' REPLY TO QUESTIONS BY Dr. J. T. Mc INTYRE

Maintenance costs for inclined shafts are about 11 cents per tonne hoisted, as opposed to about 5 cents per tonne hoisted for vertical shafts, Both costs apply to concrete-lined shafts with steel furnishings.

Graph V indicates the types of winders suitable for hoisting duties at various depths of shafts. It is clear that hoisting should be done from the maximum depth in a single lift, thereby avoiding the additional excavations required for sub-shafts, which consist of larger winder chambers, headgears, stations, access ways, and workshops, with the support steelwork and concrete, at a cost of between two and three million rand. Hoisting from greater depths depends on the availability of engineering design and manufacturing facilities for large-diameter ropes and hoist drums able to hoist at higher winding speeds.

THE SOUTH AFRICAN CHEMICAL INSTITUTE
DIE SUID-AFRIKAANSE CHEMIESE INSTITUUT

PRETORIA BRANCH

The Organization of Science

by

PROFESSOR SIR DEREK BARTON, F.R.S.

CHAIRMAN: Dr S. M. Naude, the Scientific Adviser to the Prime Minister.

Venue: Van Der Bijl Hall, Chancellor's Building, University of Pretoria.

DATE: Wednesday, 18th July, 1973 at 20 h 00.

Sir Derek Barton, the Nobel Laureate for Chemistry in 1969, is visiting the Republic at the invitation of the South African Chemical Institute. As President of the Chemical Society and having served on the Council for Scientific Policy of the United Kingdom, he is outstandingly qualified to speak on matters of scientific policy and organization.

PRETORIA-TAK

The Organization of Science

deur

PROFESSOR SIR DEREK BARTON, F.R.S.

VOORSITTER: Dr S. M. Naude, die Wetenskaplike Raadgewer van die Eerste Minister.

PLEK: Van Der Bijl-Saal, Kanseliersgebou, Universiteit van Pretoria.

DATUM: Woensdag, 18 Julie 1973 om 20 h 00.

Sir Derek Barton, die Nobel-pryswenner vir Chemie in 1969, besoek die Republiek op uitnodiging van die Suid-Afrikaanse Chemiese Instituut. Hy is die President van The Chemical Society en het reeds in die Council for Scientific Policy van die Verenigde Koninkryk gedien. Sir Derek Barton is dus by uitstek bevoeg om met gesag oor wetenskaplike beleid en organisasie te praat.