The Application of the Electrical Resistance Analogue to Mining Operations*

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SYNOPSIS

The theory of elasticity has been shown to provide a means of determining the stresses and displacements induced by mining in deep level hard rock mines. This theory has been incorporated into an electrical resistance analogue on which the plane of a reef or ore-body can be modelled to enable the rapid determination of stresses and displacements at points in the plane of the reef.

This electrical analogue has been used to determine energy release rates caused by scattered mining, which, to some extent, is a measure of the degree of hangingwall fracture. A correlation between predicted energy release rates and actual underground conditions in stopes in the O.F.S. goldfields has been encouraging enough to define parameters on which future stoping layouts can be based.

The use of the elastic theory through the analogue has been extended to the assessment of the influence of stoping operations on off-reef excavations. Underground conditions at damaged and undamaged areas have been compared with predicted elastic stresses at the positions under review, resulting in the derivation of design parameters for use in siting off-reef excavations in the prevailing geological conditions in the O.F.S. and Klerksdorp goldfields.

The analogue has also been used to assess the vulnerability of certain unmined dykes to bursting. Correlations between stresses induced on dykes by stoping and dyke behaviour underground has enabled critical stress levels to be determined for several persistent dykes occurring in the O.F.S. and Klerksdorp goldfields.

A number of long and short term planning projects at mines operating in the Anglo American Corporation and General Mining and Finance Groups are described, where the data derived from the electrical resistance analogue is processed by computer programmes and assessed on the strength of the design factors derived from the correlation of field work and the elastic theory.

CONTENTS

INTRODUCTION

The behaviour of a remnant, haulage or dyke is often a critical feature in the layout of a stoping programme. If advanced knowledge of the mode of damage to an excavation is available, remedial action can often be taken by altering the layout or installing extra support prior to fracture.

Until the advent of the electrical analogue, no method existed whereby the varying stress conditions to which an excavation will be subjected, could be predicted. Although it was possible to predict damage from experience, there were no means of establishing the degree of damage. The result was that fracture had already occurred before the support was installed.

The uses to which electrical analogues have been put on the mines of the Anglo American Corporation Group and the General Mining & Finance-Federale Mynbou Group in the Welkom and Klerksdorp districts respectively, are outlined in this paper, together with examples.

The electrical resistance analogue used in the Welkom area is the prototype developed by the Mining Research Laboratory of the Chamber of Mines and has been used to solve mining layout problems for the Anglo American Corporation in the O.F.S., and to a lesser extent, in the Klerksdorp area.

An electrical resistance analogue was constructed on Stilfontein Gold Mining Company, Ltd., between July, 1966, and April, 1967, at a cost of approximately R5 000 excluding labour. This analogue was constructed to serve the needs of Buffelsfontein and Stilfontein G.M. Companies and is almost an exact replica of the prototype constructed by the Mining Research Laboratory.

The availability of I.B.M. 360/40 digital computers in both areas has greatly increased the scope and amount of work that can be carried out on these analogues.

For convenience, this paper is presented in two parts, namely:

Part I: The development, operation and use of the electrical resistance analogue.

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Part II: The practical application of the electrical resistance analogue in the Klerksdorp and Orange Free State goldfields.

PART I: THE DEVELOPMENT, OPERATION AND USE OF THE ELECTRICAL RESISTANCE ANALOGUE

1. The development of the electrical resistance analogue

The theoretical elastic response of an idealized stope in a tabular orebody was compared with the movements measured in the vicinity of longwall excavations at depth[1] in 1964. This investigation indicated that the elastic theory could be used to describe, or predict within limits, the deformations induced in the unfractured rock mass by stoping.

At this stage, only stope configurations which approached simple idealized shapes (e.g. a parallel sided slot, circle) could be investigated, because of the complicated mathematics involved in obtaining analytical solutions for more complex configurations. This limitation of the elastic theory was overcome when Salamon[2] developed the face element principle. This technique enabled the theoretical elastic response for any stope configuration to be calculated, provided the elastic closure distribution in the stoped areas was known.

Although the closure distribution could be calculated using mathematical techniques, it was time-consuming. Nevertheless, a technique was now available for predicting in quantitative as well as qualitative terms, the rock behaviour as a result of stoping.

The next step in the development of the elastic theory was the utilization of the mathematical analogy that exists between the elastic laws and the equations governing the steady flow of electricity.

This analogy was exploited to develop the electrolytic tank analogue[3] which permits the determination of the closure and ride components in the plane of the reef for any given stope configuration. The elastic closure distribution obtained from the electrolytic tank analogue was used together with the face element principle to calculate the theoretical displacements at points in the rock mass, for comparison with measured displacements at these points in relation to complex mining configurations. This approach met with a high degree of success[4, 5] and for the first time the calculations could be performed with relative ease.

The electrical resistance analogue was subsequently developed by the Mining Research Laboratory of the Chamber of Mines[6]. This analogue embodies a number of features which renders it superior to the electrolytic tank analogue, the most important of which are:

1. The stresses normal to the reef plane can be measured directly.
2. The inclusion of boundary features enables scaling to be accomplished with the retention of outside influences.
3. Numerous configurations can be modelled in a relatively short time by merely inserting or removing plugs which represent solid ground.
4. There is no need to prepare special models for each stope configuration investigated.

2. Description and operation of the electrical resistance analogue

The electrical resistance analogue has been described in detail by Cook and Schumann[6]. The following is a brief description of the instrument.

The operating area of the analogue comprises a brass electrode or pinboard in which 1 500 holes are drilled (Fig. 1). A resistance network replaces the electrolyte of the electrolytic tank analogue and is situated behind the pinboard and arranged so that the nodes of the resistance network can be short- or open-circuited individually by plugs passing through the drill holes of the pinboard. Each node can be addressed by means of switches situated on a panel to the left of the pinboard (Fig. 1). The open-circuit voltage or closed-circuit current at each node is indicated by a digital voltmeter. Current is applied between the brass electrode and the opposite side of the network by a regulated D.C. power supply.

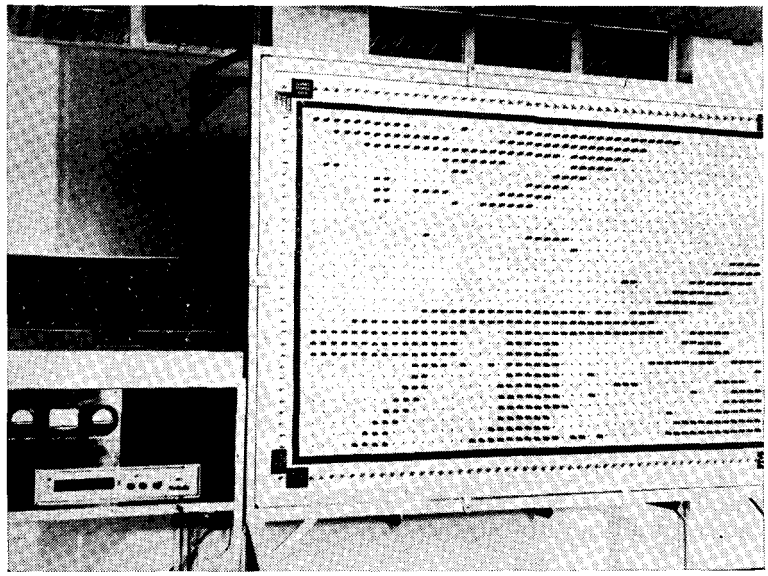

Fig. 1—Electrical resistance analogue

The pinboard represents the reef plane of the tabular orebody. The unmined area is represented by placing plugs in the drill holes, thereby short-circuiting the nodes, and the stoped-out area is represented by unplugged drill holes or open circuit nodes.

A 50 × 30 grid of squares is placed on a plan of the area requiring investigation. Each square of the grid represents a hole in the pinboard. The stoping configuration is then projected onto the pinboard by placing or removing plugs to conform with the solid or stoped areas indicated by the grid on the plan. Fig. 2 shows a typical stoping configuration with the grid in position. This configuration is shown in position on the analogue pinboard in Fig. 1.

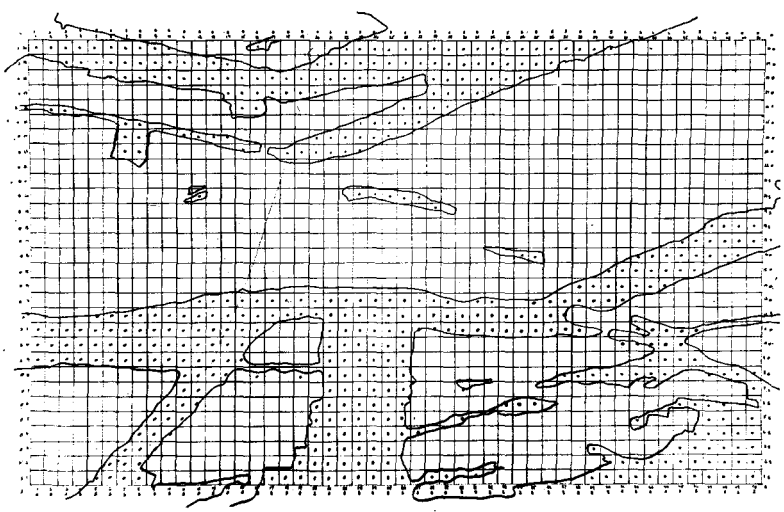

Fig. 2—Mine plan as projected onto analogue

If the reef plane has a dip of 10° or less, no adjustment is necessary to the square grid to allow for dip. In the event of a reef dip greater than 10°, the projection of each mine section onto the reef plane is achieved by using a grid in which the squares are distorted into rectangles corresponding to the plan projection of the reef plane squares.

3. The determination of stresses, displacements and energy release rates from the electrical resistance analogue

When using the electrical resistance analogue the current density at a node is proportional to the mean stress on the area represented by the plug. Similarly, the open-circuit voltage across a node is proportional to the elastic convergence in the mined-out area.

The use of the above measurements from the analogue for the determination of stresses and displacements at positions both off and on the reef plane, are given below:

3.1 *The determination of the stress normal to the reef plane.*

The ratio between the mean stress at an area represented by a plug and the primitive stress at the same position prior to nearby stoping is equivalent to the ratio between the current density at that node for the stoping configuration considered, and the current density at the node when all the plugs are in position in the analogue pinboard.

Thus, the mean stress at any point on the reef plane can be found by multiplying the ratio between the two current densities by the primitive stress (i.e. virgin rock stress).

e.g. Current density when pinboard is full of plugs = 0·0067 mA
Current density at the same position after stoping takes place nearby = 0·0201 mA
Depth below surface = 5 000 ft and the specific weight of quartzites = 169 lb/ft^3
The primitive stress at this position = 5850 lb/in^2 (approximately), and the new stress at the position under consideration

$$\frac{0{\cdot}0201}{0{\cdot}0067} \times 5850$$
$$= 17\,550 \text{ lb/in}^2.$$

3.2 *The calculation of the convergence in a mined-out area.*

Since the open-circuit voltage is proportional to the elastic convergence in the modelled stope, then the constant of proportion can be found by modelling a two-dimensional slot for which an analytical solution exists.

The ratio between the open-circuit voltage and the elastic convergence calculated at the point provides the factor by which the convergence in a stope configuration can be calculated.

3.3 *The determination of stresses and displacements at any point in the rock mass resulting from nearby stoping*[7].

Until recently, the only readily available method for determining the theoretical elastic displacements and stresses at any point due to a given stope configuration was by a numerical technique described by Salamon[2]. This method is based on a circular integrator and pretabulated factors of influence, which enables the calculation of displacement, strain and stress components from a known closure distribution (obtained from either the electrical resistance analogue or the electrolytic tank analogue).

Although the calculations are reduced to multiplications and summations which can be carried out as a routine operation without any special skills it is extremely time-consuming and laborious especially if displacement and stress components are required at several points.

In order to determine rapidly and relatively cheaply these values at a number of points in the rock for a particular stope configuration, a programme has been prepared for use with the I.B.M. 360 digital computers. This programme requires minimal input data from the analogue computer, and is used extensively in both the Klerksdorp and O.F.S. Rock Mechanics Departments. The determination of the stresses and displacements occurring at a number of positions in, say, a footwall haulage, is carried out in the following stages:

3.3.1 The preparation of the mine plan for use on the analogue.

3.3.2 The preparation of punching documents for input to the digital computer.

3.3.3 The measurement of the convergence distribution in the mined-out areas on the analogue.

3.3.4 The calculation of stresses and displacements on the digital computer.

3.3.1 *Preparation of mine plan data for the analogue.*

A mine plan showing the stoped-out areas, haulage positions and any other excavations under consideration, is required on a scale of 1:1000 or 1:2500. This plan should also show the following information:

(*a*) Faults, dykes and any major geological anomalies.

(*b*) Average dip of the reef over the area to be investigated, together with its direction (or a representative section on dip).

(*c*) The mine co-ordinates of survey pegs or reference points at positions within the haulage under investigation, with respect to datum.

In order to assist in the co-ordinate transformation from the mine co-ordinate system to a generalised system within the programme, it is necessary to orientate the 50 × 30 grid used for modelling the area under investigation on the analogue, so that the dip and strike of the reef coincide with two adjacent sides of the grid. The 50 × 30 grid is governed by the number of squares (pin positions) forming the plane of the reef on the working area of the analogue.

As far as possible the sections of the haulage or excavations under investigation should be located in the central area of the 50 × 30 grid. Once the section has been suitably orientated it is necessary to note the mining conditions outside of this rectangle so that the relevant boundary conditions can be simulated when the mine section is set up on the analogue.

3.3.2 *Preparation of the punching documents.*

The computer programme requires the following input data in order to determine elastic stresses and displacements at the points under investigation for the stope configuration modelled on the analogue.

(*a*) Rock properties, i.e. Young's Modulus, Poisson's ratio, specific weight of rock and the ratio of lateral to vertical stresses.

(*b*) The constant for the analogue in use.

(*c*) Data regarding the co-ordinates of the reference points under investigation, the average dip of reef, shaft collar elevation, the co-ordinates of a reference point for computing purposes, and the angle between the mine X axis and the strike of the reef.

(*d*) The voltage distribution obtained from the analogue, i.e. convergence in the mined-out area.

(*e*) The output information required, i.e. displacements, induced stresses, total stresses or principal stresses, caused by stoping.

Fig. 3 shows a typical set of punching instructions for a problem requiring all displacements and stresses to be calculated.

3.3.3 *The measurement of convergence distribution on the analogue.*

The section of a mine in which the problem is located is transferred to the analogue pinboard in the manner described above. Open-circuit potential readings are noted at the positions where pins have been left out, and the results are tabulated as shown in Fig. 4.

3.3.4 *The calculation of stresses and displacements from the analogue data by the digital computer.*

The data presented in Figs. 3 and 4, when prepared on punch cards and suitably assembled, are processed through a computer. The speed at which the arduous calculations are performed makes this an extremely useful programme and facilitates the examination of several points along a haulage.

Fig. 5 shows a print-out for an off-reef point investigated. The output data is given with respect to the mine co-ordinate system, that is, the generalized programme co-ordinates are re-transformed back to the mine co-ordinate system after processing.

The interpretation of the output data can be briefly explained as follows:

(*a*) *Displacements.*

The displacements U, V and W correspond to movements in the mine X, Y and Z co-ordinate axes respectively. (Z positive downwards). The notation $V = 0{\cdot}150\text{E} \ -02\text{ft}$ implies a movement of 0·0015 ft at the reference point in the mine co-ordinate Y direction. Likewise $Z = -0{\cdot}615\text{E}00$ ft means a movement of 0·615 ft upwards.

(*b*) *Induced stresses.*

The stresses induced by mining operations at the points considered are given by *Exx*, *Eyy* and *Ezz*, corresponding to the mine X, Y and Z co-ordinate axes respectively. Negative stresses are compressive and positive stresses are tensile.

The induced shear stresses in the planes XY, XZ and YZ are given under *Exy*, *Exz* and *Eyz* respectively.

(*c*) *Total stresses.*

The three total stress components at a point are obtained from the summation of the primitive stress and the induced stress. The virgin stress is calculated from the average depth below surface to the reef plane, and the average density of the rock.

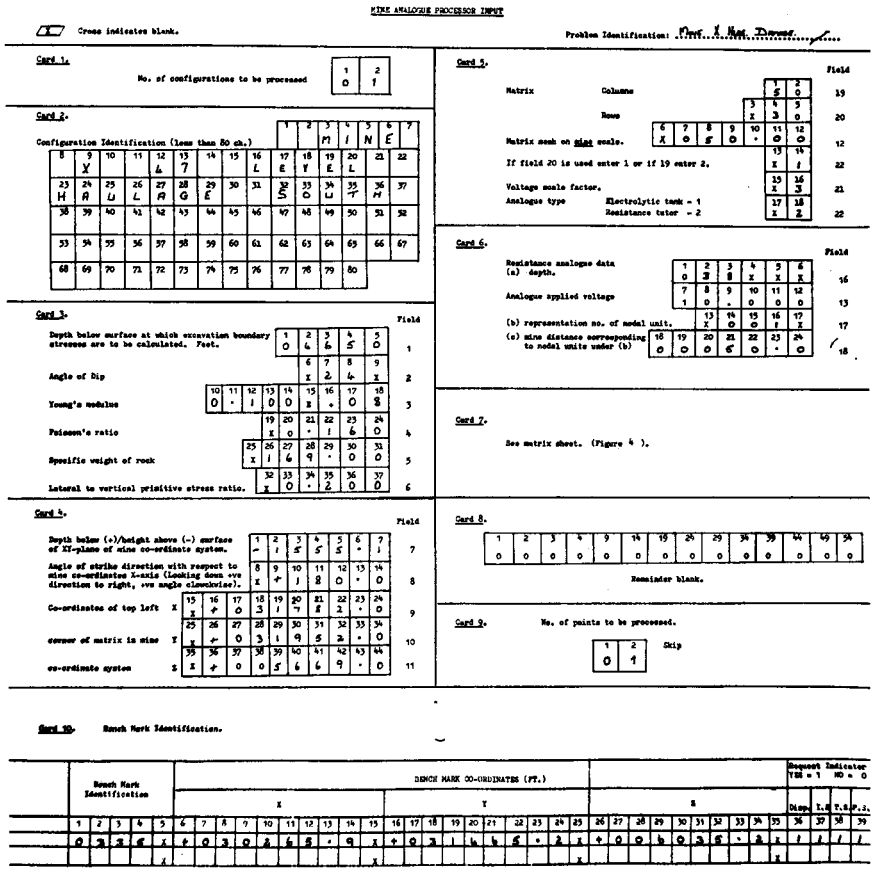

Fig. 3—Computer input data for determination of off-reef stresses

VOLTAGE MATRIX (x 10^{-3} Volts)

Column / Row	1	2	3	4	5	6	7	8	9	10	11 to 50 →
1	4720	4720	4710	4690	4660	4520	4500	4480	4480	4990	4520
2	4670	4650	4630	4580	4540	4390	4360	4330	4320	4330	4360
3	4630	4580	4530	4480	4410	4240	4180	4120	4100	4110	4160
4	4570	4500	4440	4360	4260	4060	3950	3830	3790	3810	3900
5	4490	4410	4330	4230	4090	3840	3620	3350	3270	3310	3520
6	4350	4300	4220	4100	3930	3570	3160	2450	2310	2390	2980
7	4260	4210	4120	3980	3770	3330	2510	0	0	0	2200
8	4200	4120	4030	3890	3700	3320	2860	2020	1490	0	1650
9	4130	4030	3930	3800	3640	3340	3050	2640	2260	1540	0
10	4030	3920	3830	3710	3580	3320	3140	2910	2720	2470	2260
11	3810	3790	3710	3600	3480	3240	3100	2980	2900	2830	2820
12	3700	3660	3490	3480	3340	3080	2920	2910	2930	2940	2970
13	3600	3530	3450	3330	3140	2760	2300	2640	2840	2920	2970
14	3490	3390	3310	3180	2930	2280	0	2180	2710	2860	2910
15	3310	3210	3150	3060	2910	2570	2140	2500	2710	2790	2810
16	2920	2950	2950	2920	2870	2700	2600	2650	2710	2710	2690
17	2640	2600	2690	2740	2770	2680	2660	2650	2640	2600	2530
18	2260	1980	2320	2530	2630	2600	2610	2590	2550	2460	2360
19	1463	0	1770	2260	2460	2480	2500	2480	2420	2280	2120
to 30 ↓											

Fig. 4—Input data for determination of off-reef elastic stresses and displacements

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                    KERNEL INTEGRATION ON MATRIX  I.
Mine X 47 Level Haulage South
        HO = 4650 FT, ALPHA = 24 DEGREES, E = 0.100E 08 P.S.I., V = 0.160 , GAMMA = 169.00 LBS/CU. FT, K = 0.200
        DEPTH B.S. OF MINE CO-ORDINATE SYSTEM -1555, 1 FT, STRIKE ANGLE 180.00 DEGREES
        MATRIX ORIGIN X =  31782.0 FT,  Y=  31952.0FT,  Z =  5669.0 FT
 BENCH MARK   0335        CO-ORDINATES  X (FT)          Y(FT)       Z(FT)
                                       30265.9        31465.2     6035.2
        DISPLACEMENTS       U = 0.3776E-03 FT,     V = 0.3563E-02 FT,     W = 0.1023E  00 FT
        INDUCED STRESSES (P.S.I.)      EXX        EYY        EZZ        EXY        EXZ        EYZ
                                       272.7      988.5     -2323.7     494.6      1398.4     1134.7
        TOTAL STRESSES (P.S.I.)        EXX        EYY        EZZ        EXY        EXZ        EYZ
                                       -778.9     -63.1     -7581.6     494.6      1398.4     1134.7
        PRINCIPAL STRESSES (P.S.I.)              DIRECTION COSINES                L          M          N
                -7989.5                                                          -0.1803    -0.1283    0.9752
                  538.0                                                           0.5282     0.8237    0.2061
                - 972.0                                                           0.8297    -0.5523    0.0807
                                   * * * * * * * * * * * * * * * * * * * * * * * * * * * *
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Fig. 5—Print-out of complete stress analysis at a point

(*d*) *Principal stresses.*

The three principal stresses at a point are given in pounds per square inch. Again, negative values signify compressive stresses and positive values represent tensile stresses. Normally, the orientation of the principal stresses is defined in terms of direction cosines with respect to the strike, dip and perpendicular directions of the reef plane. In the print-out shown in Fig. 5 the direction of the principal stresses is given in terms of the direction cosines L, M and N with respect to the mine X, Y and Z co-ordinate axes.

3.4 *The determination of energy release rates caused by the enlargement of stopes*[8].

An excavation in an elastic rock mass causes the rock above the excavation to move downwards under the influence of gravity, by an amount equal to the volumetric closure of the excavation, but no net movement of the rock below the excavation takes place. As a result of this a change in the gravitational potential energy occurs which is equal to the product of the overburden stress and volumetric closure of the excavation. Some of this gravitational energy appears in the form of strain energy in induced stress concentrations in the rock, and the remainder must be released in one form or another.

A portion of the released energy is converted, without violence, into heat and surface energy in the course of crushing rocks, but the remainder can appear violently in the form of kinetic energy. It is this latter form which manifests itself in the form of rock bursts and the former which fractures rock and causes difficult strata control conditions underground. The overall rate of energy release must be controlled to reduce the severity of rockbursts and difficult strata control in stopes.

It has been shown by Cook[9] that the energy released by mining operations, or by the complete fracture of any small pieces of reef, is equal to half the product of its area, the mean normal stress and the mean convergence between the hanging and footwall across that area when the reef is removed.

The latter two quantities can be found directly by the closed-circuit current and open-circuit potential measurements at the centre of the square representing the block of reef, on the electrical resistance analogue.

The use of the analogue and the results of field investigations conducted at the Chamber of Mines[10], has shown that the rate of energy release per unit area increase in the size of an excavation, is in fact a significant parameter for predicting the extent of rock failures, the incidence of stope damage and the problems of maintaining satisfactory strata control, arising from different mining layouts and depths.

It was concluded from their work that, with current mining systems and types of support used in gold mines, the problems arising from rock failures could only be kept within manageable proportions if the rate of energy release is less than 1×10^8 ft.lb per fathom mined.

The work involved in determining energy release rates for a mine is performed in three stages, namely:

(i) The preparation of sectional plans of the mine for modelling on the analogue.

(ii) The extraction of relevant data from the analogue.

(iii) The calculation of energy release rates in the areas to be mined.

(i) *The preparation of mine plans for the analogue.*

Experience has shown that a plan of the mine drawn to a scale of 1:2500 or 1:1000 and indicating the proposed stoping forecast at six-monthly intervals, together with faults, dykes, worked-out areas, estimated stoping widths and average depth below surface, is sufficient to enable an assessment to be made of the degree of hanging-wall fracture to be encountered over the period covered by the stoping forecast.

Although almost any scale can be used in modelling mining configurations on the analogue, a convenient scale for performing this type of analysis is one square on the analogue to represent an area of 50 ft $\times$ 50 ft in the plane of the reef. This caters for stope face advances achieved in most mines where quarterly face advances usually exceed 50 ft on panels of the order of 100 ft in length.

Using the above scale, the current workings of a mine are divided into sections of 2 500 ft $\times$ 1 500 ft in order to cover all proposed mining scheduled to take place over a convenient period of time (normally a two year forecast).

Once a mine has been sectionalized each section is traced onto grid paper containing 50 $\times$ 30 squares, where each square represents a 50 ft $\times$ 50 ft block of ground on one of the

scales mentioned earlier. Stoped-out areas, proposed sequences of mining and 'intact ground' are clearly marked on the grid paper and set out on the analogue for investigation.

For the purpose of identifying mining sequences in panels and stopes on a mine plan, a suitable coding system has been arranged. This eliminates ambiguities by defining each time sequence discretely.

Fig. 6 shows a section of a mine where the stoping sequence is given and the 50 × 30 grid is superimposed over the area under investigation, together with the boundary conditions around the perimeter of the section.

The calculation of energy release rates for stoping areas has been computerized and punch document sheets similar to that shown in Fig. 7 are prepared to cover all pins to be observed on each mine section for all periods of time under consideration.

(ii) *Performance of work on the electrical resistance analogue.*

Each mine section is set out on the analogue peg-board by inserting coloured pins into positions corresponding to the mining sequence code selected, e.g.

Green — First six months
Blue — Second six months
Red — Third six months, etc.

Where stoped-out areas occur pins are omitted and boundary conditions are set according to the relevant data. Closed-circuit currents are then recorded on all pins representing the first period in time (say, green pins).

After recording the current on the last green pin, all pins of that colour are removed—thus simulating that mining has taken place. Open-circuit potential readings are then noted at the positions where the green pins were located originally.

This procedure is repeated in chronological order for all time sequences under observation in each mine section.

(iii) *Calculation of energy release rates.*

A computer programme provides a print-out of the energy release rates for each period of time in each block of ground mined, according to the stoping forecast under investigation.

Fig. 8 shows a typical computer print-out, where the average energy release rates for each block are given in ft/lb per fathom mined. The energy release rate data, when marked on the stoping forecast plan modelled on the analogue, shows the changes in energy release rates as a result of stope enlargement according to the mining sequence provided.

PART II: THE PRACTICAL APPLICATION OF THE ELECTRICAL RESISTANCE ANALOGUE IN THE KLERKSDORP & O.F.S. GOLDFIELDS.

1. The correlation of data obtained from the analogue with underground observations of rock failure.

For some time now observations have been made in stopes, haulages and other off-reef excavations, in an effort to relate the degree of rock failure experienced in the mine with the predicted data obtained from the analogue. The adoption of this approach in both the Klerksdorp and Welkom areas has shown that:

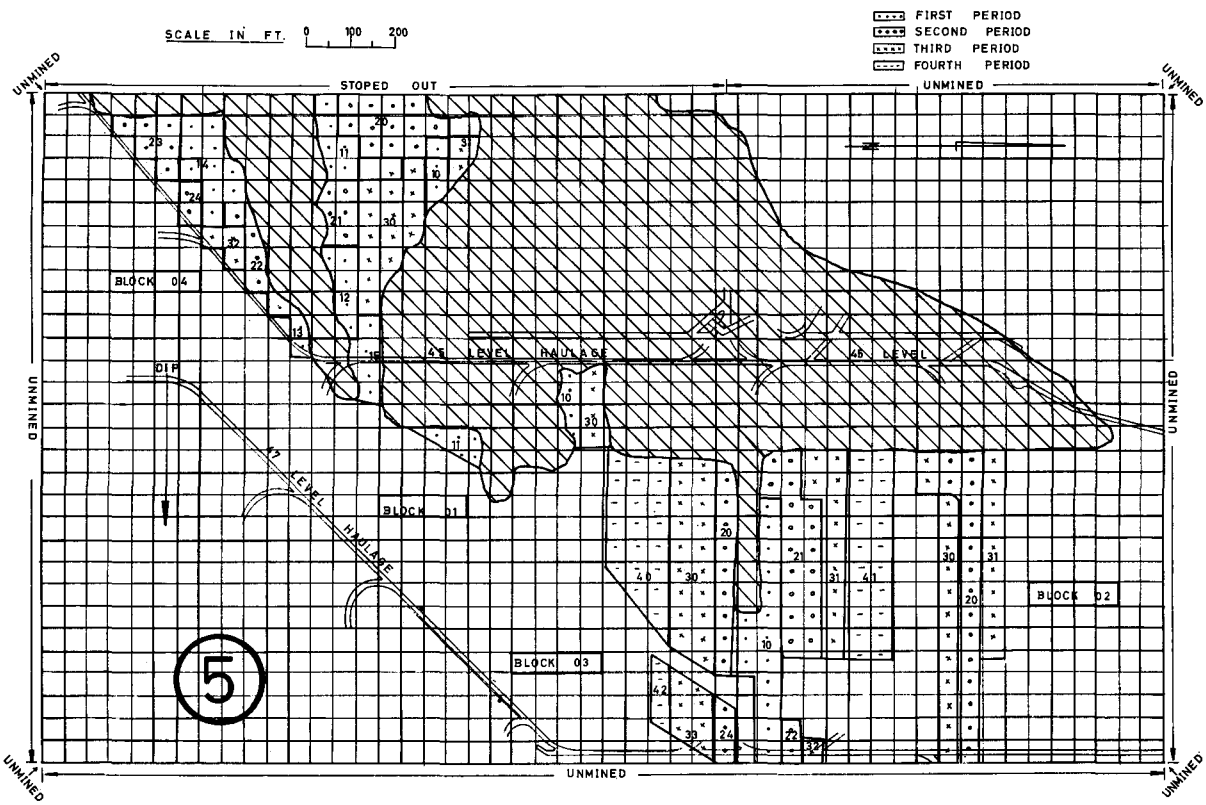


Fig. 6—Portion of mine prepared for energy release rate determinations. Section 5 of a mine showing detailed two-year forecast and 50 x 30 grid

ENERGY RELEASE RATE CALCULATIONS

MINE CODE		SHEET NO.		BLOCK		TIME PERIOD		CO-ORDS H		V		VOLTS					AMPS				DEPTH				
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
0	9	0	1	0	3	1	0	1	8	1	9	0	0	2	3	7	0	1	2	3	0	4	6	5	0
	←	Duplicate					→			2	0	0	0	2	6	3	0	1	0	3	←	Duplicate			→
										2	1	0	0	2	6	8	0	1	0	3					
										2	2	0	0	2	5	5	0	0	9	3					
										2	3	0	0	2	5	3	0	0	9	1					
										2	4	0	0	2	3	4	0	0	6	5					
										2	5	0	0	2	1	7	0	0	6	9					
										2	6	0	0	1	6	8	0	0	6	3					
								1	9	2	4	0	0	2	3	9	0	0	8	9					
	←	Duplicate					→			2	5	0	0	2	0	5	0	0	8	1					
				0	1	1	0	2	7	1	3	0	0	9	0	7	0	4	3	7	0	4	5	0	0
	←	Duplicate					→			1	4	0	0	7	4	7	0	2	4	9	←	Duplicate			→
										1	5	0	0	6	4	7	0	2	7	9					
						1	1	3	1	1	6	0	0	6	4	3	0	2	8	5					
	←	Duplicate					→	3	2	1	6	0	0	5	6	1	0	1	8	9					
				0	2	2	0	0	9	1	7	0	0	4	3	1	0	1	7	7	0	4	6	0	0
	←	Duplicate					→			1	8	0	0	3	1	2	0	0	8	7	←	Duplicate			→
										1	9	0	0	2	3	2	0	0	8	9					
										2	0	0	0	1	9	1	0	0	7	3					
										2	1	0	0	1	9	0	0	0	8	3					
										2	2	0	0	1	7	1	0	0	6	9					
										2	3	0	0	1	8	0	0	0	7	9					
										2	4	0	0	1	6	6	0	0	6	7					
										2	5	0	0	1	7	2	0	0	7	7					
										2	6	0	0	1	6	1	0	0	6	3					
								1	0	1	7	0	0	4	2	3	0	1	7	7					
	←	Duplicate					→			1	8	0	0	2	7	3	0	0	8	3					
				0	3	2	1	1	6	1	9	0	0	3	7	5	0	0	8	9	0	4	6	5	0
	←	Duplicate					→			2	0	0	0	4	3	0	0	0	7	7	←	Duplicate			→
										2	1	0	0	4	3	7	0	0	7	9					
										2	2	0	0	4	1	7	0	0	7	1					
										2	3	0	0	3	9	9	0	0	7	5					
										2	4	0	0	3	4	7	0	0	6	9					
										2	5	0	0	2	6	1	0	0	7	1					
								1	7	1	7	0	0	6	7	5	0	2	2	7					
	←	Duplicate					→			1	8	0	0	5	8	8	0	1	0	5					
										1	9	0	0	6	4	1	0	1	4	1					
										2	0	0	0	6	3	9	0	1	3	7					

Fig.—7 Computer input data for calculation of energy release rates

MINE NO.	SHEET	BLOCK	PERIOD	CO-ORDS	VOLTS	CURRENT	DEPTH	ENERGY RELEASE IN FT.LBS/50FT. SQ	
09	01	01	10	27-13	.907	437	4500	1,789,528,308	
09	01	01	10	27-14	.747	249	4500	973,087,980	
09	01	01	10	27-15	.647	279	4500	793,267,989	
							TOTAL IN FT.LBS/FATHOM MINED		17,068,246
09	01	01	11	31-16	.643	285	4500	958,711,622	
09	01	01	11	32-16	.561	189	4500	602,517,105	
							TOTAL IN FT.LBS/FATHOM MINED		11,240,848
09	01	02	20	09-17	.431	177	4600	359,907,603	
09	01	02	20	09-18	.312	87	4600	148,387,269	
09	01	02	20	09-19	.232	89	4600	96,096,953	
09	01	02	20	09-20	.191	73	4600	76,221,769	
09	01	02	20	09-21	.190	83	4600	73,394,467	
09	01	02	20	09-22	.171	69	4600	65,541,572	
09	01	02	20	09-23	.180	79	4600	67,087,264	
09	01	02	20	09-24	.166	67	4600	61,780,945	
09	01	02	20	09-25	.172	77	4600	63,350,496	
09	01	02	20	09-26	.161	65	4600	59,084,388	
09	01	02	20	10-17	.423	177	4600	409,294,990	
09	01	02	20	10-18	.273	83	4600	136,908,127	
							TOTAL IN FT.LBS/FATHOM MINED		1,940,467
09	01	03	10	18-19	.237	123	4650	162,842,025	
09	01	03	10	18-20	.263	103	4650	164,368,490	
09	01	03	10	18-21	.268	103	4650	156,687,343	
09	01	03	10	18-22	.255	93	4650	146,420,492	
09	01	03	10	18-23	.253	91	4650	130,684,419	
09	01	03	10	18-24	.234	65	4650	92,290,034	
09	01	03	10	18-25	.217	69	4650	83,641,510	
09	01	03	10	18-26	.168	63	4650	64,220,758	
09	01	03	10	19-24	.239	89	4650	118,823,118	
09	01	03	10	19-25	.205	81	4650	80,051,525	
							TOTAL IN FT.LBS/FATHOM MINED		1,728,043
09	01	03	21	16-19	.375	89	4650	186,437,947	
09	01	03	21	16-20	.430	77	4650	204,426,838	
09	01	03	21	16-21	.437	79	4650	195,961,351	
09	01	03	21	16-22	.417	71	4650	182,798,714	
09	01	03	21	16-23	.399	75	4650	169,861,931	
09	01	03	21	16-24	.347	69	4650	145,279,441	
09	01	03	21	16-25	.261	71	4650	105,186,681	
09	01	03	21	17-17	.675	227	4650	728,704,577	
09	01	03	21	17-18	.588	105	4650	344,889,254	
09	01	03	21	17-19	.641	141	4650	429,832,262	
							TOTAL IN FT.LBS/FATHOM MINED		3,878,466
TOTAL CARDS READ	37								

Fig. 8—Computer print-out of energy release rates

1. Critical field stresses can be defined for off-reef excavations situated in various stratigraphic horizons in both gold mining areas.
2. Certain magnitudes of energy release rates cause hanging-wall damage in stopes in the Orange Free State mines, and
3. Critical stress levels can be defined for dykes, provided data is available with regard to the strength of the dyke material and its degree of homogeneity.

1.1 *The development of a "criterion of damage" for haulages and other off-reef excavations.*

In both mining districts a common approach to the solution of problems involving tunnels of near square section has been adopted.

These problems include all layouts including tunnels, whether they are situated in a shaft pillar, loss of ground, or in the process of being overstoped. If two or more haulages are involved in an investigation the solution is only valid if there is no interaction of the field stresses around the haulages or other nearby excavations off the reef plane.

The stress distribution around a haulage is dependent on the shape of the haulage, the environmental field stresses and the elastic properties of the rock. Thus, provided haulages are similar in cross-section, then the degree of sidewall damage can be compared with the vertical component of the field stress for a particular rock type.

Although it is appreciated that the horizontal stress components have a bearing on the extent of damage to a haulage, experience obtained to date suggests that the criterion based on the magnitude of the vertical stress is sufficiently accurate for mining purposes.

In Fig. 9 the degree of sidewall damage for a number of haulages is plotted against the calculated vertical field stress component for a particular geological horizon in the Klerksdorp area. This illustration shows that there is a marked correlation between the degree of damage and the vertical field stress component. This has been repeated for the various geological horizons in which haulages are commonly situated.

The degree of damage permissable before support is installed is a matter of opinion. For this reason, a number of senior officials were questioned on site as to what conditions they considered required support. Invariably, the degree of damage designated 'moderate-severe' was thought to warrant additional support.

Similar work on seven mines in the Welkom area has shown that a field stress level of 8 000 lb/in.^{2} can be regarded as critical for tunnels situated in average O.F.S. footwall quartzites.

By using these and similar results it is possible to predict cheaply, quickly and accurately, not only the most likely areas of damage in a haulage, but also the extent or degree of damage. From this information steps can be taken to support a haulage prior to failure, or to alter the mining layout such that the damage is minimized. This information is invaluable, particularly where tunnels cannot be

overstoped due to losses of ground or unpayable ore blocks.

1.2 *The energy release rate concept as applied to stoping methods in the O.F.S. Goldfields.*

The encouraging results obtained from the energy release rate concept derived by the Mining Research Laboratory, led to the application of this technique for the assessment of the two year stoping forecast of all seven Anglo American Corporation mines operating in the Orange Free State. It was hoped that such an investigation would enable "trouble-spots" to be highlighted in time to allow alternative stoping sequences to be considered with a view to reducing the energy release rates to manageable proportions. Furthermore, it was envisaged that a correlation would be sought between actual mining conditions in stopes and the predicted energy release rates in each mine, and if possible, over the district as a whole.

To date, the comparison between predicted energy release rates and *in situ* conditions on all seven A.A.C. mines in the O.F.S. Division, has revealed that there is a remarkably high degree of correlation. Close examination of mining areas where energy release rates in excess of $0 \cdot 2 \times 10^8$ ft/lb per fathom mined were encountered has shown that the problems associated with hanging-wall control are manageable until a rate of $0 \cdot 4 \times 10^8$ ft/lb per fathom is reached. Above this rate conditions deteriorate rapidly and rates of energy release in excess of $0 \cdot 9 \times 10^8$ ft/lb per fathom mined have been known to cause sudden violent failure of reef pillars. The following alterations to stoping sequences usually assist in improving or avoiding poor hanging-wall conditions within stopes:

(*a*) Seek an alternative mining method or sequence of mining to reduce the rate of the 'energy release'. For example, swing faces to advance towards solid ground.

(*b*) If this is not possible, reconsider the overall stoping sequence in the area to ensure that the last blocks of ground to be mined are as near as possible to solid ground.

(*c*) In the latter case, spans between the working faces and the nearest line of support should be an absolute minimum.

(*d*) Some form of rapid bearing support should be used, for example, sandwich or composite packs.

(*e*) Stoping widths should be kept to a minimum.

1.3 *The assessment of the vulnerability of strong dykes to 'bumping'.*

Although rock-bursts as such are not encountered in the Welkom area of the O.F.S. goldfields, it is not uncommon to experience tremors on the surface and occasional 'bumps' underground in some mines operating in the area. Experience over the years has shown that such 'bumps' are often associated with the failure of hard, fine-grained dykes which have been left as solid ribs traversing sections of a mine, since their thickness and extent have been too excessive to justify mining.

The application of the analogue to an historical survey of dyke 'bump' problems in the A.A..C O.F.S. mines has revealed that, if the total stress on a dyke (induced stress plus primitive stress) exceeds the uniaxial compressive strength of the dyke material, 'bump' conditions occur.

PLOT OF VERTICAL COMPRESSIVE STRESS AGAINST RATED DEGREE OF DAMAGE

FOR A SINGLE HAULAGE - M.B. 6 GEOLOGICAL HORIZON.

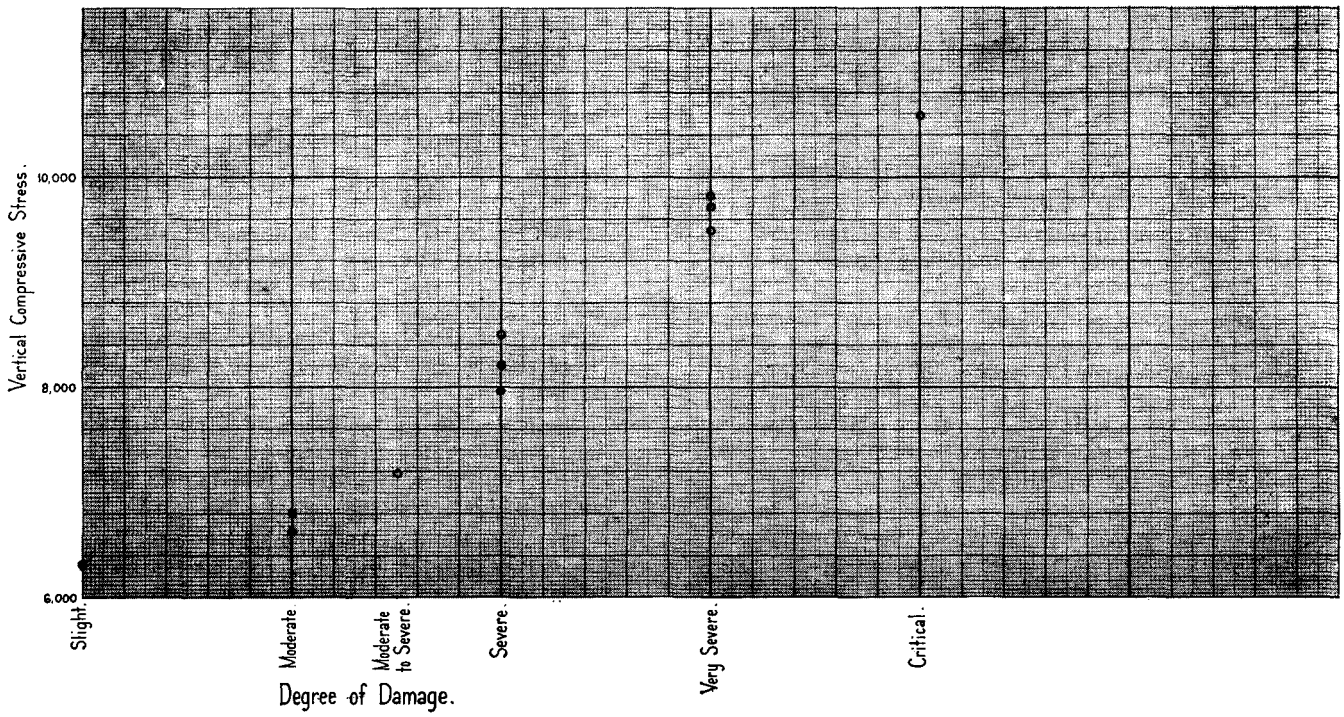


Fig. 9

A brief description of the application of the analogue for assessing the significance of an unmined dyke known to have 'bumped' is described:

The calculation of stresses on dykes:

Fig. 10 illustrates a portion of a mine in which stoping operations have stripped onto both sides of a dyke approximately 50 ft in thickness. The 46 and 48 level footwall haulages located 80 ft below reef were completely closed by a 'bump' emanating from the unmined dyke.

This section of the mine was modelled on the analogue in the usual way and 'plug-currents' were observed along the whole length of the dyke. In order to expedite the conversion of current values to stress values in pounds per square inch, a small programme was written for use on the Group digital computer. In addition, samples of the dyke were sent to the Council for Scientific and Industrial Research for uniaxial compressive strength tests. These were used for correlation with the predicted stress values obtained from the analogue. When considering the normal stress levels along the dyke this revealed that the uniaxial compressive strength of the dyke material was exceeded at the damaged positions underground.

The correlation described above has been substantiated on a number of "dyke-bump" problems in the O.F.S., and, this information has been used in reviewing several long and short term planning layouts.

2. Some solutions to problems encountered in mines of the Klerksdorp and Orange Free State goldfields.

The problems and solutions outlined in this section are typical of those encountered in the two mining areas and have been selected to provide a variety of case studies involving the use of the electrical resistance analogue.

2.1 *Klerksdorp Mining Area.*

2.1.1 *Mode of sidewall fracture in a haulage of near-square section and a method of support.*

Two experiments were conducted some time ago to establish the depth to which the sidewall of a haulage was fractured and to assess the effect of bolting and strapping on the retardation of rock fracture.

In both cases measuring pins were anchored at different depths in the sidewall of a haulage (Fig. 11) and the relative displacements between the end of the pins of equal depth were measured. In both cases the displacements were plotted on a time base.

In the case of the unbolted haulage (Fig. 12) the anchors of the pins close to the skin of the haulage moved away from those at depth. It is also evident from this graph that, as the sidewall moved into the haulage, the depth of the fracture zone increased. Points at which the fracture zone reached the anchors of the pins are shown on the graph. The strike haulage in question was being overstoped by a breast face.

A similar graph of a bolted and strapped sidewall under approximately constant stress conditions is shown in Fig. 13. In this case the haulage was severely fractured prior to the installation of the support. The retarding effect of the bolts on the movement of the skin is evident from the graph.

As a result of these experiments it is now standard practice to bolt and strap the sidewalls of all haulages which will be subjected to a high vertical stress.

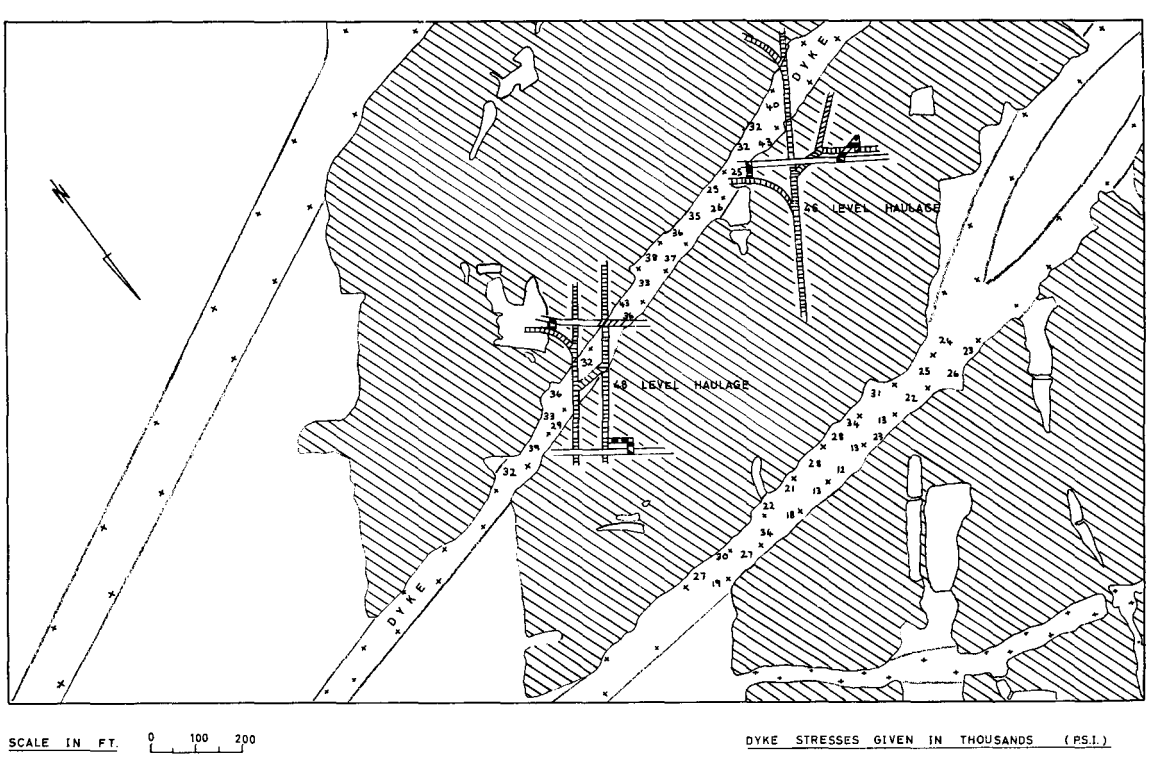


Fig. 10—Area of a mine experiencing bump on unmined dyke

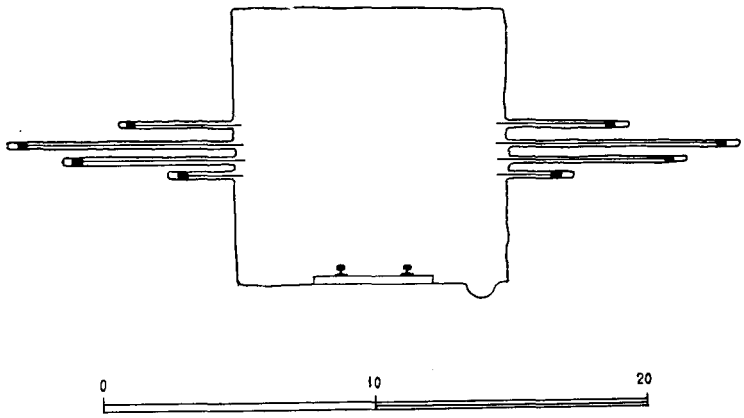


Fig. 11—General arrangement of measuring pins in a haulage sidewall

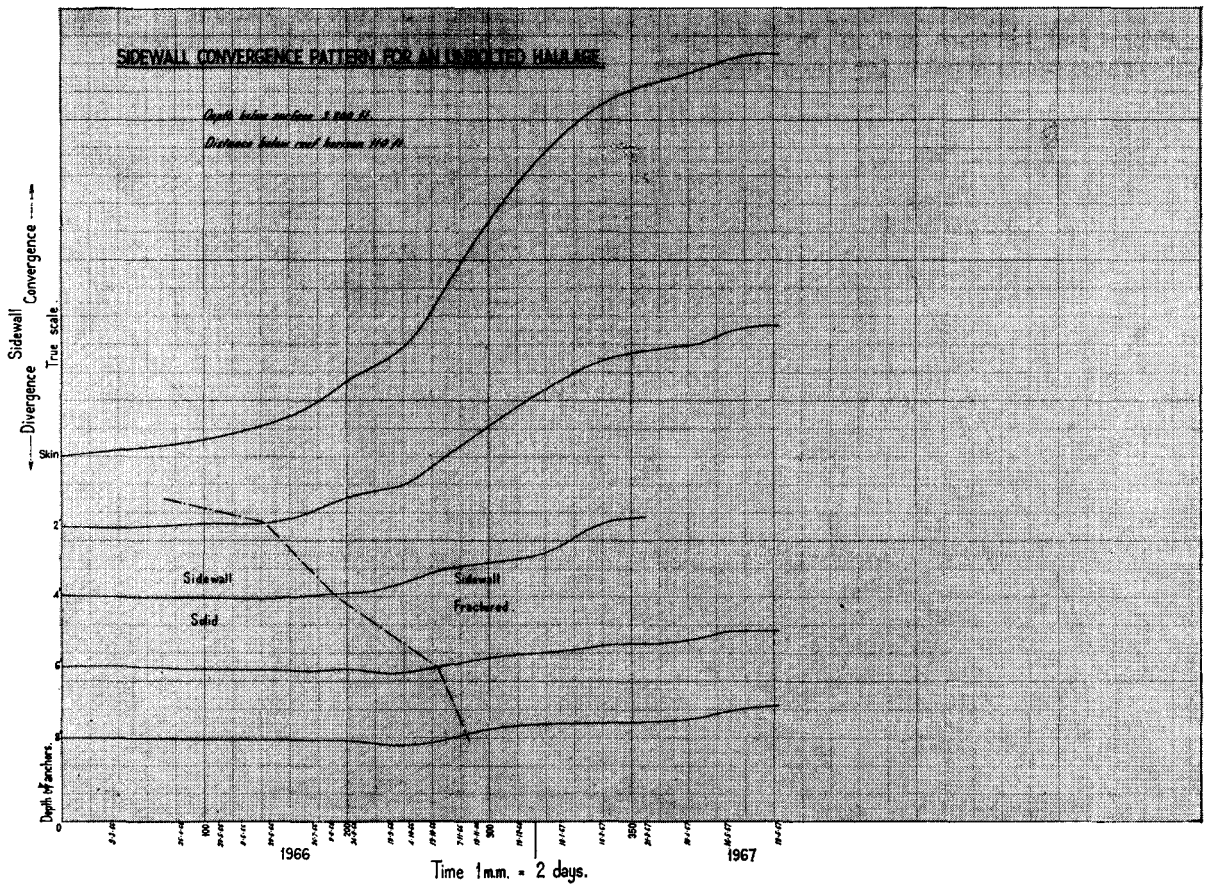


Fig. 12

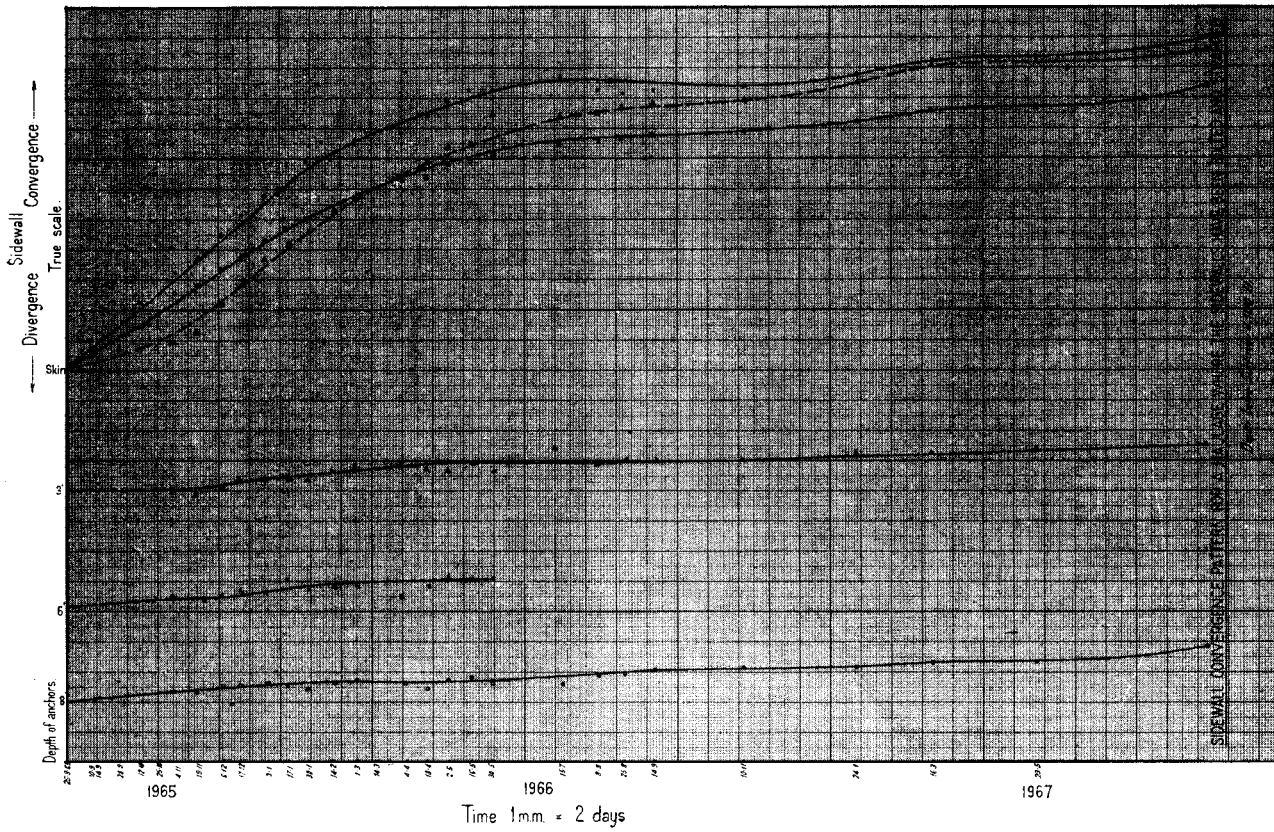


Fig. 13—Sidewall convergence pattern for a haulage where the sidewalls have been bolted and strapped

To date only one case of sidewall bolting failure has occurred. However, it is envisaged that as more evidence becomes available, it will be possible to establish an upper limit for the vertical stress, beyond which some form of yielding bolt must be used, or failing this, waste stoping will be required to reduce the stress level.

Fig. 14 shows the collapse of the unbolted sidewall of a haulage after a seismic event in the area. Fig. 15 shows the sidewall of the same haulage after a second seismic event of roughly the same intensity in this area. The haulage sidewalls were then secured during the interim period, with 80 in. under-gauge drill steel drilled through 2 in. × 8 gauge diamond mesh and grouted in position. The sidewall was too fractured for conventional bolting. The haulage in question is situated beneath a remnant and was exposed to a vertical field stress of approximately 13 000 lb/in^2 on both occasions. Note that the sidewall in Fig. 15 is severely fractured but retained by the mesh.

2.1.2 *Examples of the use of the vertical stress criterion to predict the behaviour of near-square tunnels.*

In order to illustrate the use of the analogue in predicting the behaviour of tunnels, two examples have been selected from practice:

(i) *The prediction of zones of damage in a haulage.*

Fig. 16 shows a footwall haulage (± 100 ft below reef) in the process of being overstoped by a number of faces related to three raise-winze connections. Three anticipated face positions are shown in the plan. These three configurations were modelled on the analogue and the vertical field stress components at a number of points on the haulage horizon were calculated by the digital computer for each of the three configurations. The calculated stresses are shown graphically in Fig. 16.

An examination was made of the rock type in the haulage to establish a vertical stress criterion for bolting, and the criteria established is shown on the graph in Fig. 16. Fig. 17 shows the condition of the unbolted sidewall at point A for face position 1.

A feature worthy of note is the position of maximum vertical stress at the haulage elevation. In this case it is situated towards the centre of the unstoped blocks and not immediately ahead of the face as is the case on the reef horizon.

From these results action can be taken to support the sidewall by bolting while the stress levels are still moderately low. With the early installation of the sidewall bolts these bolts have a reasonable chance of diminishing the extent of sidewall damage.

In this example the analogue has been used to predict the zones of damage in a haulage which will eventually be overstoped. Where the the haulage is situated in a loss of ground or below an unpayable reef block, it is essential to model the worst possible condition, i.e. with all the payable reef removed. An example of this is given below.

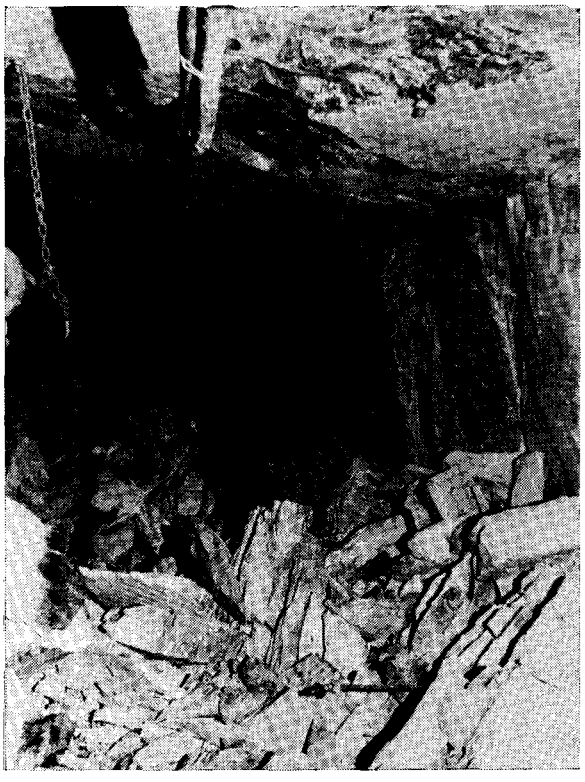


Fig. 14—Photograph of unbolted haulage

Fig. 15—Photograph of same haulage shown in Fig. 14 where support installed

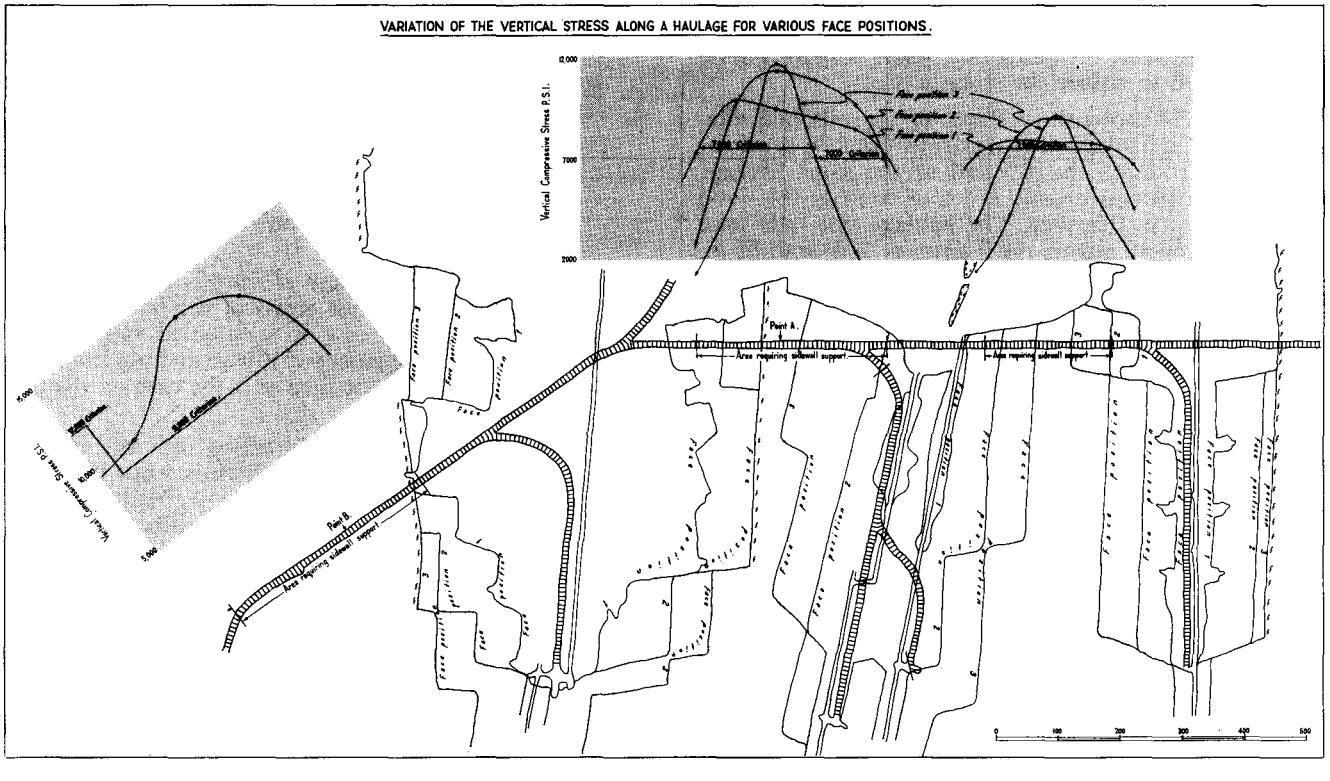


Fig. 16

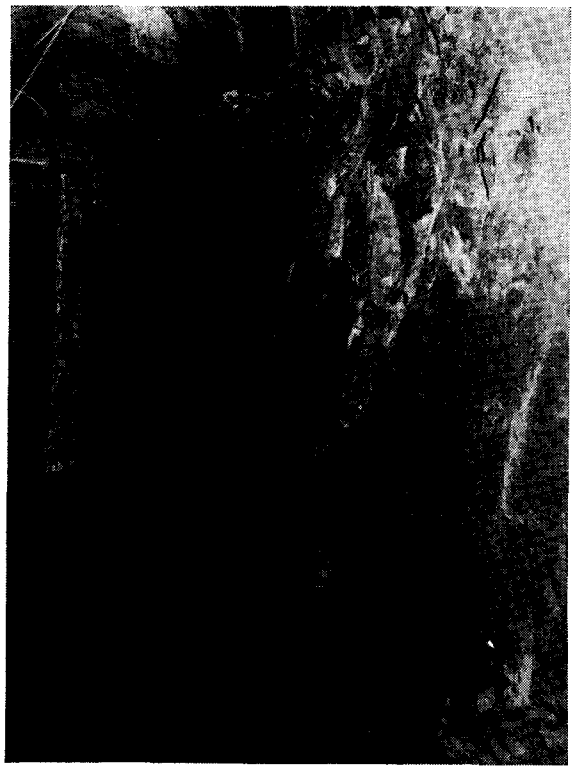


Fig. 17—Photograph of unbolted sidewall at position A shown in Fig. 16

To the left of the plan shown in Fig. 16 a haulage is situated in a large loss of ground. The variation of stress along the haualge as a result of stoping the reef to the right of the loss of ground is shown. In this case the stress level at which bolting is required is greater than in the positions described above.

This example illustrates how a problem of this nature is approached. If the stoping programme is flexible, a number of configurations may be investigated to establish the stoping programme which will require the least expenditure on support.

(ii) *The use of stope pillars to limit damage in a cross-cut below the stope.*

In this example use is made of the knowledge of the behaviour of haulages under stress to control cross-cut failure peculiar to some portions of the Klerksdorp mining district.

Failure of the cross-cut hanging wall, after it is overstoped, is a common occurrence on Stilfontein G.M. Co., Ltd. From the nature of failure of a large number of caps, it was evident that some form of sidewall movement was the cause of this damage. The sidewalls of the cross-cut were, however, relatively unfractured so that 'fretting' of the sidewall fracture zone as in the case of haulages under pressure, could be ruled out. In addition, the cross-cut was under reduced, and not increased, stress.

A small experiment was designed to provide information on the mechanism of the sidewall movement. Measuring pins were installed in the sidewall in the same manner as shown in Fig. 11. These stations were situated from 40 ft to 200 ft below the reef horizon. In addition, bench marks were installed on the 'solid' footwall of the cross-cut at each measuring station, and observed by precise levelling relative to a point approximately 200 ft in the reef footwall. The position of the raise relative to the cross-cut was similar to those shown in Fig. 16.

Measurements commenced during the ledging of the overlying raise. The convergence recorded was plotted on a time base and the results obtained from a station situated 120 ft below the reef horizon, as shown in Fig. 18.

It should be noted that, in this case, all the points start and stop converging simultaneously. The footwall uplift commences with the sidewall convergence and ceases when the convergence ceases.

The horizontal convergence can be explained as follows:

The concentration of phylosilicate minerals along the bedding planes in the area gives rise to a low coefficient of friction along the bedding planes. The rock in the undisturbed state is confined horizontally. As the vertical stress in the rock adjacent to the cross-cut decreases the frictional resistance to movement along the bedding plane also decreases, thus permitting the previously horizontally confined rock to expand into the cross-cut.

It is not possible to prevent this movement by bolting as the movements originate deeper than 8 ft into the sidewall and the forces involved are considerable.

If the vertical stress conditions around the cross-cut could be maintained at the level of the unmined state, it is possible that damage to the cross-cuts could be avoided. By superimposing pillars over the cross-cut an increased stress could be guaranteed and, provided the vertical stress level was not too high, sidewall damage similar to that already described could be avoided.

With this in mind, pillars 60 ft wide on strike were established on either side of the ledged raise and the remainder of the connection was stoped (Fig. 19).

Before stoping operations were carried out the stresses anticipated along the cross-cut were calculated. These calculations indicated that, if the whole area allocated to be stoped was extracted, it would be necessary to bolt and strap the sidewalls of a portion of the cross-cut. Subsequently the gold values dropped in a number of panels and stoping ceased prematurely. It was, therefore, not necessary to install additional support in the sidewalls, as the vertical stress had not reached a critical level.

Measuring stations similar to those mentioned before (Fig. 11) were installed in the cross-cut below the pillars. The convergence was plotted on a time base and is shown in Fig. 20.

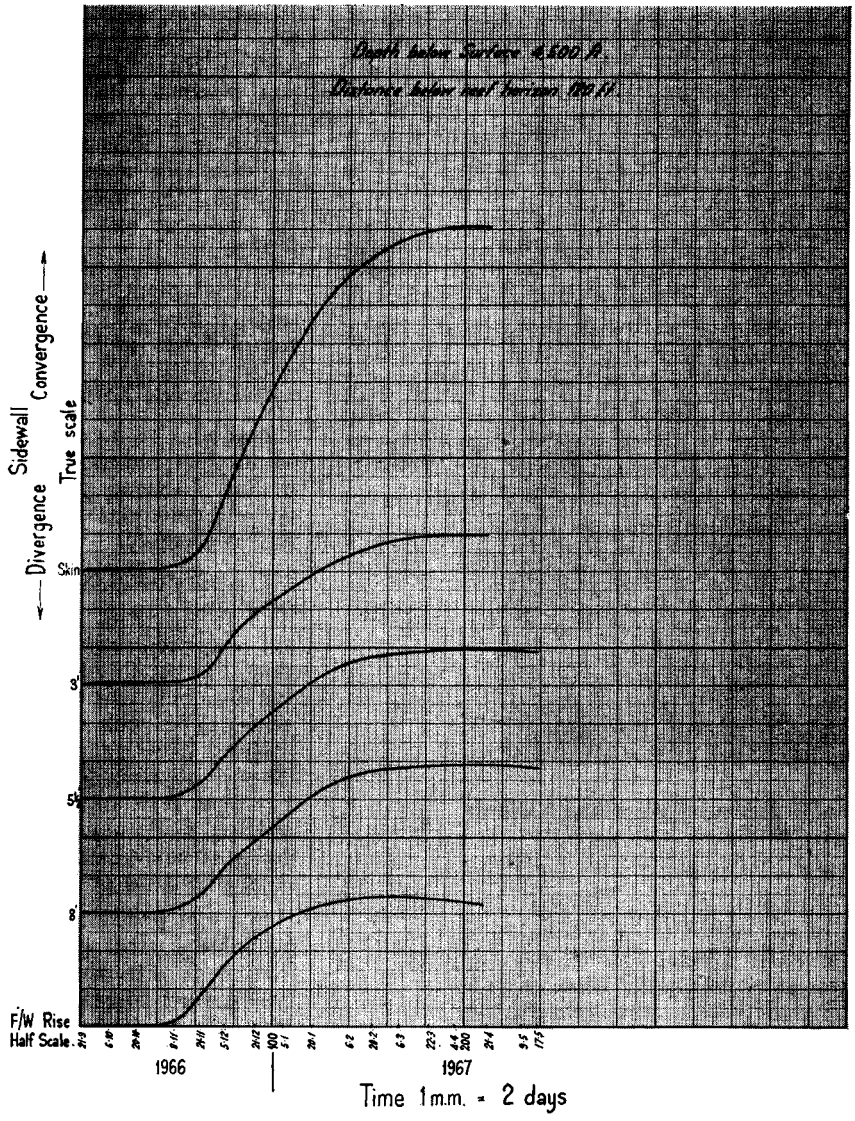


Fig. 18—Sidewall convergence for an overstoped crosscut in the Klerksdorp area

It was noted that only slight spalling of the sidewalls had occurred, which indicated that the cross-cut was experiencing a moderate vertical field stress only. There is no sign of convergence on the 6 ft and 8 ft measuring stations, thus indicating that conditions normal to overstoped cross-cuts were absent. General conditions in the cross-cut were excellent.

Without the aid of the electrical resistance analogue and knowledge of the critical stress level in haulages, this experiment would never have been attempted, as no method of predicting the degree of damage was available. In this example the analogue provided a means of utilizing a normally dangerous practice to advantage for a limited period of time.

2.1.3 *Approach adopted in planning the stoping of residual blocks.*

Because of faulting and unpayable reef blocks longwall stoping methods cannot be attempted in the Klerksdorp area. Consequently, raises are developed at strike intervals of approximately 500 ft. The extraction of the resultant residual blocks is planned by using the analogue with a view to maintaining average face conditions throughout the process of extraction.

Generally, the greater the vertical stress component, the greater the damage at the stope face. Therefore, uniform face conditions can be maintained if the vertical stress at the face remains unchanged. The greater the variation in the stress concentration factor, the greater the variation in conditions.

A grid, large enough to enable the surrounding solid blocks to be included, is superimposed on the plan of the area. This ensures that the influence of the surrounding blocks is taken into account. If stoping proceeds in the surrounding blocks the progress of stoping in these blocks must also be taken into account by removing pins to represent the face advance occurring during the removal of the residual block. The relevant portion of the grid is shown in Fig. 21.

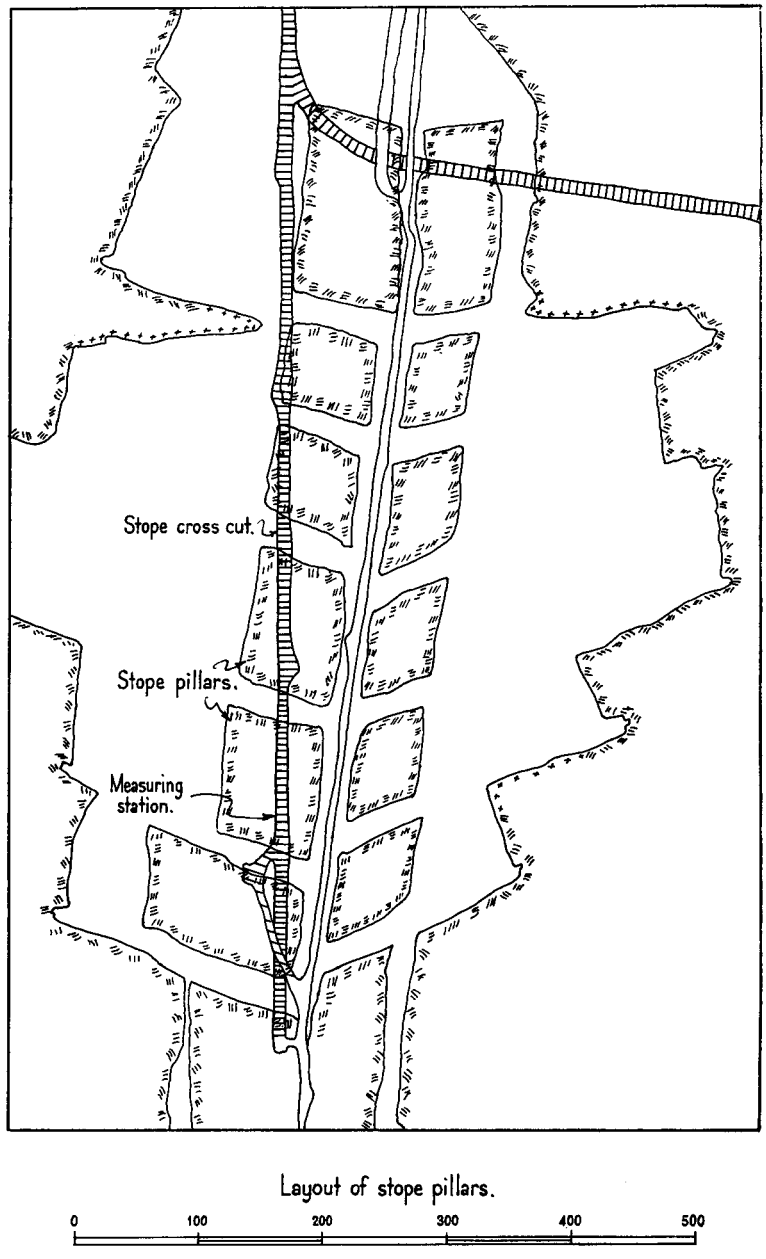


Fig. 19—Location of reef pillars over crosscut

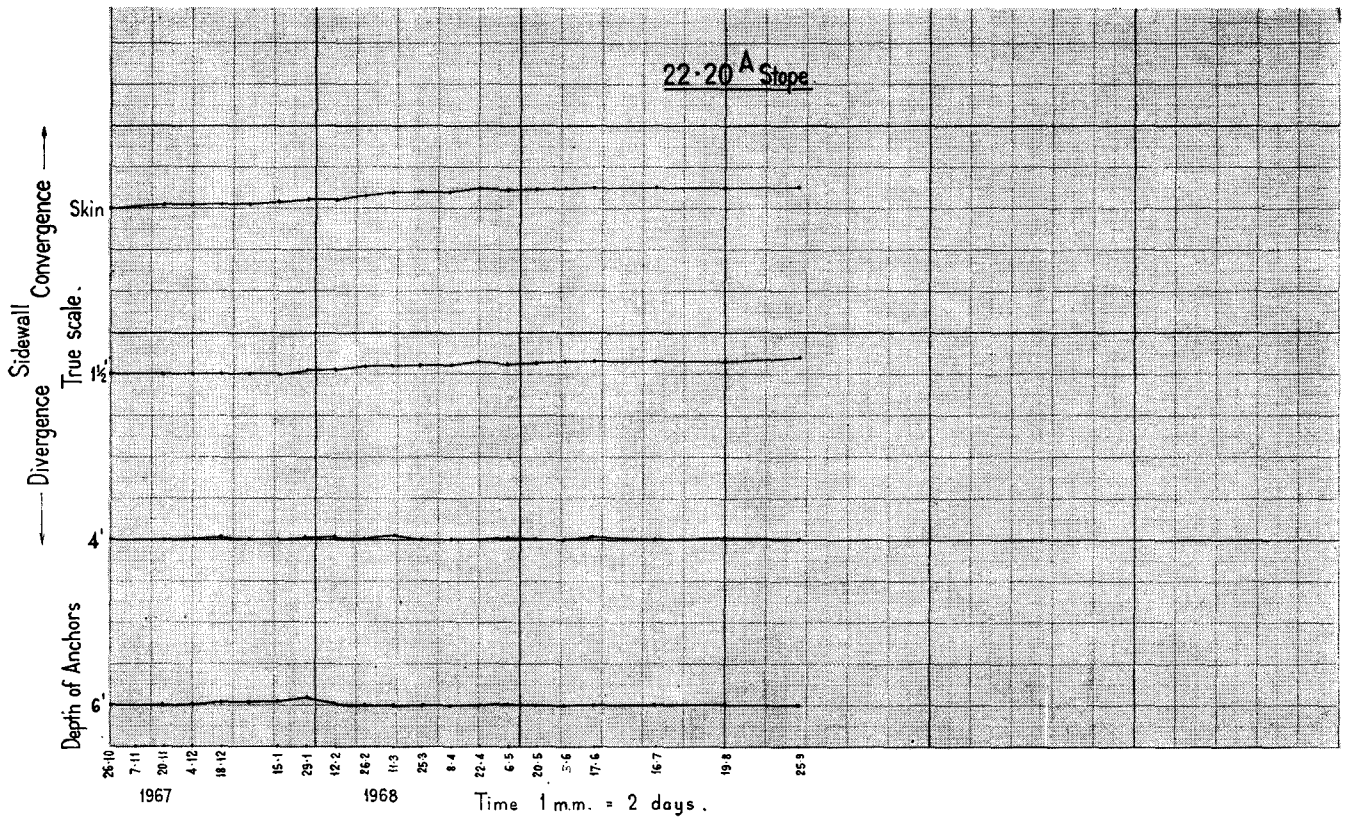


Fig. 20—Sidewall convergence for crosscut beneath pillars shown in Fig. 19

Fig. 21—The extraction of residual reef pillars and the resulting stress concentration factors

The stress concentration factors at the pins representing the edges of the residual block to be stoped are measured. The pin representing the block with the highest stress concentration factor is then removed, the position numbered on the grid, and the stress concentration factor recorded (Fig. 21 (*a*)). This process is repeated until all the pins representing the residual block have been removed.

By examining the sequence of removing the pins the general direction of stoping for the residual block can be ascertained. Clearly, it is not possible to adhere entirely to the ideal situation in practice, but a general pattern of stoping can be applied.

Fig. 21 (*a*) shows the ideal sequence of extracting the residual pillars in a stope. From the numbering it is clear that the pillar marked '1' must be stoped first and that the best direction of stoping is up-dip. The order of extraction for the remaining pillars is shown, together with the general direction of extraction. Note that the stress concentration factor for the last pin is not much greater than that of the first pin removed (8·3 as against 7·3).

For comparison purposes, two other possible stoping sequences were selected and modelled on the analogue (Figs. 21(*b*) and 21(*c*)). Bar charts were drawn of the frequency distribution of the stress concentration factors for the three stoping sequences and are shown above the plans in Fig. 21. From these two charts it will be seen that the spread of points about the mean stress concentration factor is far less for the recommended method of extraction than for the other two methods selected at random.

2.1.4 *Location of possible pressure-burst positions in dykes, and steps taken to avoid these.*

A number of pressure bursts have occurred in dykes on the reef horizon. These dykes have been petrographically typed and tested for uniaxial compressive strength, using diamond drill cores. The results obtained are as follows:

Dyke Type	*Uniaxial Compressive strength*
Porphyritic Diabase (grey-greenish in colour)	22 700 (10 800 — 45 000)
Epidiorite (green in colour)	49 000 (40 600 — 62 800)

Experience has shown that the epidiorite dykes are the chief source of pressure bursts on the reef horizon. In a number of cases the site of the pressure-burst occurrence has been modelled on the analogue and the vertical stress component calculated at the position of the burst. The vertical stress at these points consistently fell between 39 000 and 41 000 lb/in.^{2}.

If a policy is adopted to stope out all dykes on a property, care must be taken to lay out the stoping configuration to enable these dykes to be stoped at a low stress. In this case the analogue may be used to study the proposed stoping programme to ensure that the stress level remains below the critical value. The method of analysis is then similar to that described in the previous example, except that conditions on the reef face are sacrificed for the sake of low stresses on the dyke face. This method has been used in the past with considerable success.

If dykes remain unmined, then steps must be taken to ensure that the stresses in the dyke remain below the critical value.

Holings through burst-prone dykes on the reef horizon are avoided because of the high stress concentration factors adjacent to such holings.

It is common knowledge that a pillar in the reef plane has a higher stress concentration factor at the edges than at the centre of the pillar. If a pillar edge is comprised of reef, and the centre is of dyke material, the high stresses would be concentrated in the non-bursting reef portion, whereas the dyke core would be at a lower stress. The reef would also provide a lateral confining force which would strengthen the core.

In the Klerksdorp area the stress on dykes seldom reaches a very high level because of the added support provided by large losses of ground and unpayable ore blocks. However, steps have been taken to avoid the possible accumulation of high stress concentrations in the dykes, by leaving solid reef barriers along the dyke edges where they exceed the critical value.

The dyke under investigation is modelled on the haulage under the assumption that all the payable ore is stoped. The edges of the dyke are probed to establish where the stresses exceed 40 000 lb/in.^{2} When these positions are located a reef barrier 20 ft wide is simulated along the edge of the highly stressed portion by inserting pins into the pinboard. The position of the anti-burst barrier is then transferred to the plans.

To date, four dykes have been treated in this manner. Stoping in one of these areas has been completed without a burst occurring in the dyke, which would otherwise have attained a stress level of 56 000 lb/in^2. A typical layout involving anti-burst reef barriers is shown in Fig. 22. The zone of possible pressure bursting is shown on the plan if the anti-burst barriers were not installed.

2.1.5 *Use of the analogue and elastic theory to predict displacements in a vertical shaft and the effect of these displacements on the guides.*

As stoping proceeds around a shaft pillar the vertical stress in the pillar increases for a considerable distance above and below the reef plane. This increase in vertical stress is accompanied by a vertical shortening in the rock. If the buntons are connected to the rock (as is usual in the case of monolithic concrete-lined shafts) these buntons will move in the same direction as the rock to which they are attached. As the vertical dimensions of the rock tend to shorten so must the vertical distance between the buntons shorten.

If there is insufficient space in the slotted holes to permit this relative movement between the guides and the buntons, the guides will be compressed between adjacent buntons. If this compression is large the guides will buckle.

Recently, stoping commenced at the edge of the Toni shaft pillar (Stilfontein G.M. Co., Ltd.) to remove some blocks of ore which were difficult to stope due to faulting. Shortly after stoping commenced, buckling of guides occurred some 25 ft

above 13 station. A short time later buckling occurred 55 ft above 15 station. Although some damage was expected in the station areas due to this stoping, buckling of the guides was totally unexpected.

The question arose as to whether or not stoping should cease at the edge of the pillar, until the inner pillar had been removed.

In order to establish the magnitude of the differential displacements in the shaft which caused the guides to buckle, the vertical displacement between points at 25 ft intervals along the shaft was computed. Four different stoping configurations were considered.

Fig. 23 shows an east-west section through the shaft pillar area. It will be noted that, due to faulting, there are roughly two stoping horizons; on 13 level where reef is intersected in the shaft and ± 14½ level where most stoping has taken place.

In order to compute the induced displacements along the shaft, each of the four configurations studied were first processed in the usual manner on the analogue. The displacements induced by the two different stoping horizons were computed separately and then summed algebraically for each configuration.

From the calculated elastic displacements in the shaft the differential displacements between adjacent points 25 ft apart were derived and plotted in Fig. 24.

The minimum differential displacement between adjacent buntons necessary to cause buckling of the guides was calculated for the two extremes of end loading, taking into account eccentric loading. These values are 0·00145 ft and 0·002546 ft respectively for the 12½ ft bunton, or twice as much for the 25 ft, corresponding to the points calculated.

It will be noted that the minimum value is exceeded in the vicinity of 13 and 15 levels, the position where buckling of the guides occurred.

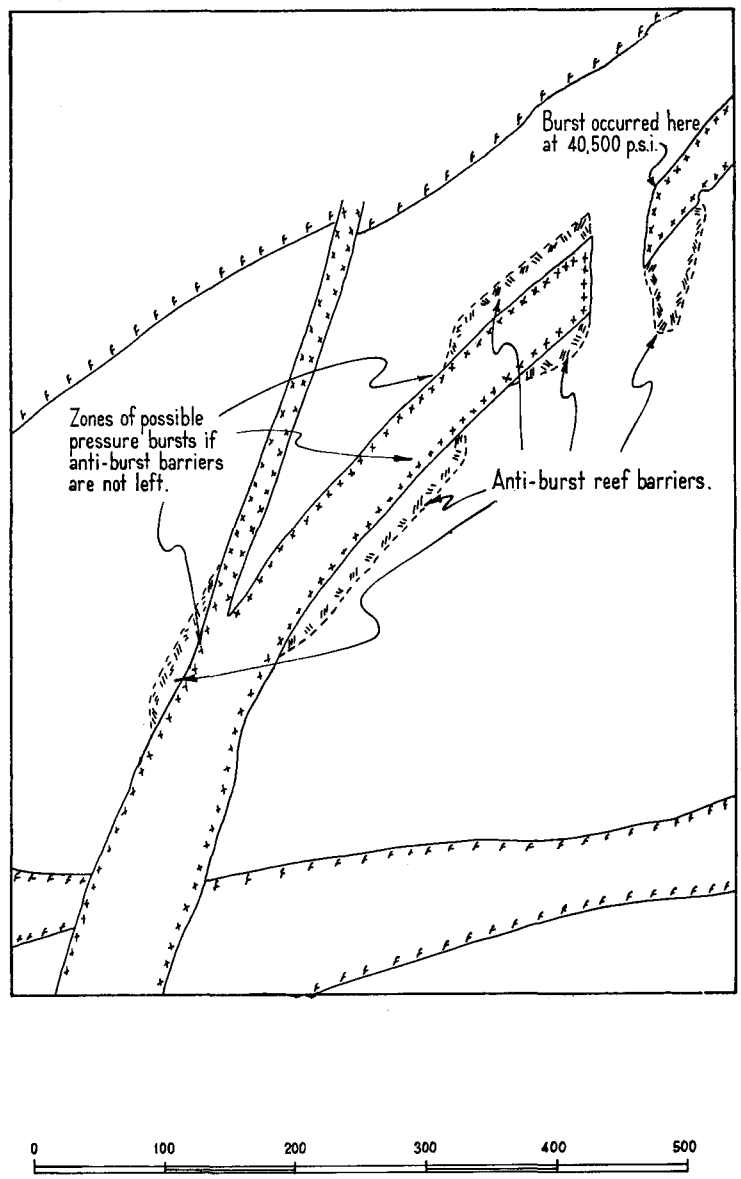


Fig. 22—Dyke burst protection pillars

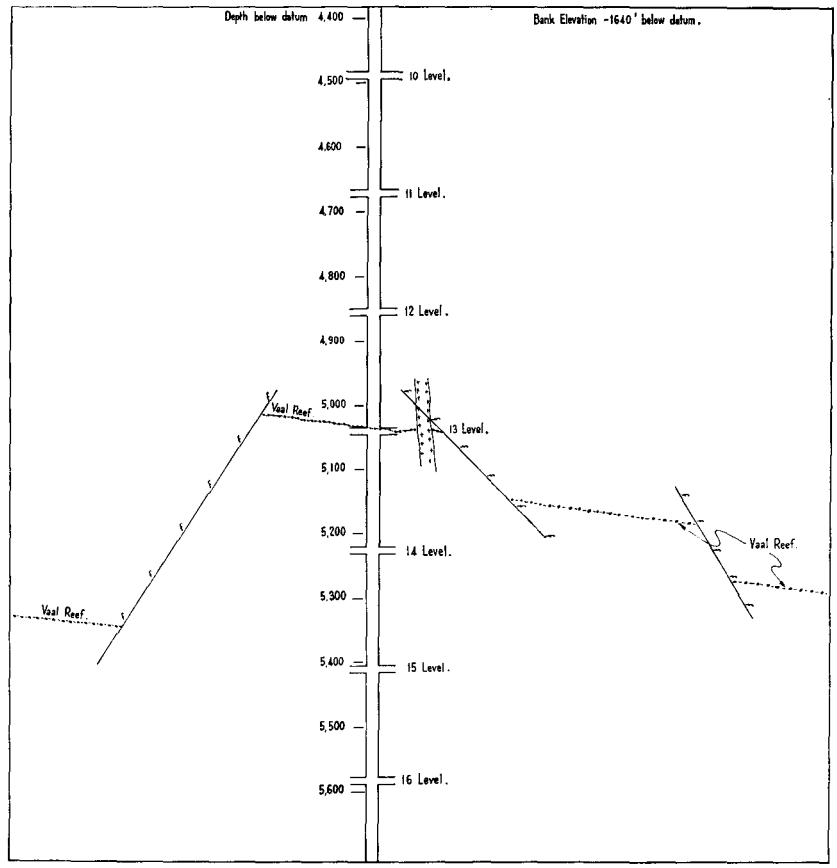


Fig. 23—Section of Toni Shaft, Stilfontein G.M. Co., Ltd.

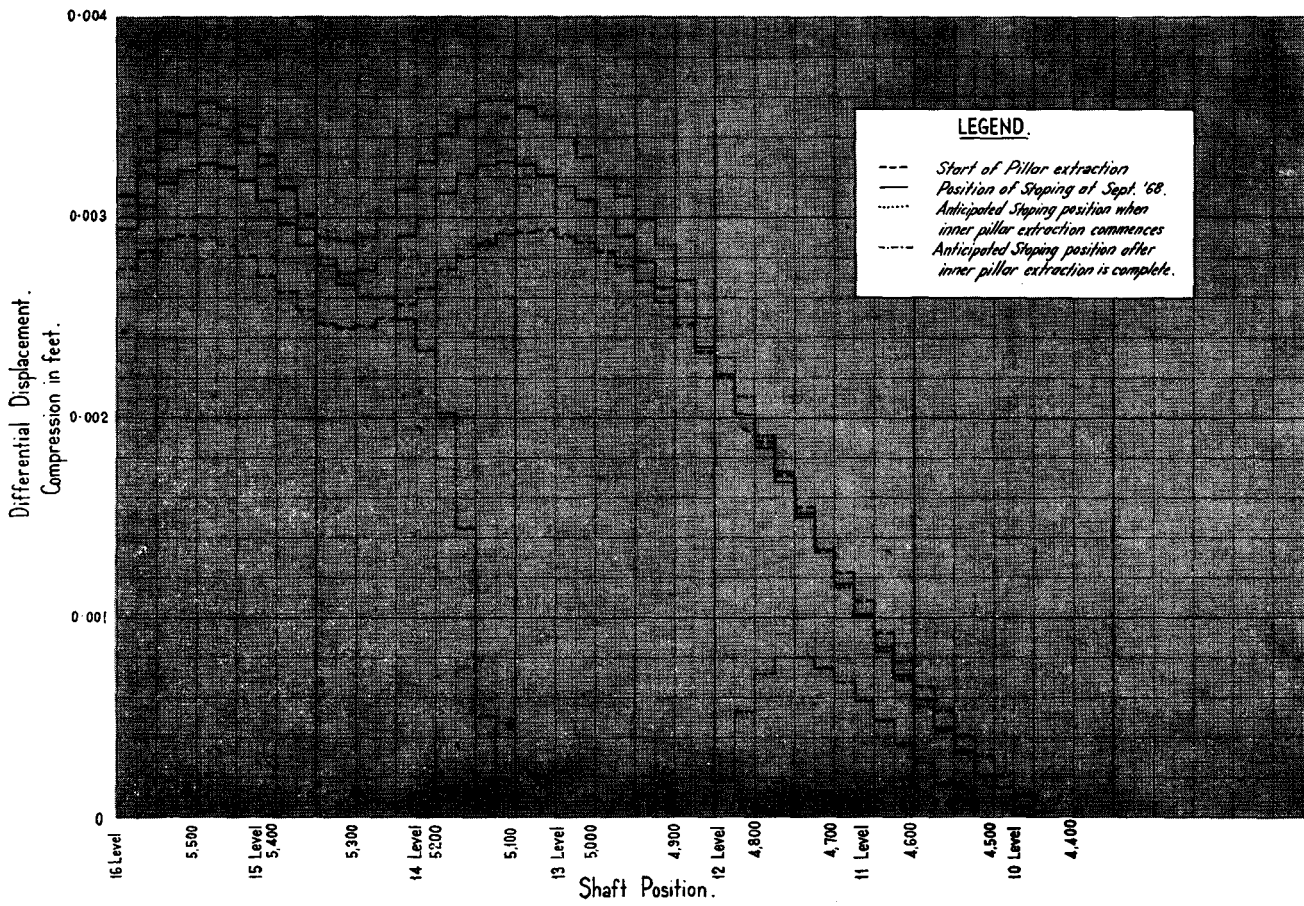


Fig. 24—Vertical differential movements in Toni Shaft

From this information it was evident that a careful check had to be kept on the gauge of the guides between a point 75 ft below 12 level and 16 level if stoping was to be continued as planned. Alternatively, steps would have to be taken to decompress the guides. Once the inner pillar had been stoped out observations for buckling would only be necessary between 15 and 16 levels.

2.2 *Examples from the Orange Free State goldfields.*

Several long- and short-term planning projects are described, where the data derived from the electrical resistance analogue has been processed by computer programmes and assessed on the strength of the criteria derived from the correlation of field observations and the elastic theory.

2.2.1 *An investigation into the effects of future stoping operations on the stability of the No. 1 Shaft system at President Brand.*

At the time of the analysis, distortion was strongly evident at the positions in the shaft shown in Fig. 25. By calculating elastic stresses and displacements at the troublesome sections of the shaft for the mining configuration in existence, the effects of future stoping operations on the overall stability of the shaft was assessed.

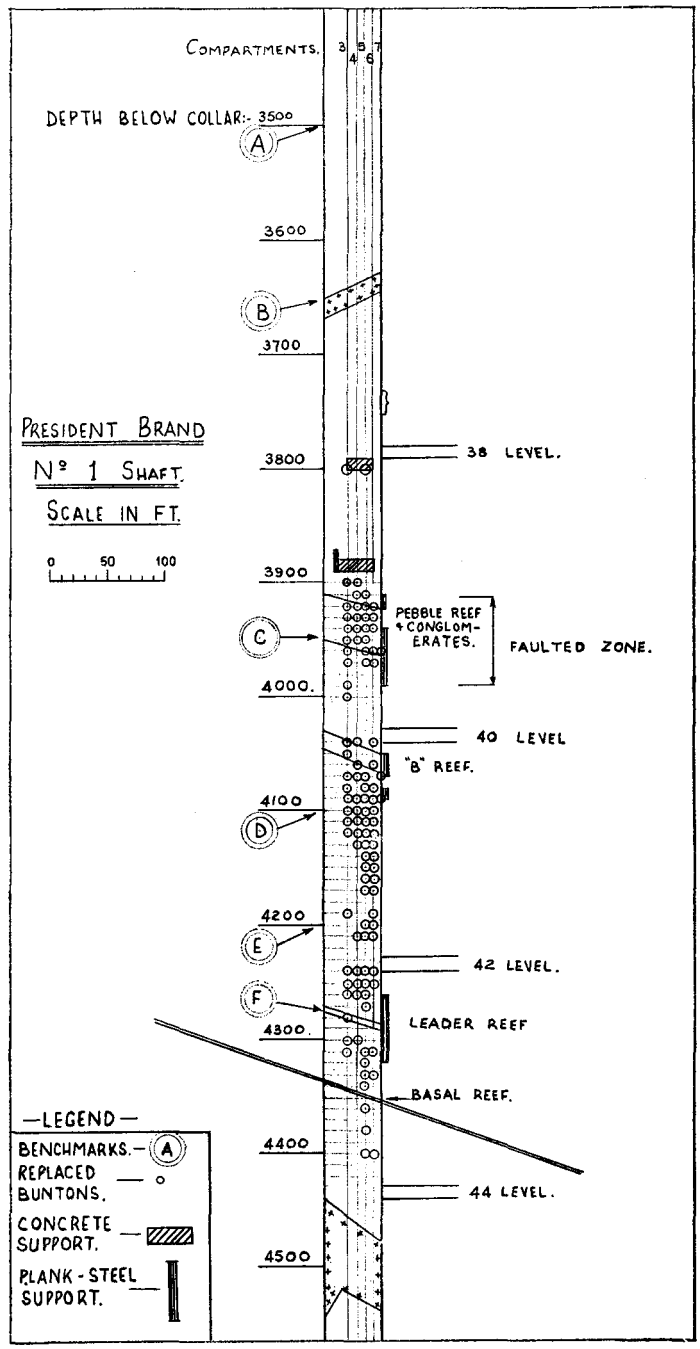


Fig. 25—Section of President Brand No. 1 Shaft showing zones of damage

(*a*) *Method of investigation.*

The area shown in Fig. 26 was modelled on the analogue and the appropriate stress values at the various elevations shown in Fig. 25 were calculated as described earlier in this paper. A similar analysis was performed at yearly intervals, consistent with the proposed stoping forecast for this shaft, and the elastic stresses were re-calculated. Table I summarizes the changes in vertical stresses at the reference points selected in the shaft for the stoping forecast given.

(*b*) *Discussion of results and recommendations*

Table I shows that the stress levels in the worst sections of the shaft can be expected to increase by almost 20 per cent over the ensuing two years compared with existing conditions. On the basis of current experience in this shaft it seemed reasonable to deduce that such a large increase in stress concentrations could well cause the operations within the shaft to be unsafe, if not altogether impossible.

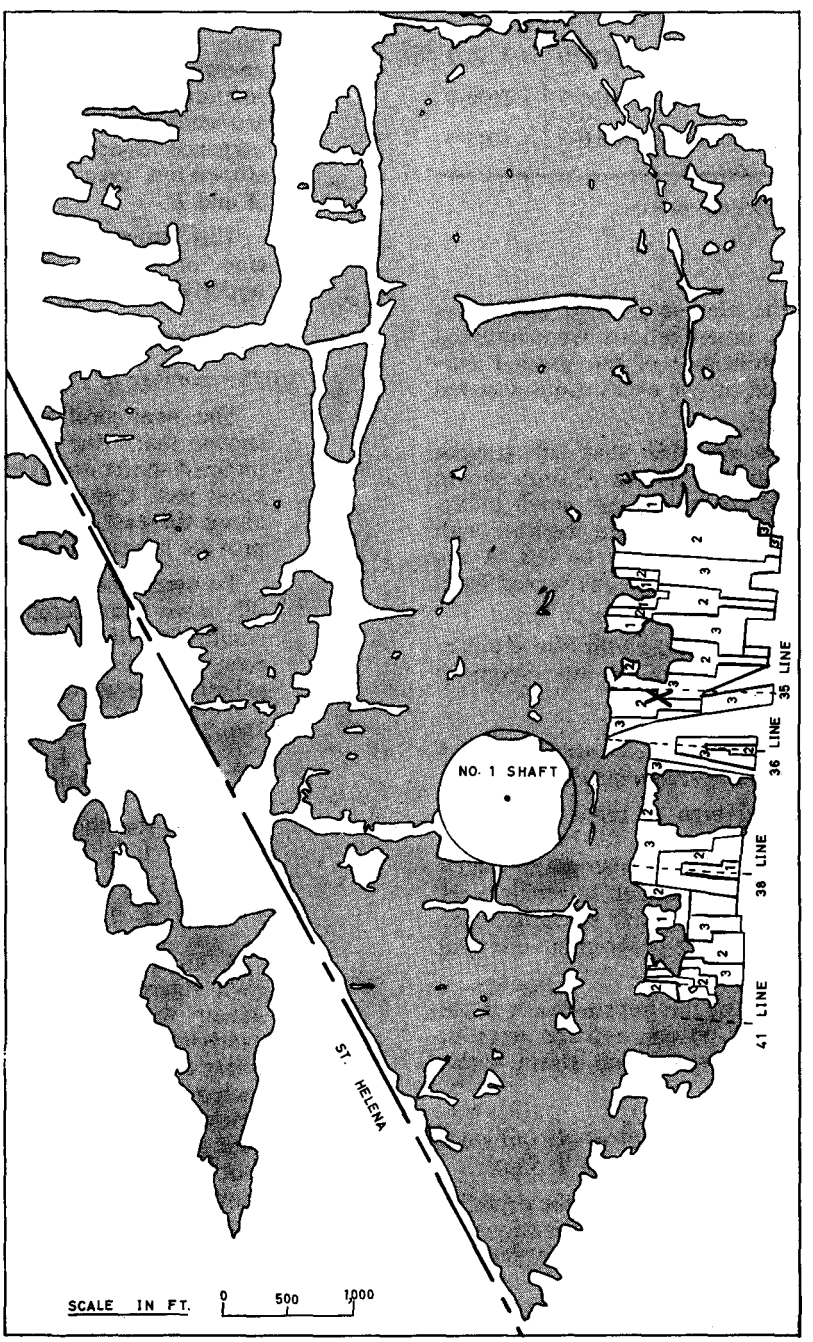


Fig. 26—No. 1 Shaft, President Brand, showing section and area modelled on the analogue tutor

TABLE I

PRESIDENT BRAND NO. 1 SHAFT PILLAR EXERCISE

Bench Mark	Depth Below Collar (ft)	Total Stresses lb/in² *EZZ*			
		Present @ 1967	Pillar 35·41 @ 1969	Pillar 35·38 @ 1969	Total Mining @ 1969
A	3500	−4389·6	−4585·1	−4603·0	−4479·1
B	3650	−4958·2	−5205·9	−5230·5	−5192·9
C	3950	−6336·3	−6610·4	−6649·5	−7022·2
D	4100	−7083·4	−7333·0	−7379·0	−8062·3
E	4200	−7566·7	−7806·0	−7856·0	−8726·2
F	4275	−7844·9	−8085·2	−8136·5	−9086·0
G	4440	−7909·1	−8153·2	−8200·5	−9045·9

Negative stresses compressive.

To minimise the 'change in stress' conditions within the shaft, investigations were initiated to assess the significance of the ground currently being mined on the excavations situated in the shaft pillar.

These analyses revealed that all stoping operations between the 35 and 38 lines shown in Fig. 26 should cease, and the small pillars scattered around the shaft pillar, together with the ground marked *X*, should be left *in situ* until all future mining operations served from this shaft are terminated.

It was recommended that all the above pillars should be extracted only in the later stages in the life of the shaft.

(*c*) *Behaviour of the shaft system one year after implementing the above recommendations.*

The amount and type of repair work performed on the No. 1 Shaft as far back as 1963 is shown in Fig. 27. This histogram clearly shows that a marked reduction in repair work in the shaft has resulted since the alterations were made to the stoping programme one year ago.

This is indicative of the improvement in the stability of the shaft system, and the arresting of severe changes in stress and strain within the shaft.

2.2.2 *An investigation into the stability of the proposed inclined shafts at President Steyn No.* 1 *Shaft.*

An investigation was made to assess the probable effects of proposed nearby stoping operations on the two inclined shafts currently being sunk at No. 1 Shaft.

Resulting from this investigation it was hoped to assist Management in deciding:

(i) whether a reef pillar should be left to protect the two inclines;

(ii) whether the inclines should be overstoped completely, and if so, how feasible the proposed five years' stoping forecast in this area would be.

(*a*) *Approach to the problem.*

This investigation was performed in two parts, namely:

(i) to determine the energy release rates for the five years' stoping programme;

(ii) to obtain the changes in field stresses and displacements at selected points within the inclined shafts at yearly intervals, in accordance with the five years' stoping programme.

(*b*) *Execution of the work.*

The areas *A* and *B* in Fig. 28 were set up in turn on the analogue for the determination of energy release rates, as described in an earlier section of this paper. After each mining sequence was extracted the complete convergence distribution was recorded for all stoped-out areas occurring within the areas *A* and *B*.

This latter data was used for the determination of stresses and displacements in the underlying inclined shafts.

(*c*) *Discussions of results.*

(i) *The stability of the inclined shafts:*

The geological section shown in Fig. 29 shows that, due to faulting, the proposed inclined shafts are at varying depths below the basal reef. Consequently, an average reef dip along the east-west axis of the inclines would provide unrealistic results.

To overcome this discrepancy three angles of dip were considered to provide meaningful results for all reference points along the east-west axis of the inclines. The reef dips considered were as follows:

Dip 25°—for the analysis of the reference points 1, 2, 3, 4, 5, 6, 91 and 92 (Fig. 29).

Dip 28°—for the analysis of reference points 1, 2, 3, 4, 5, 6 and 91.

Dip 37°—for the analysis of reference points 6, 7, 8, 9, 10, 11, 93 and 94.

A complete elastic analysis was made for each reference peg at yearly intervals, consistent with the enlargement of the stopes overlying the inclined shafts. Table II illustrates the variation in total vertical stresses for each reference peg for the seven mining configurations considered. Accepting the critical field stress of 8 000 lb/in.² for average O.F.S. footwall quartzites mentioned earlier, Table II shows that only reference points 4 and 92 can be expected to encounter critical compressive stresses. Although reference point 4 only reaches this critical field stress level after overstoping has been extensively carried out to the north and south (some time after 1972), point 92 (which is 50 ft below reef) will be influenced by critical stress concentrations as early as 1969.

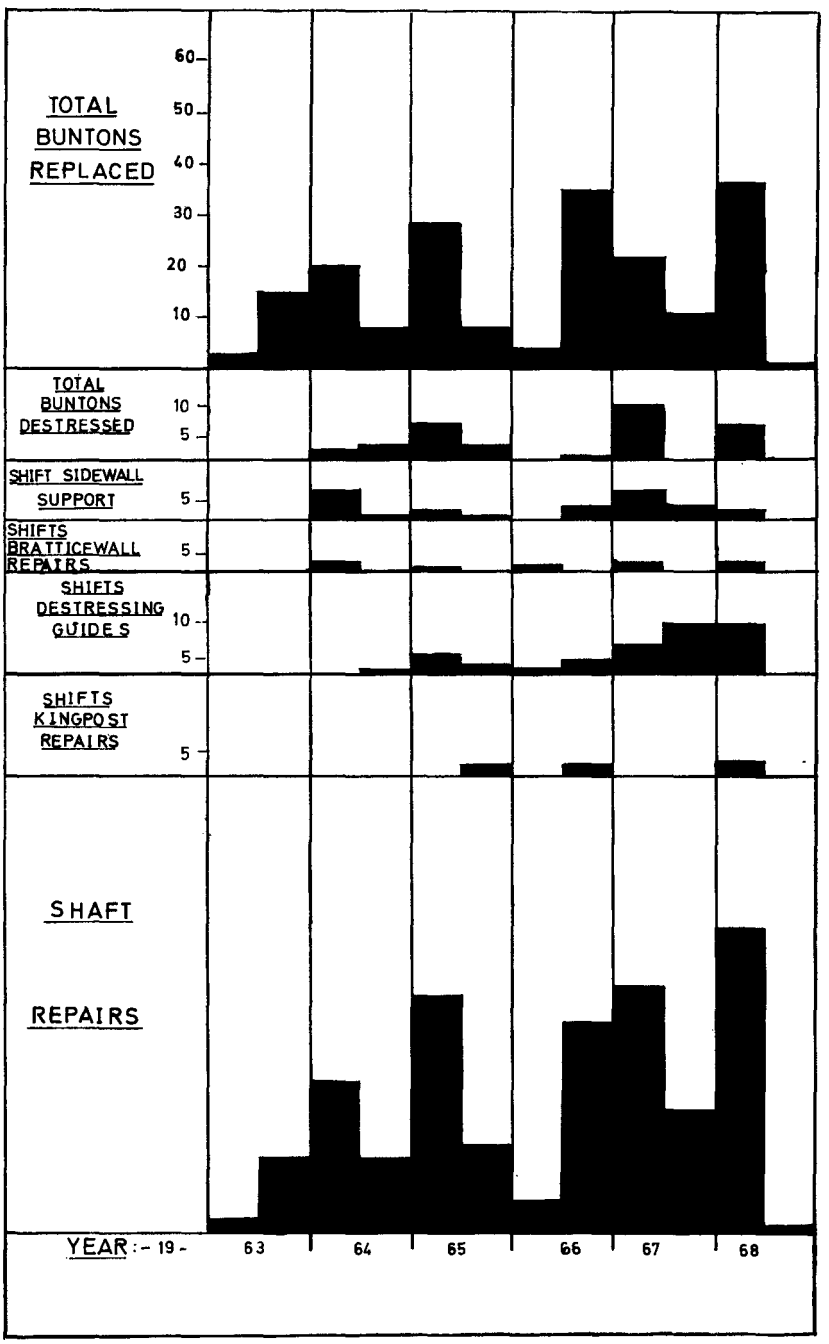


Fig. 27—Histogram of shaft repairs, President Brand No. 1 Shaft

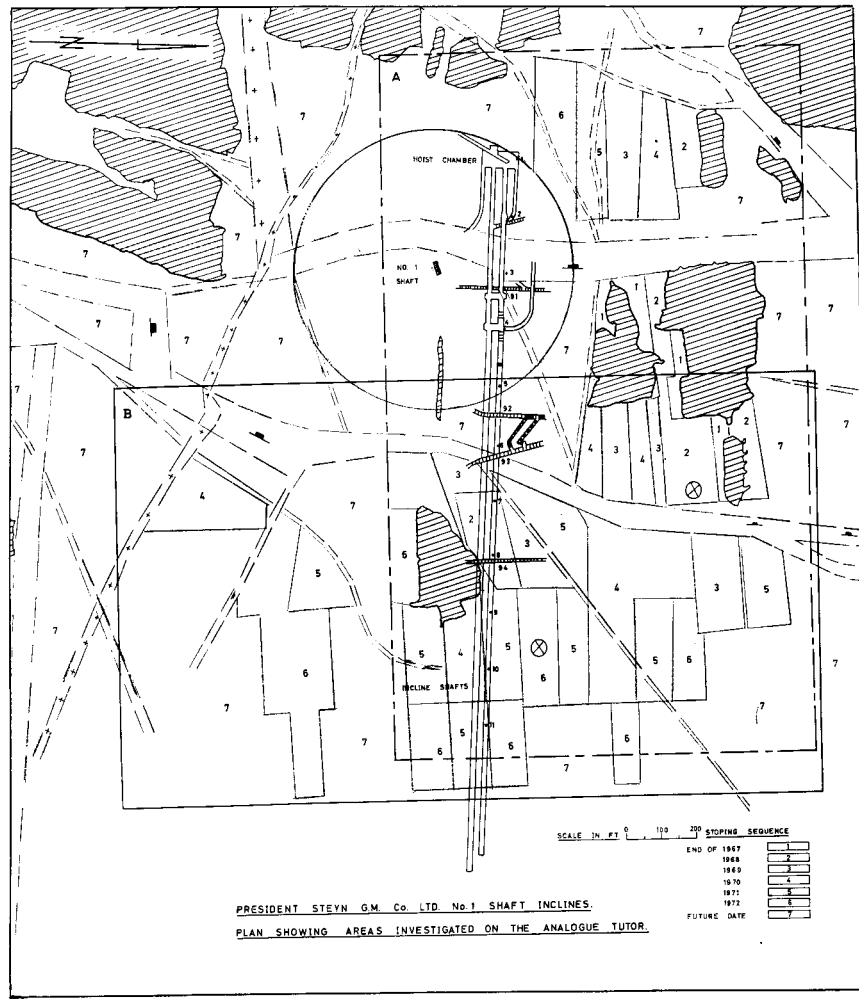


Fig. 28

(ii) *Energy release rates predicted for the five years' stoping programme.*

The analysis of energy release rates (i.e. hanging-wall conditions to be expected whilst stoping) over the five years' period for both sections modelled indicated that in two areas (*X* in Fig. 28) excessive energy release rates were predicted (i.e. above $0 \cdot 4 \times 10^8$ ft/lb per fathom mined). In both instances the conditions can be improved by 'down-dip mining' towards solid ground.

(*d*) *Conclusions.*

The analysis described above revealed that there was no need to leave a pillar to protect the inclines, provided a suitable support system was installed as soon as possible in the vicinity of reference points 4 and 92.

The stoping programme envisaged over the next five years is not expected to create unmanageable strata control conditions, even in the two areas referred to above. Some relief can be obtained in these areas by down-dip stoping towards solid ground.

2.2.3 *An investigation into the stress concentrations likely to occur in two dykes traversing the upper workings at Free State Geduld No. 1 Shaft and No. 4 Shaft.*

As a result of frequent 'bumps' in the planes of two dykes traversing a portion of the ground mined by Nos. 1 and 4 Shafts, varying degrees of damage had resulted in footwall haulages penetrating the dykes. Moreover, injuries to workmen in nearby stopes have resulted from the 'shake-up' associated with the 'bumps' which tend to dislodge loose rocks and cause minor roof falls.

A detailed analysis was carried out on the effect of a proposed mining schedule on the above dykes at the lower levels of No. 1 Shaft and No. 4 Shaft, from which it was found necessary to make a number of alterations to the proposed stoping programme in order to minimize future damage. These alterations succeeded in eliminating further bumping from the dykes, and showed that the analogue indications were valid. As the same two diorite dykes extend up to the overlay areas of the abovementioned shafts (600 ft reverse fault), a similar analysis was performed. The object of this analysis was to assess whether waste stoping would be necessary to protect footwall haulages, and also to ensure that high stresses were avoided on the dykes to minimize the possibility of 'bumps'.

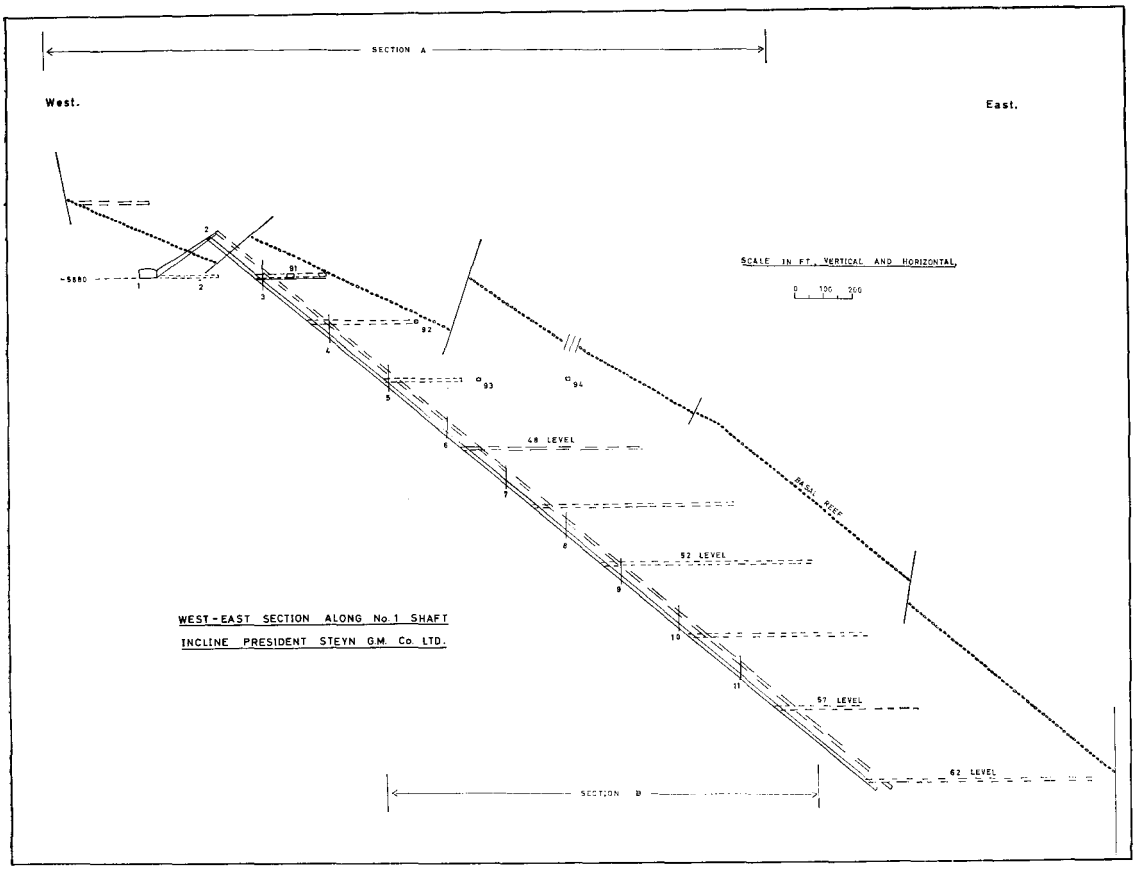


Fig. 29

TABLE II

PRESIDENT STEYN NO.1 SHAFT INCLINES
TOTAL STRESSES IN VERTICAL DIRECTION (lb/in²)

Negative stresses compressive.

Bench Mark	1967	1968	1969	1970	1971	1972	Later Date
1	−5004	−5039	−5074	−5150	−5232	−5294	−5662
2	−4891	−4938	−4976	−5062	−5164	−5356	−6136
3	−5042	−5102	−5147	−5252	−5350	−5476	−6626
4	−5264	−5339	−5430	−5570	−5720	−5877	−8442
5	−5427	−5551	−5748	−5824	−6164	−6413	−5955
6	−5591	−5386	−4905	−5055	−5106	−5399	−4979
7	−5690	−5596	−5545	−5100	−4520	−4520	−2915
8	−5925	−5919	−5909	−5860	−5620	−4928	−3358
9	−6140	−6141	−6141	−6124	−5868	−5652	−4604
10	−6343	−6343	−6347	−6348	−6188	−6179	−5768
11	−6543	−6543	−6547	−6550	−6548	−6523	−6447
91	−5089	−5153	−5204	−5314	−5419	−5547	−6864
92	−5263	−5502	−8697	−8979	−10238	−11192	−22300
93	−5316	−5163	−5012	−5025	−4525	−4304	−3475
94	−5436	−5475	−5525	−5360	−3137	−1829	+2605

(*a*) *Investigation on the analogue.*

The area modelled on the analogue is shown in Fig. 30. This illustration depicts the probable stoping sequence for this area, together with the proven geological anomalies. In order to measure the change in stresses along the dykes as mining operations progressed, pins representing the two dykes were recorded after each mining sequence was extracted. Fig. 31 shows the normal stress values along the two dykes for the last mining sequence shown in the area.

(*b*) *Discussion of results.*

According to tests carried out by the C.S.I.R. Rock Mechanics Division in Pretoria, strength values of rock specimens taken from dykes *A* and *B* at 49 level, revealed uniaxial compressive strengths of 35 000 and 40 000 lb/in^2, respectively, in each case the standard deviation being $\pm$ 5 000 lb/in^2 about the mean strength. A consideration of Fig. 31 shows that reef stripping on dyke *B* from the north side by No. 1 Shaft will create stresses on the dyke approaching the level associated with 'bump conditions' for this dyke.

(*c*) *Conclusions and recommendations.*

The indications obtained from this analysis coupled with 'bumping' experience encountered at Free State Geduld, suggested that portions of both dykes should be extracted in order to prevent the build-up of stress on the dykes, and thus minimize the possibility of 'bumps' originating from them.

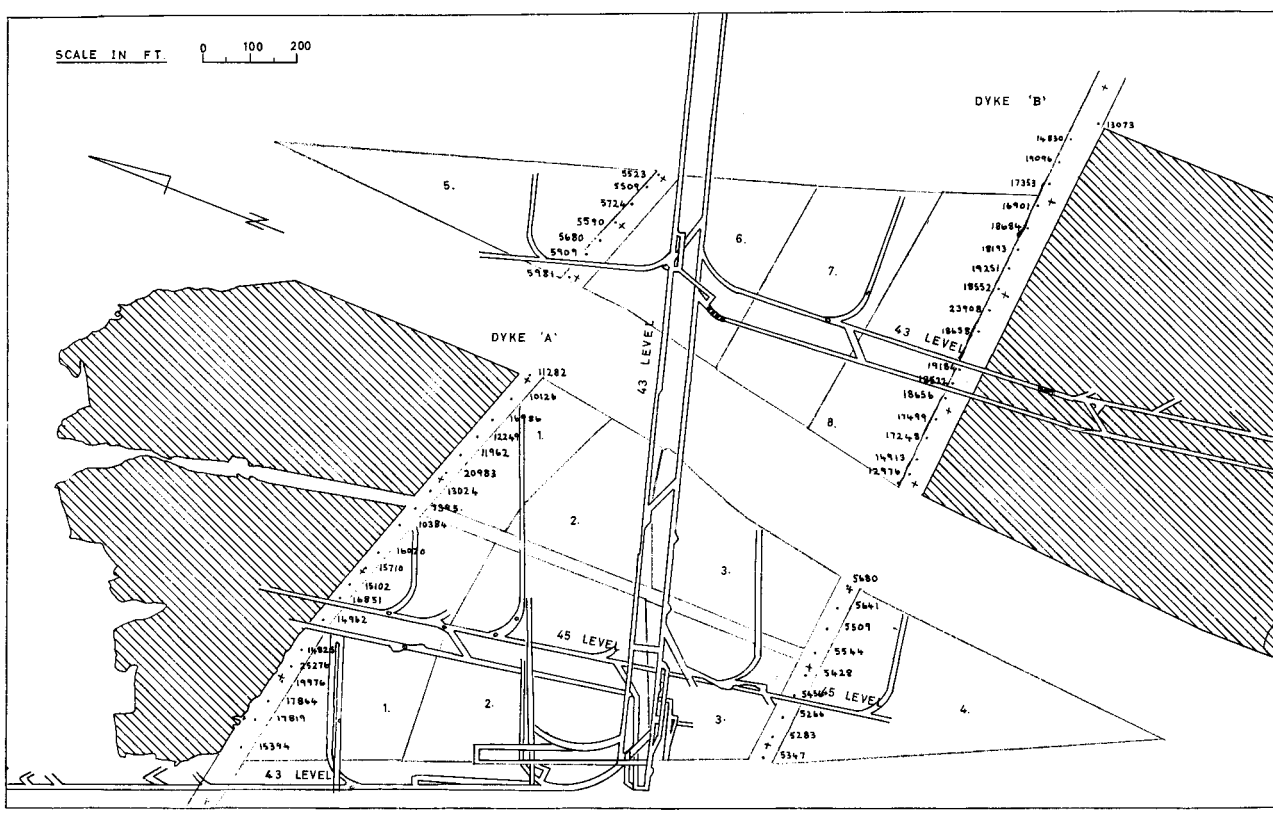
The increase in stress on dyke *A*, due to stoping, will cause the dyke to be exposed to a higher pressure than that known to have existed when previous 'bumps' occurred in the lower levels of the mine. As a result of these findings it was agreed to extract the dyke completely over the underlying haulages.

Dyke *B* has given considerable trouble over the period 1967 to 1968 in the 49 level areas at both No. 1 and No. 4 Shafts. Consequently, where the workings are currently stripping on dyke *B* at No. 4 Shaft boundary (43 level), it was recommended that the dyke be extracted between the limits shown in Fig. 31 before No. 1 Shaft stoping programme commences in this area.

To date, no adverse reports have been received regarding 'bumps' emanating from the dykes.

2.2.4 *The planning of stoping sequences in the* 45 *North area adjacent to the Free State Geduld boundary at Western Holdings.*

An early investigation of the two-year stoping programme for the extraction of the basal reef in the northern section of Western Holdings, Ltd., clearly indicated that energy release rates would be far greater than those known to have caused severe damage in some stopes in adjacent mining areas.



STRESS CONCENTRATIONS ON DYKES 'A' AND 'B' IN THE OVERLAY WORKINGS AT F.S.G. NOS. 1 AND 4 SHAFTS, PRESENT SITUATION.

Fig. 30

Because of the extensive mined-out area north of the area to be mined, it was apparent that mining operations in this area should retreat from the Free State Geduld boundary in a general south-easterly direction if these energy release rates were to be reduced. However, because the raise connections required for the mining of this area were already pre-developed, stoping operations would have to start from them.

Before mining was commenced in the most northerly raise connection the stoping sequences proposed were investigated on the analogue using the energy release rate concept. Fig. 32 shows the energy release rates resulting from the stoping sequence given. In view of the extremely high values obtained in several areas it was decided to carry out experiments on the analogue to find the optimum stoping sequence for the area—consistent with the output requirements from the area and the present knowledge of geological anomalies.

Conclusions and recommendations.

It is known that energy release rates can be reduced by advancing working faces towards solid ground. This approach was borne in mind in the experimentation with mining sequences on the analogue, and eventually, the mining sequence shown in Fig. 33 was considered to be the optimum method for the given specifications.

In this latter illustration the critical energy release rates are only obtained on one stope face at any one period in time, thus difficulties associated with stoping operations in this area will be minimized and great fluctuations in output from the various rock-breakers should be avoided.

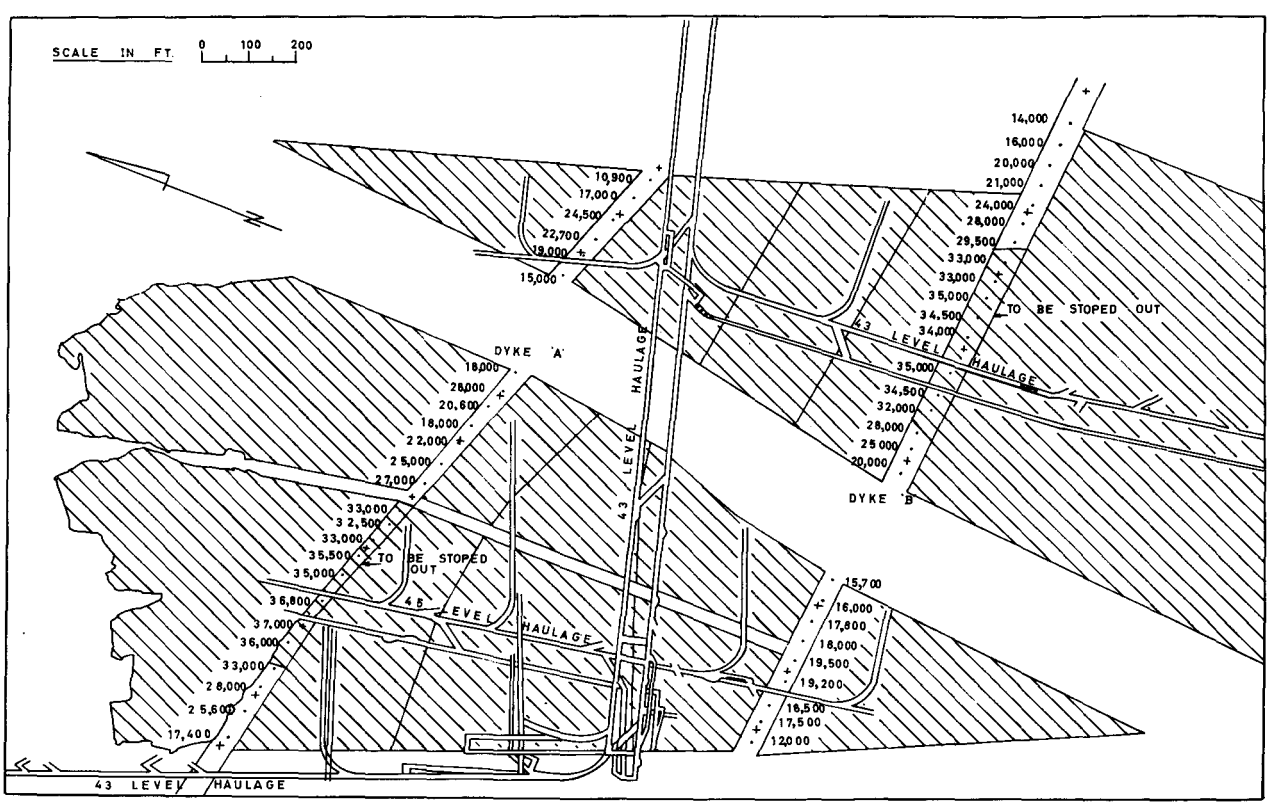
Before implementing this stoping plan it was emphasized that down-dip mining in the last 50 ft or more of each panel advancing towards the F.S.G. old workings would assist in reducing the energy release rates in this final condition.

Fig. 34 shows the general idea of the stope configuration likely to create the least hanging-wall problems in this area, and at the same time, be a reasonable proposition from a mining point of view.

Results to date.

Output from the 45-21 and 47-21 stopes for the five months' period of concentrated mining in this area to date has been as follows:

Upper section (45-21)	*Lower section* (47-21)
407 fathoms per month	444 fathoms per month
509 fathoms per month	521 fathoms per month
446 fathoms per month	514 fathoms per month
441 fathoms per month	469 fathoms per month
302 fathoms per month	353 fathoms per month



CHANGES IN STRESS CONCENTRATIONS ON DYKES 'A' AND 'B' IN THE OVERLAY WORKINGS AT F.S.G. NOS. 1 AND 4 SHAFTS AFTER LAST SEQUENCE.

Fig. 31

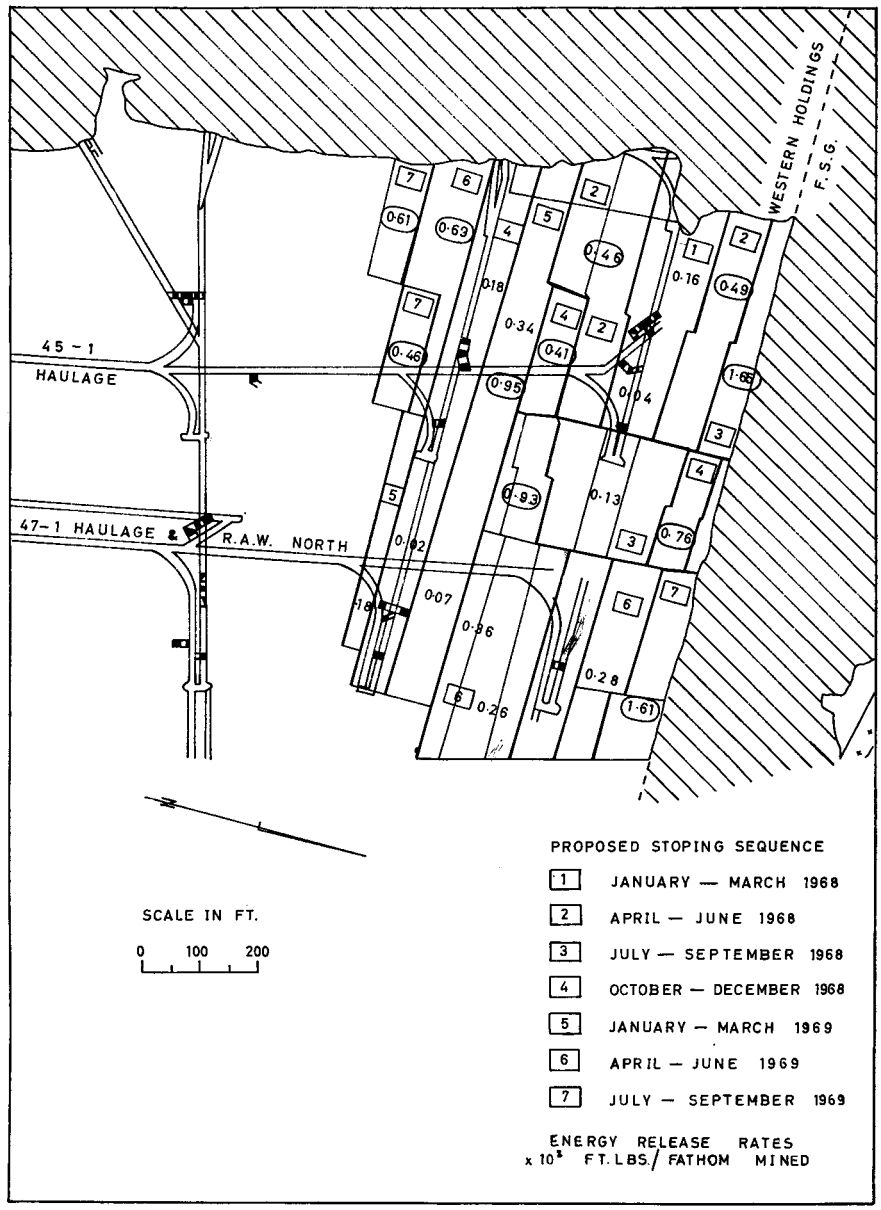


Fig. 32—Energy release rates for proposed stoping plan in the 45-47 North Area Western Holdings

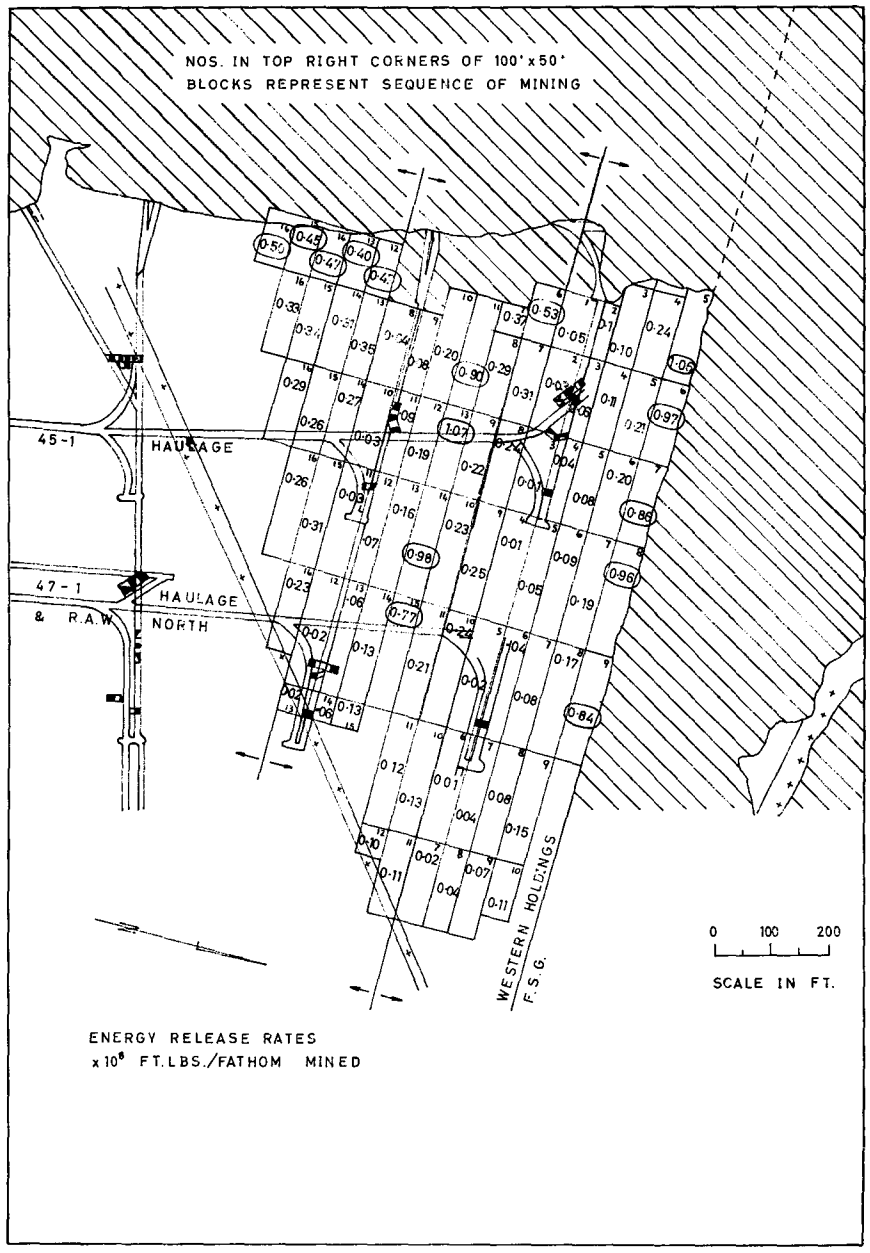


Fig. 33—Energy release rates on 100 ft long panels at 50 ft intervals of face advance, Western Holdings 45 North Area

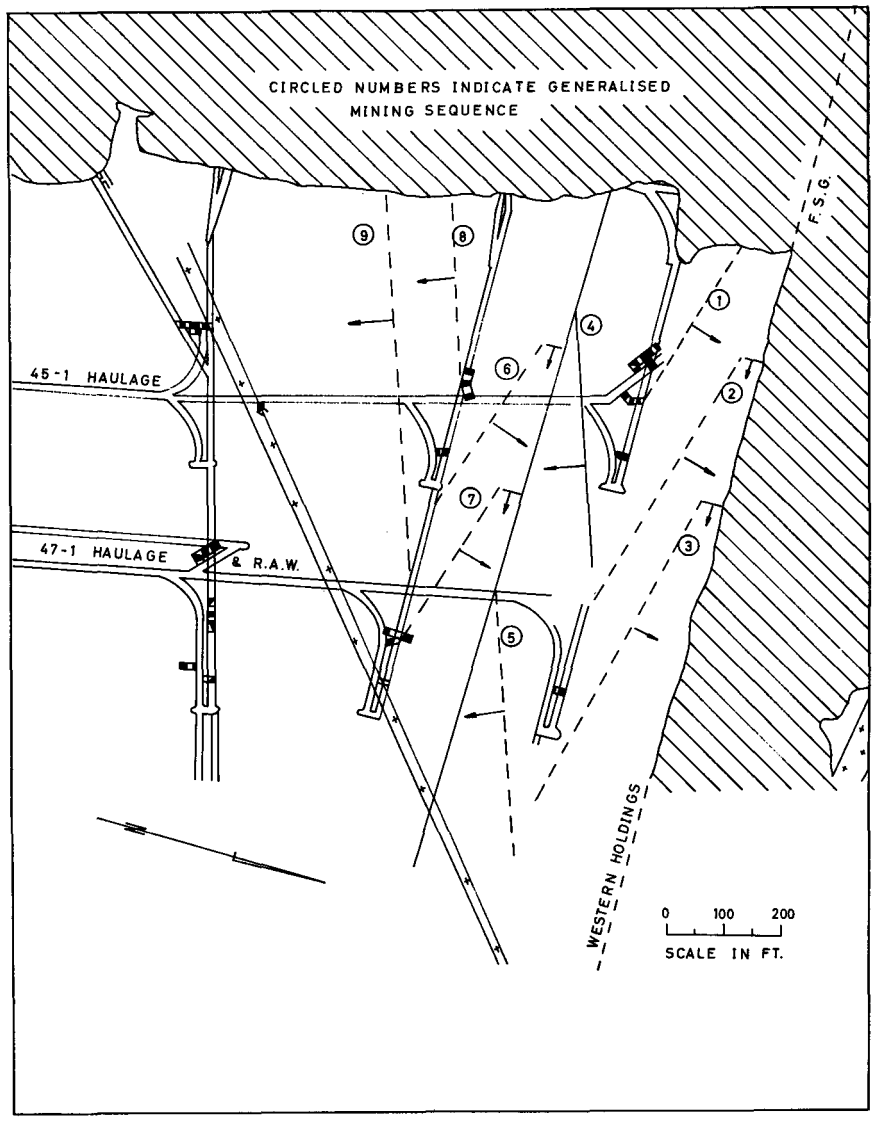


Fig. 34—Western Holdings No. 1 Shaft 45 North Area

Unfortunately, minor faulting on the upper south faces has prevented the maintenance of the stoping sequence given, and a drop in production has been unavoidable. It is anticipated that once the faulted faces are completed the output from the upper contract should return to about 350 fathoms per month.

2.2.5 *The determination of the reef pillar size required to protect the 45B/44 inclines at Welkom G.M. Co., Ltd.*

As a result of unpay blocks and complicated faulting traversing the 45B/44 inclines it was apparent that a reef pillar would be required to protect the inclines as stoping in a northerly and southerly direction progressed.

This application of the analogue describes a method of determining the optimum reef pillar necessary to preserve the inclines for the required life of the incline shaft system.

Method of approach to the problem.

As no rapid method is available for determining the minimum pillar size to protect off-reef excavations, it was considered necessary to measure the influence of future stoping operations on the inclines for a variety of different pillar sizes.

This was achieved by modelling the area of the mine in the vicinity of the proposed inclines on the analogue, as shown in Fig. 35, and making allowances for the mining which will ultimately take place in the area.

Four stope configurations were considered for the investigation, namely:

1. After the two years, stoping forecast is completed.
2. All mineable ground is extracted, except a 480 ft wide pillar symmetrically located over the inclines.
3. All mineable ground is extracted, except a 320 ft wide pillar over the inclines.
4. All mineable ground is extracted, except a 160 ft wide pillar over the inclines.

The theoretical convergence distribution in the stoped-out areas for each stope configuration was used to calculate the field stresses at the reference points shown in Fig. 36.

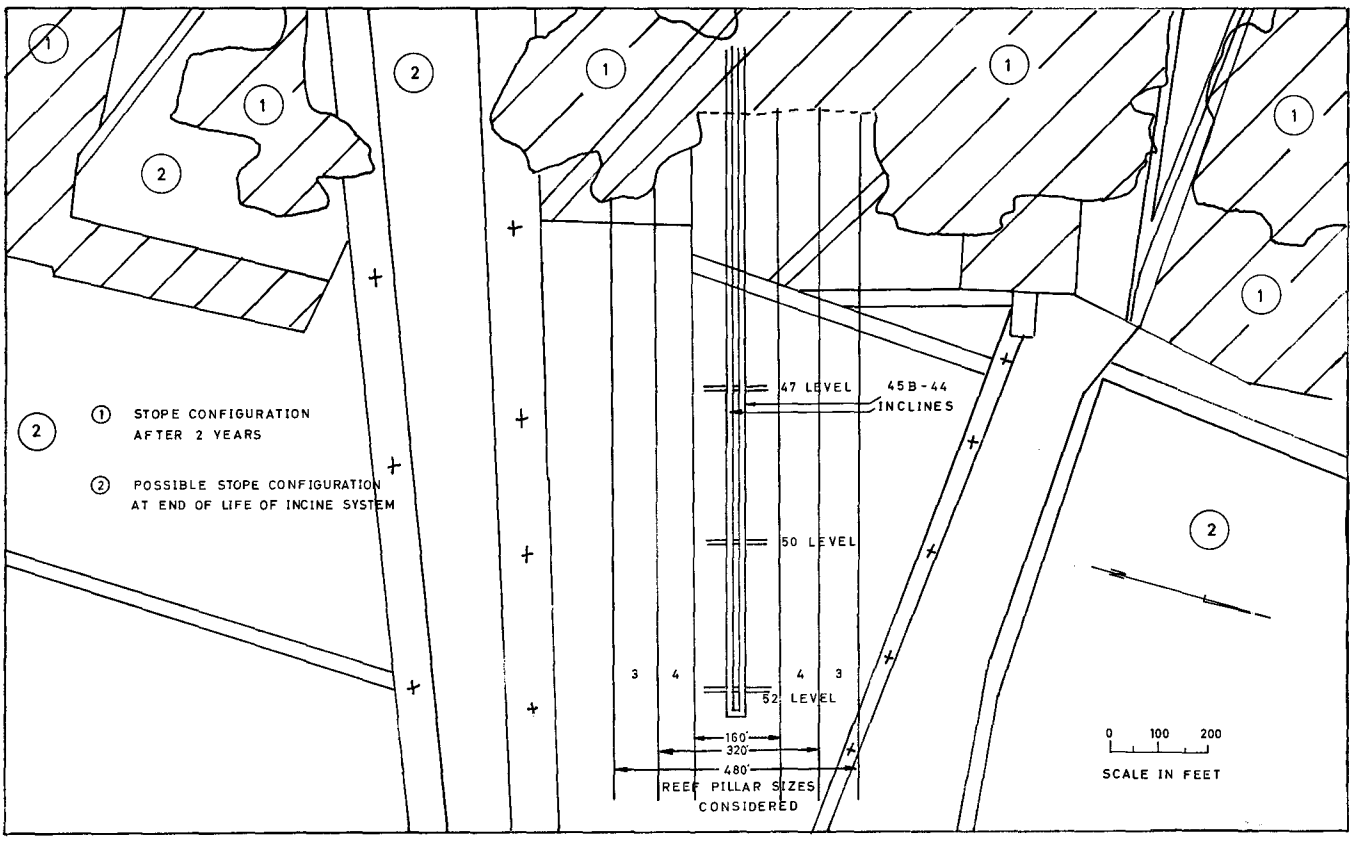


Fig. 35—Area to be mined in the vicinity of 45B—44 inclines, Welkom G.M. Co. Ltd.

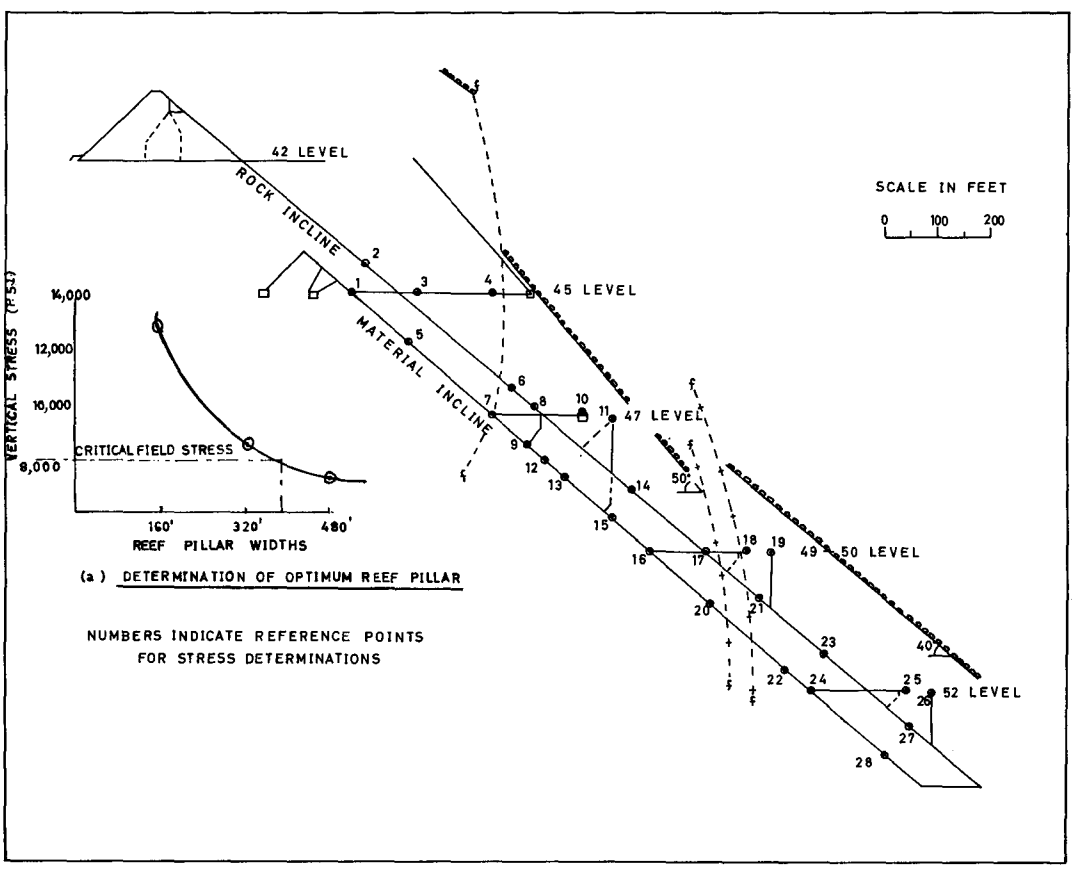


Fig. 36—Welkom G.M. Co., Ltd.—45B—44 inclines.

Discussion of results and conclusions.

Table III shows the changes in vertical field stresses at points selected in the incline system for the three pillar sizes investigated.

The stress levels shown in Table III suggest that a pillar size of 320 ft will be sufficient to avoid critical field stresses. However, excavations located between the upper rock incline and the basal reef (i.e. the station and tippler cross-cuts) would be exposed to vertical stresses in excess of 8 000 lb/in². Consequently, a pillar size greater than 320 ft and less than 480 ft will be optimum.

Fig. 36 shows a typical graph of stress level plotted against pillar size, for a point representing an important excavation in the 45B/44 incline system. This graph shows that a pillar of 410 ft in width throughout the length of the incline will be required to ensure that the critical field stress of 8 000 lb/in² is avoided at all times in all excavations.

TABLE III

VERTICAL STRESS VALUES (lb/in²) IN THE WELKOM 45B/44 INCLINE SHAFT SYSTEM

Reference Points	Pillar Sizes		
	160 ft	320 ft	480 ft
POINTS IN THE ROCK INCLINE			
2	−5,995	−5,951	−5,767
6	−7,356	−6,787	−6,586
8	−7,884	−7,061	−6,654
14	−8,338	−7,338	−6,718
17	−8,841	−7,850	−7,342
21	−8,787	−7,854	−7,323
23	−8,467	−7,733	−7,287
27	−8,030	−7,502	−7,235
POINTS IN THE MATERIAL INCLINE			
1	−5,751	−5,776	−5,636
5	−6,098	−6,089	−6,063
7	−6,640	−6,528	−6,451
9	−6,537	−6,526	−6,433
12	−6,900	−6,748	−6,571
13	−6,948	−6,780	−6,591
15	−7,090	−6,900	−6,660
16	−7,259	−7,020	−6,714
20	−7,647	−7,467	−7,241
22	−7,535	−7,403	−7,217
24	−7,545	−7,414	−7,221
28	−7,232	−7,188	−7,161
POINTS IN THE STATION AND TIPPLER CROSSCUTS			
3	−6,792	−6,672	−6,404
4	−10,109	−7,851	−6,873
10	−9,246	−7,741	−6,717
11	−10,631	−7,886	−6,804
18	−10,702	−8,567	−7,488
19	−12,978	−8,563	−7,591
25	−9,814	−8,263	−7,305
26	−10,611	−8,855	−7,187

Negative indicates compressive stresses.

CONCLUSION

The applications of the electrical resistance analogue described in this paper demonstrate the rapid advances made in the field of practical applied rock mechanics in deep-level hard-rock mines.

It has been shown that, despite the geological differences between the O.F.S. and Klerksdorp goldfields, the analogue can provide data on which to design mining layouts in both areas—provided the analogue indications are interpreted on the basis of the rock behaviour as experienced underground.

By providing information of the type described the Rock Mechanics departments associated with this paper are playing an increasingly important role in the layout and safety of the mines with which they are associated.

It is anticipated that as more sophisticated computer programmes become available this contribution will be increased substantially.

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