Geotechnical and Geological Engineering, 1992, **10**, 185–202

Estimation of critical convergence and rock load in coal mine roadways – an approach based on rock mass rating

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Summary

Based on field instrumentation in eight different coal mines representing varying depths and strata conditions, a relation for obtaining the critical convergence value has been established. In development heading for bord and pillar workings this relation can be used successfully to control the premature collapse of the roof. An empirical relation for rock load has been established. This can be utilized for optimum design of support system. The roofs have been categorized as stable, short-term stable and unstable. Proper attention should be provided for an unstable roof and the support design is to be changed before the convergence reaches the critical value.

Keywords: Coal mine, rock mass classification, critical convergence, rock load, support design.

Introduction

Problems related to ground movements pose a major hazard in underground mines. About 40–45% of fatal accidents reported from underground coal mines can be attributed to such ground movements. Driving a roadway disturbs the rock mass penetrated. The penetrated rock mass will seek for and assume a new state of equilibrium. If the roadways are left unsupported, the new state of equilibrium would frequently be reached through collapse of the roof. Installing an appropriate support system is the method commonly used to prevent such undesirable occurrences. The function of a support system then is to permit the rock mass to reach a post drivage equilibrium state in which a safe usable roadway is guaranteed.

In most instances, the basis of support selection and design of a support system have been past experience and visual inspection at the working face. The design of a support system using a rule of thumb approach may cause the roadways to be under-supported or over-supported. Over-design of a support system leads to wasteful expenditure escalating the total cost of production; on the other hand under-design may become much more costly causing fatal accidents. Optimizing support system design calls for an adequate knowledge of rock pressure distribution and strata deformation at the extraction perimeter. Continuing efforts are being made to develop the necessary insight into this intractable problem from laboratory tests and numerical modelling. The behaviour and performance of a structure in the rock mass however may not match the laboratory determined properties. A numerical

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model which tries to provide a theoretical solution of the roof problem is often not adequate, because the pressure encountered by the support system in a real structure is very different from the usual theoretical assumptions.

In this paper, case history examples have been considered and rock mass response evaluated by in-situ instrumentation. The installation and monitoring were undertaken to estimate the expected pressure and displacements. A possible solution to the intractable problem of ground control has been attempted using in-situ measured data, which can be used for developing mine design strategies, that would minimize ground control hazards, while at the same time maximizing recovery at an acceptable cost.

Programme of work

The work described herein has two components: to measure strata deformation and support pressure. To carry out these observations, eight different coal mines, representing varying geotechnical conditions and different stress situations were selected. These mines were selected from various coalfields in India with bord and pillar as the method of work. The experimental sites include one horizon with a depth more than 300 m, four horizons with depths between 100 and 300 m and the remaining three with depths less than 100 m. Instrumentation was installed in each mine to monitor the roof and support load and strata deformation. For measuring roof and support load electronic load cells and borehole pressure cells were used. To measure the strata deformations different types of extensometers and convergence indicators were installed. These include double point borehole extensometers, multipoint borehole extensometers, remote sensing convergence indicators, suspension type convergence indicators etc. The instruments were installed at freshly exposed roofs and the monitoring was carried out for a period of 18 months in the majority of cases. In some cases the roofs were allowed to collapse and measurements were taken by remote indicating instruments. These were done to establish a reasonably accurate predictive norm for the roof falls.

CMRS rock mass classification (R)

Bieniawski' (1979) RMR method of rock mass classification was used as the basis of developing a new classification system suitable for Indian coal measure strata. Statistical analyses were carried out on geotechnical data obtained from 150 published case records and detailed field investigations in underground mines. First a general frequency survey of all causative factors was made to identify the factors exercising deleterious effects on roof stability. Then, multivariate statistical analyses were performed on the quantitative part of the data. Principal component analysis and factor analysis were used to evaluate the different groups of interrelated factors and discriminant analysis was used to determine the relative contribution of the individual variables in differentiating different types of roof conditions. Based on these analyses, five parameters were selected for the classification; namely, spacing of bedding planes, rock strength, weatherability, groundwater condition and structural disturbances. Further evaluation of the statistical data enabled the allocation of importance ratings to the parameters (Tables 1 and 2). The overall classification was evolved from these statistical studies, and the new system was satisfactorily applied to over 50 actual case studies in India (Ghose and Raju, 1981; Venkateswarlu *et al.*, 1989).

Table 1. Ratings of parameters in R

Parameter	Range of value				
1. Layer thickness (cm)	<2.5	2.5–7.5	7.5–20	20–50	>56
Rating (R)	0–5	6–12	13–19	20–25	26–30
2. Structural features (Structural Index)	>14	10–11	11–7	7–4	4–0
Rating (R)	0–4	5–10	11–16	17–21	22–25
3. Weatherability (%)	<60	60–85	85–97	97–99	>99
Rating (R)	0–3	4–8	9–13	14–17	18–20
4. Rock strength (kg cm^{-2})	<100	100–300	300–600	600–900	>900
Rating (R)	0–2	3–6	7–10	11–13	14–15
5. Groundwater (ml min^{-1})	>2000	2000–200	200–20	20–0	Dry
Rating (R)	0–1	2–4	5–7	8–9	10

Table 2. R values used for classifying roof rocks

R	Roof description	Class
0–20	Very Poor	V
20–40	Poor	IV
40–60	Fair	III
60–80	Good	II
80–100	Very Good	I

Description of the locales and observational data

Satgram Incline (Eastern Coalfields Limited)

Dishergarh seam is being worked in this mine at a shallow depth of about 50 m. The seam is 2.0 m in thickness and the full seam thickness is being extracted with 3.6 m wide galleries. The dip is gentle, 3.5° due SE. The roof is of carbonaceous shale and layered sandstone. The rock mass rating (R) for the roof rock was calculated to be 55. The dry density of the immediate roof rock was 2.215 tonne m^{-3}.

The instruments were installed in junctions as well as galleries. It was observed that under unsupported roof conditions in a four-way junction the maximum roof movement was 9 mm. The average rate of roof movement was 0.06 mm day^{-1} during the first month of the exposure of the roof. The unsupported junctions which collapsed, recorded a higher rate of convergence (0.1 mm day^{-1}), compared to other stable junctions. The maximum rock load recorded was 3.1 tonne m^{-2} in an unsupported junction (Fig. 1).

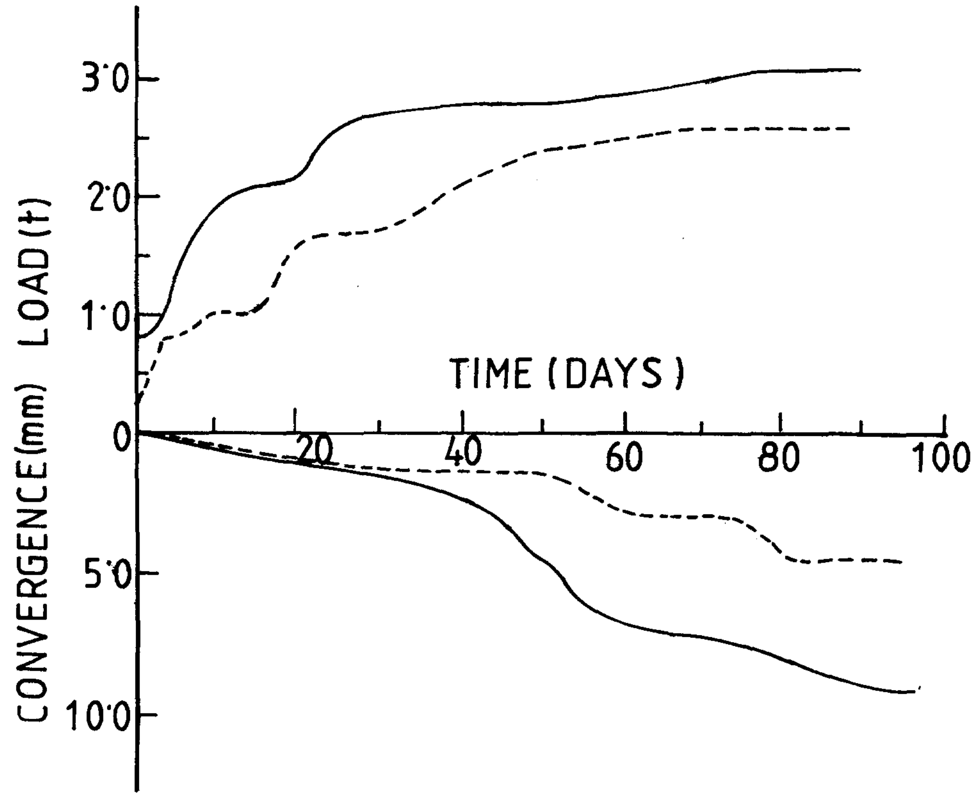


Fig. 1. Load and convergence measurements at Satgram Incline

Bansra Colliery (Eastern Coalfields Limited)

Purandip seam, 2.8 m in thickness, is being worked at a depth of 60 m, with bord and pillar as the method of work. The seam is gently dipping about 4° due S 55° E. The development galleries are in the bottom 2 m section with 3.6 m gallery width. The immediate roof consists of coal, shale, clay and shale in that order. The rock mass rating (R) was calculated to be 37. The immediate roof rock had a dry density of 2.063 tonne m^{-3}.

The maximum closure values recorded in unsupported four-way junctions ranged from 10 mm to 24 mm. The initial convergence rates were significantly high and was approximately 0.3 mm day^{-1} for the first month of the exposure of the roof. A maximum load of 6.1 tonne m^{-2} was recorded in an unsupported junction (Fig. 2).

Durgapur Rayatwari Colliery (Western Coalfields Limited)

At Durgapur Rayatwari colliery the Rayatwari seam is 17 m thick and occurs at a depth of 200 m. The dip of the seam is 12° due N 72° E. The bottom 3 m section of the seam is developed with bord and pillar working. The width of the gallery is 4.2 m. The roof consists of interlayered coal and shale bands. The rock mass rating (R) for the roof rock was calculated to be 24. The dry density of the immediate roof rock was 2.00 tonne m^{-3}.

Under unsupported roof conditions the maximum closure in a four-way junction was 45 mm. The rate of roof movement was about 1.0 mm day^{-1} during the first month after the

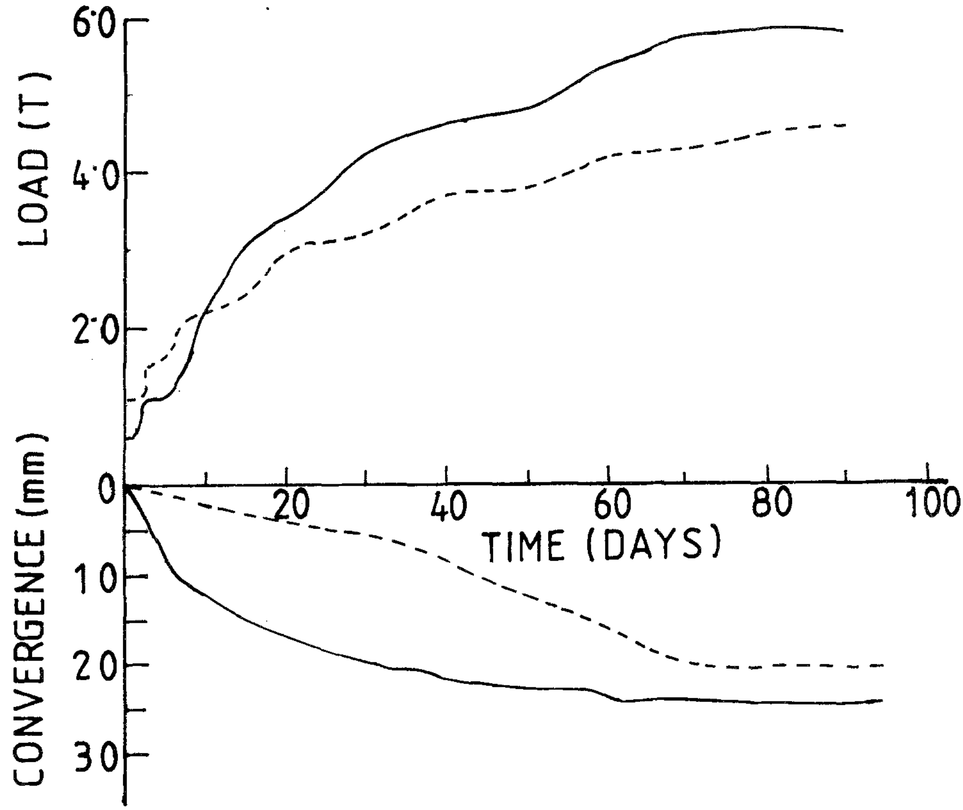


Fig. 2. Load and convergence measurements at Bansra colliery

exposure of the roof and flattened off thereafter. A maximum load of 8.8 tonne m^{-2} was recorded in an unsupported junction (Fig. 3).

Inder Colliery (Western Coalfields Limited)

At Inder colliery, 5B seam (3.5 m in thickness) is being worked at a depth of about 135 m. The method of work is bord and pillar with 3.6 m wide galleries. The seam is dipping at 14° due S 22° W. The roof consists of 0.6–0.8 m coal (the workings being at the bottom of the seam), overlain by sandstone. The rock mass rating (R) of the roof rock was 39 and the rock dry density of the immediate roof was 1.8585 tonne m^{-3}.

It has been observed that a maximum convergence of 23.5 mm took place in an unsupported junction. The initial rate of convergence was 0.45 mm day^{-1} for the first month after the exposure of the roof. The maximum load in an unsupported junction was recorded as 5.0 tonne m^{-2} (Fig. 4).

Parbelia Colliery (Eastern Coalfields Limited)

The 2.1 m thick Sanctoria seam in this mine is worked by bord and pillar system. The seam occurs at a depth of 550 m and has a gentle dip of 1° due S 11° E. The full thickness of the

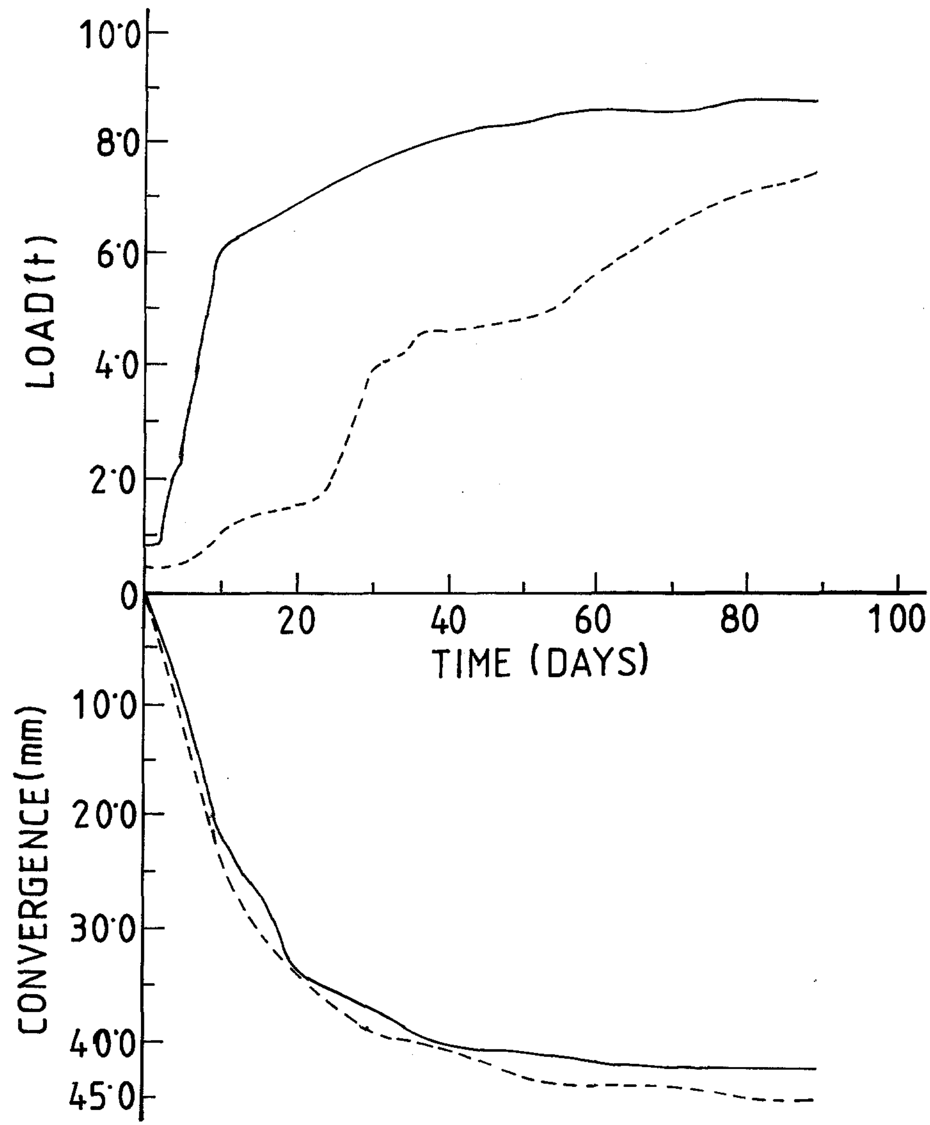


Fig. 3. Load and convergence measurements at Durgapur Rayatwari colliery

seam is extracted with 4.2 m wide galleries. The immediate roof is shale overlain by sandstone. The rock mass rating (R) of the roof rock was calculated to be 39. The rock dry density was 2.004 tonne m^{-3}.

The maximum roof movement of 25.3 mm in an unsupported junction was observed in this mine. The initial rate of closure was 0.55 mm for the first month after the exposure of the roof. The maximum load recorded in an unsupported junction was 6.1 tonne m^{-2} (Fig. 5).

Jamuna 9 & 10 Inclines (South Eastern Coalfields Limited)

In the 9 and 10 inclines of Jamuna colliery, the middle Kotma seam (1.7 m in thickness) is being developed by bord and pillar system at a depth of about 50 m. The beds are gently

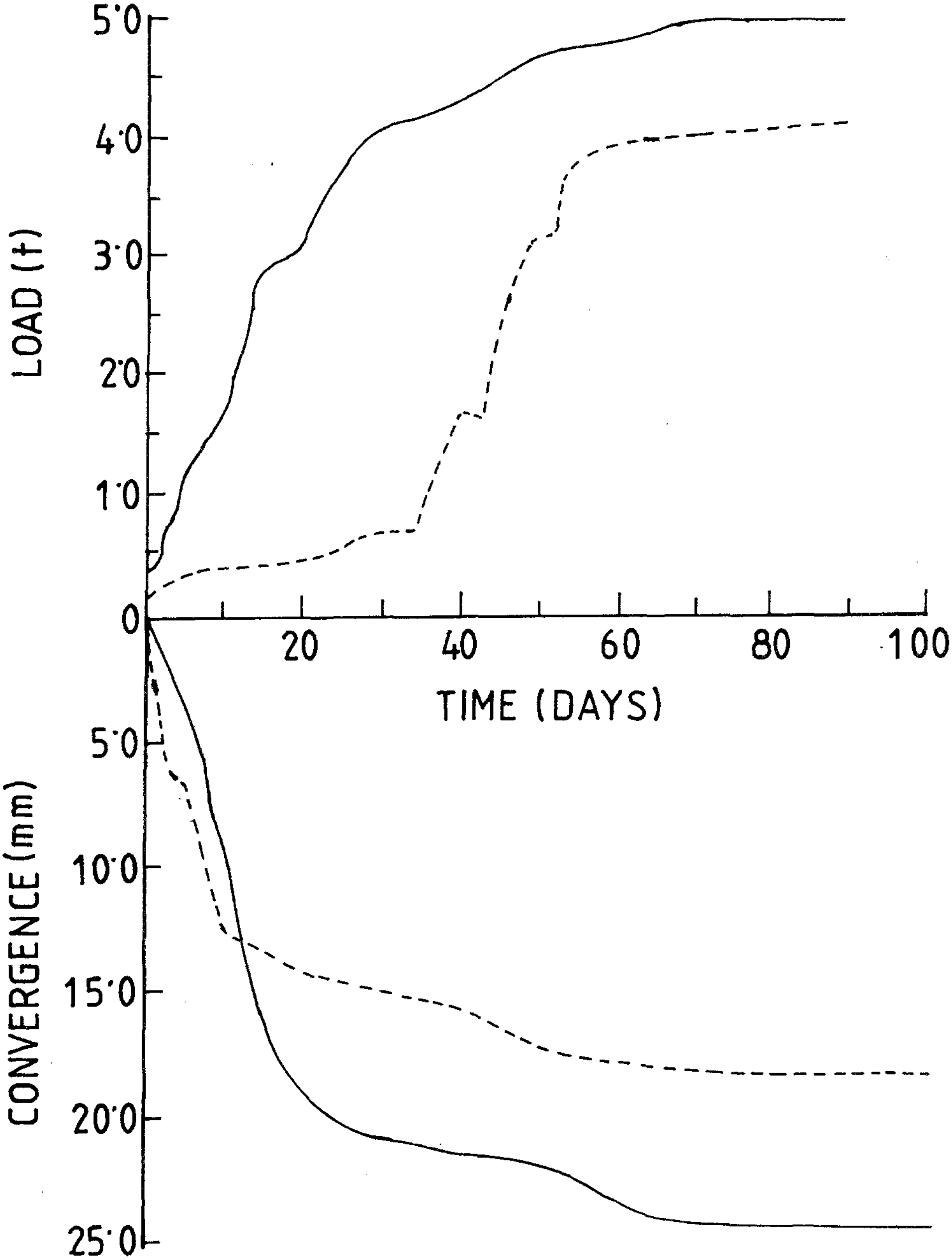


Fig. 4. Load and convergence measurements at Inder colliery

dipping at 2°. The full seam thickness is worked with a gallery width of 3.6 m. The immediate roof consists of laminated sandstone, shale and clay in that order. The rock mass rating (R) for the roof rock was calculated to be 30. The rock dry density of the immediate roof was 1.988 tonne m^{-3}.

The instrumentation was done both in supported and unsupported areas. It was observed

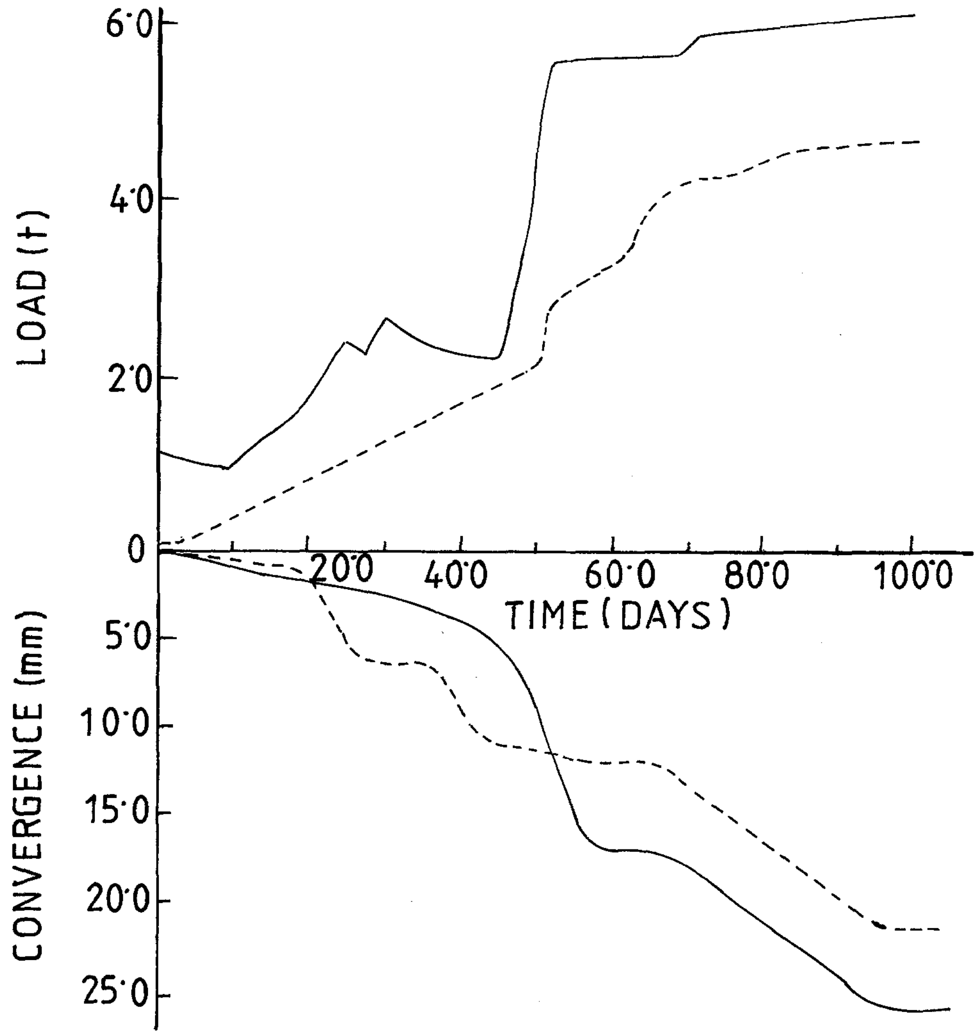


Fig. 5. Load and convergence measurements at Parbelia colliery

that the maximum convergence of 33 mm took place in an unsupported junction with the initial rate of convergence of 0.7 mm per day in the first month after the exposure of the roof. The maximum load recorded was 7.0 tonne m^{-2} in an unsupported junction (Fig. 6).

Godavari Khani-9 Incline (Singareni Collieries Company Limited)

The 3 seam is 11 m thick, and dips nearly 7° due S 53° E. Depth of working is about 130 m. The bottom 3 m of seam is being developed with 4.8 m wide galleries. The immediate roof in the bottom section is four alternating layers of coal and shale. The rock mass rating (R) of the roof was 40 and the rock dry density was 1.862 tonne m^{-3}.

The maximum ground movement in an unsupported junction was found to be 22 mm. The initial rate of convergence was 0.4 mm day^{-1} for the first month after the exposure of the

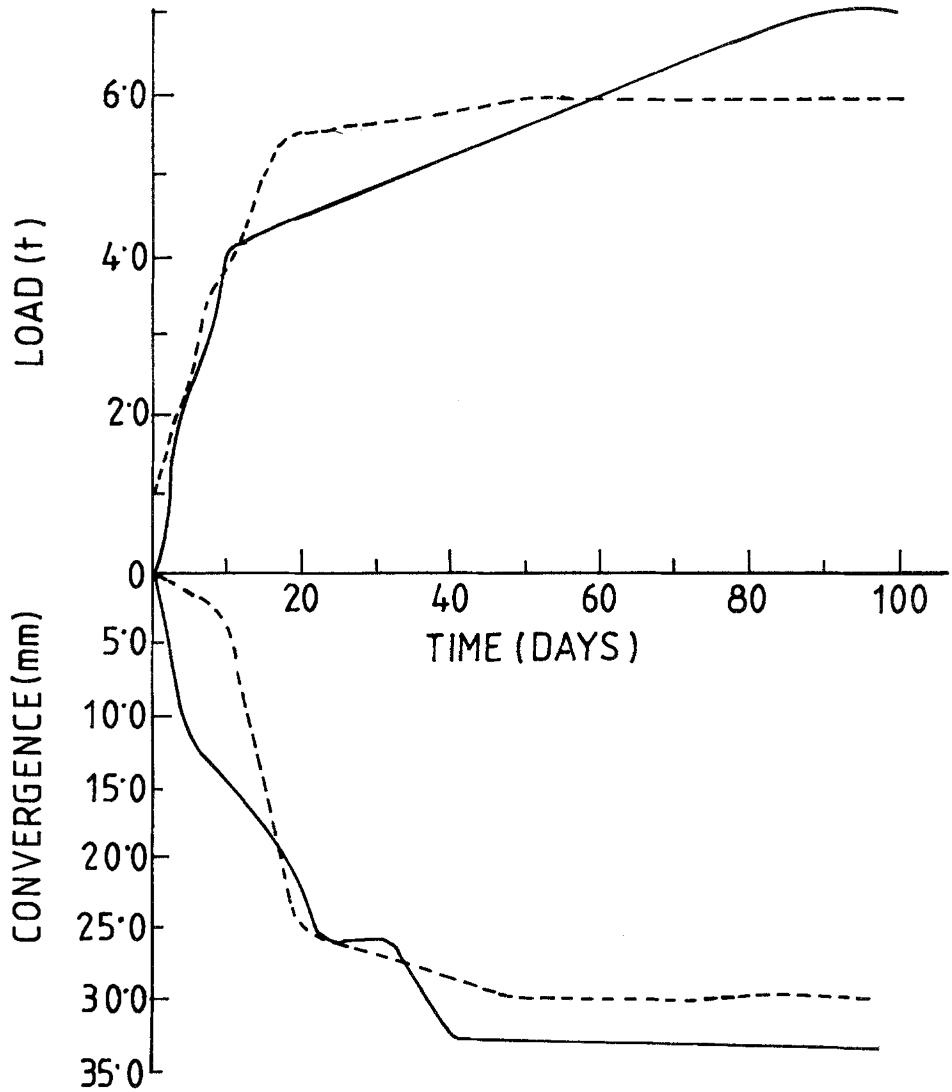


Fig. 6. Load and convergence measurements at Jamuna 9 and 10 Incline

roof. The maximum load was recorded as 5.8 tonne m^{-2} in an unsupported junction (Fig. 7).

Venkatesh Khani-7 Shaft (Singareni Collieries Company Limited)

The top seam is being worked by both the mechanized longwall system and the bord and pillar method. The seam occurs at a depth of about 250 m and dips at 7° due N 59° E. The seam is 11.5 m in thickness and contains a number of shale and mudstone bands of varying thickness. The workings are done at the bottom section of the seam with 4.2 m wide galleries. The rock mass rating (R) of the roof was found to be 33. The rock dry density of the immediate roof was 2.105 tonne m^{-3}.

Strata monitoring observations done in this mine showed a maximum roof movement of

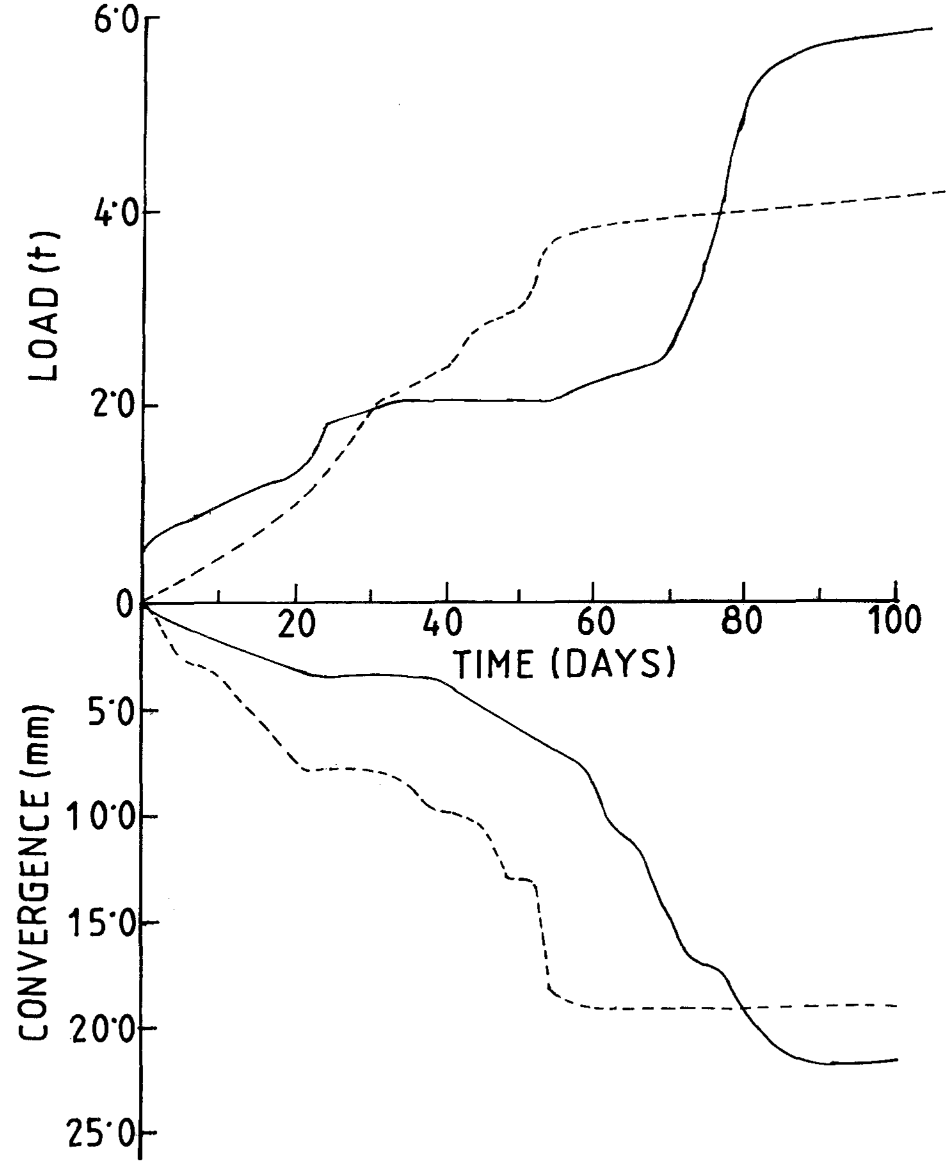


Fig. 7. Load and convergence measurements at Godavari Khani-9 Incline

32 mm in an unsupported junction. The initial rate of convergence was 0.5 mm day^{-1} for the first month of the exposure of the roof. The maximum load recorded was 7.5 tonne m^{-2} in an unsupported junction (Fig. 8).

Analysis of the field data

Roof-floor convergence instrumentation was done in all the eight mines. In each mine more than five convergence stations were installed for measuring the external ground movements.

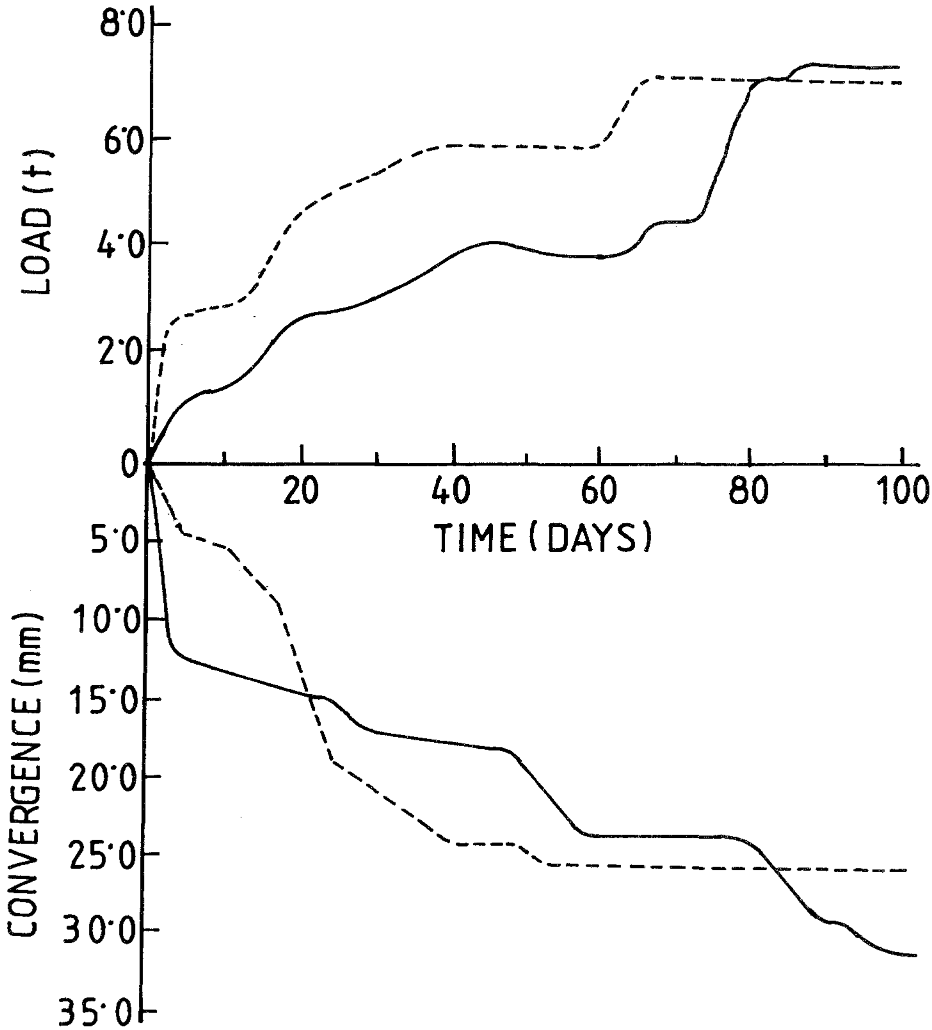


Fig. 8. Load and convergence measurements at Venkatesh Khani-7 Shaft

The roof-floor convergence stations were fixed in an attempt to determine the magnitude and distribution of convergence in freshly exposed roadways and behind the mining face. The convergence instrumentation layout was designed to measure the change in length of an imaginary line connecting a reference point in the roof and floor.

Convergence measurements were taken daily for a period of about 1 month until the convergence rate decreased significantly. In almost all the cases an attempt was made to install the convergence stations at the freshly exposed roofs. The measurements were taken with time as well as with face advance.

The convergence stations were fixed both on the junctions and galleries. The remaining three sides of the junctions were opened only after installing the convergence indicators. The results obtained from roof-floor convergence instrumentations are summarized in the graphs (Figs 1–8). These graphs provide the nature of convergence behaviour obtained at different mines with bord and pillar workings.

Convergence measurements were done both at supported and unsupported junctions. At unsupported junctions when the roof condition deteriorated, temporary supports were provided to stop the collapse of the roof. As an obvious result in all the cases the maximum amount of convergence was recorded at unsupported or temporarily supported junctions. Three possible mechanisms could be hypothesized, which in combination would yield the convergence behaviour. These are roof sag, floor heave and squeezing of coal seams.

It was observed that the maximum closure is a function of various parameters, like roadway span, rock dry density, depth, geological features, groundwater conditions, layers in the roof rock, weatherability of the roof rock etc. As all the parameters except the first two are considered in calculating the rock mass rating (R), hence it could be assumed that the maximum ground movement (C_m) is a function of three major parameters: rock mass rating (R), roadway span (B) and rock dry density (γ).

$$C_m = f(R, B, \gamma) \tag{1}$$

An attempt was made to establish a relation between the maximum ground movement (C_m) in the junction with R, γ and B using statistical regression analyses and the following relation was established:

$$C_m = 40B^{0.5}\gamma^{0.3}(1 - R/100)^3 \tag{2}$$

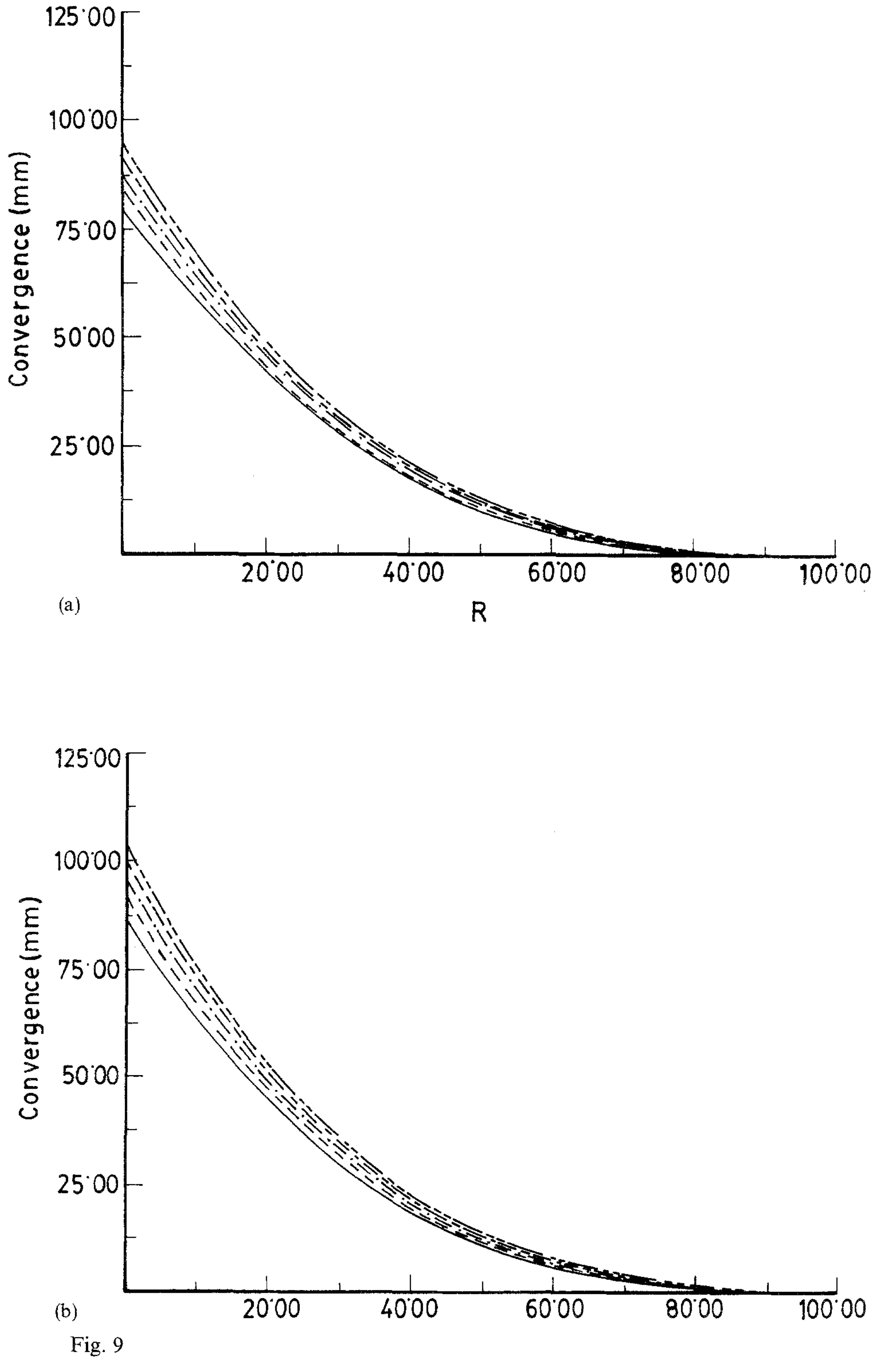
where, C_m is in millimetres, B is in metres, γ is in tonnes per cubic metres.

The nature of the curves of the above Equation 2 is shown in Figs 9a–d for different roadway widths and varying densities of the roof rock. The above relation was tried in several other mines excluding the above mentioned eight mines and a maximum variation of 10% between the calculated value and actual field value was observed. Table 3 gives the values of critical convergence for three different types of roof rocks with R ranging from 0–100.

In all the cases when the convergence has exceeded the value obtained from the above formula collapse of the roof has occurred. So this value can be considered as the critical value to predict the roof failure. If the cumulative convergence value obtained from field instrumentations reaches this value the support density needs to be increased to stop the premature collapse of the roof. On the basis of convergence analysis the excavations are termed as *stable*, *short-term stable* and *unstable*. If after the occurrence of initial convergence the rate of convergence of the excavation ceases and assumes an even decreasing value being less than the critical value, this is termed as *stable* and the existing support system is expected to hold the roof. If after the initial convergence the rate of convergence of excavation remains constant for a period of time and approaches the critical value, it can be termed as *short term stable* and additional supports are to be provided to hold the roof. If after the initial convergence the rate of convergence increases and approaches the critical value at a faster rate, this can be termed as *unstable* and the existing support system has to be changed for a better type of support system.

Rock load in bord and pillar working

Load cells were installed in all the eight mines under study. These were installed both in the galleries as well as in the junctions. Attempts were made to install the load cells at the freshly exposed roofs and the remaining three sides of junctions were opened only after installing the load cells. These were monitored with face advance as well as with time. Daily monitoring of



125·00
100·00
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50·00
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Convergence (mm)
20·00
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(a)
R
125·00
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25·00
Convergence (mm)
(b)
20·00
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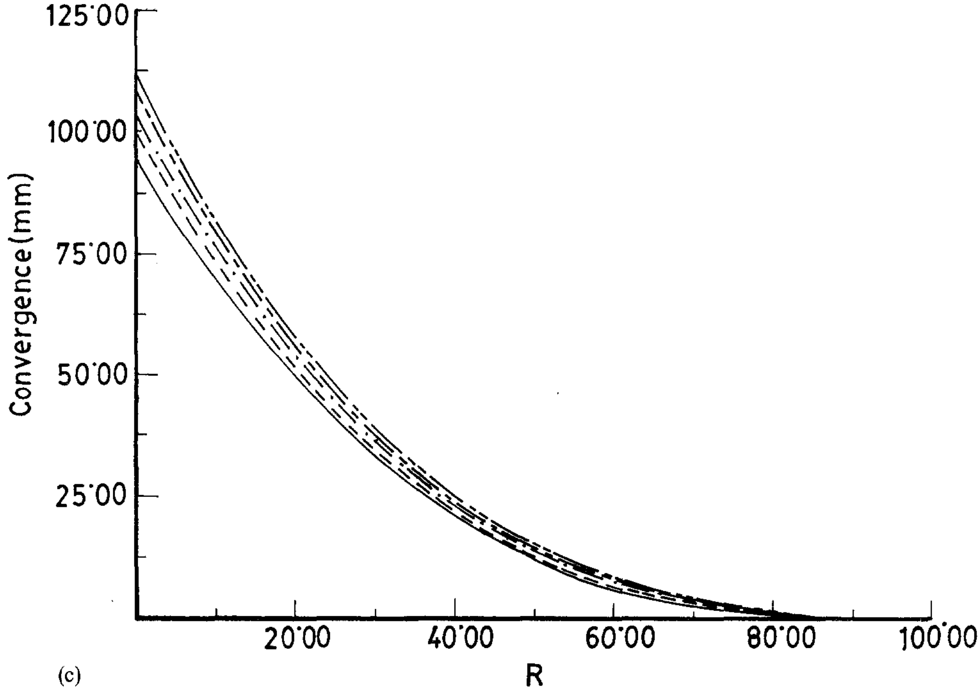
Fig. 9



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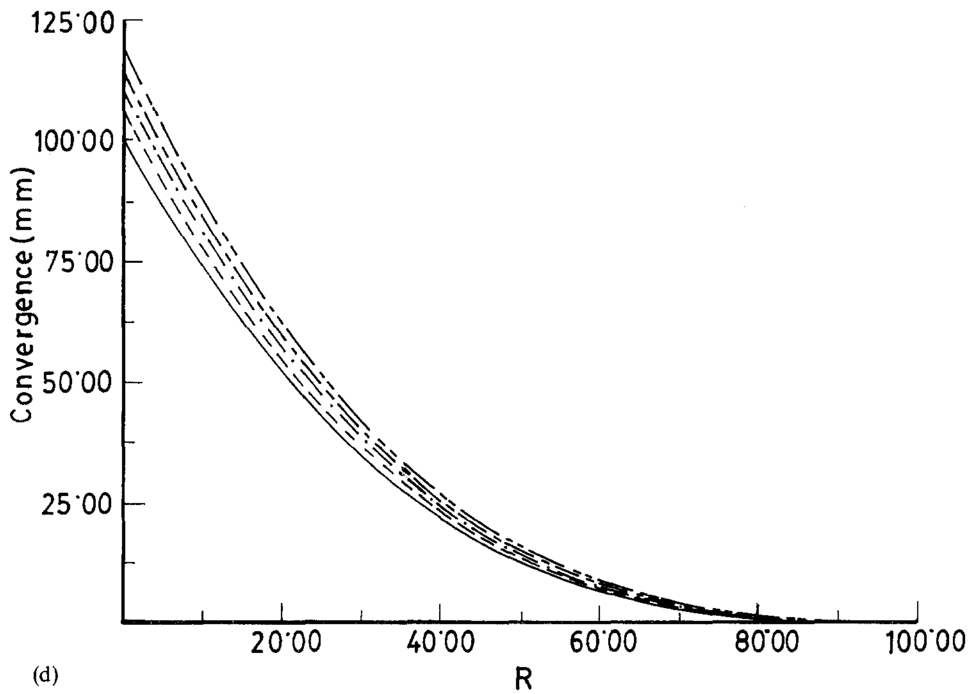


(d)

Figs 9(a–d). Convergence at different rock mass rating (R) with varying densities for a roadway width of (a) 3 m, (b) 3.6 m, (c) 4.2 m and (d) 4.8 m. $\gamma = 1.50$ (——), 1.75 (– – –), 2.00 (— · —), 2.25 (— — —), 2.50 (— — — —) tonne m^{-3}

Table 3. Estimation of critical convergence (mm) for three types of roof rocks (gallery span = 4.2 m)

R	Sandstone ($\gamma = 2.5$ tonne m^{-3})	Shale ($\gamma = 2.2$ tonne m^{-3})	Coal ($\gamma = 1.54$ tonne m^{-3})
0	110.9	106.3	94.5
10	80.8	77.5	68.9
20	56.8	54.4	48.4
30	38.0	36.5	32.4
40	24.0	23.0	20.4
50	13.9	13.3	11.8
60	7.1	6.8	6.0
70	3.0	2.9	2.6
80	0.9	0.9	0.8
90	0.1	0.1	0.1
100	0	0	0

the load cells was done for the first month after the exposure of the roof and with the stabilization of the roof the frequency of monitoring was decreased. It was observed that the load increases sharply within the first 10 to 15 days after the exposure of the roof and then gradually the rate decreases and ultimately it ceases to increase further after 3 to 4 months. A statistical regression model was tried to correlate the maximum load (P) encountered with other influencing parameters: roadway span (B), rock dry density (γ) and rock mass rating (R) and the following relation could be established:

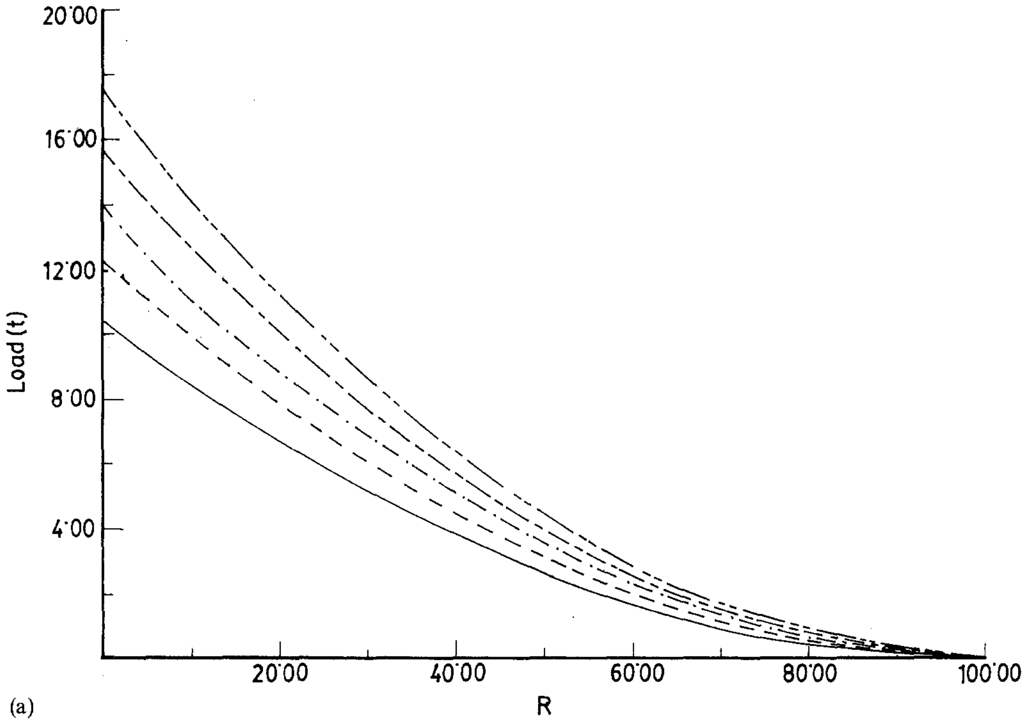
$$P = 5B^{0.3}\gamma(1 - R/100)^2 \qquad (3)$$

where, P is in tonnes per square metres. The graph of Equation 3 is shown in Figs 10a–d for different roadway widths and varying densities of the immediate roof rock. Table 4 gives the values of rock load for three different types of roof rock with R ranging from 0–100.

Conclusion

The work described in this paper is based on case studies in eight different coal mine locales representing varying depths and strata condition. An attempt has been made to correlate the critical convergence with rock mass rating (R), rock dry density (γ) and roadway span (B). Since the relation has been tried successfully in various coal mines in India, it is expected that the relation will help in determining the roof condition and that roof falls could be predicted before-hand in development headings in bord and pillar workings. On the basis of convergence the roofs are categorized as *stable*, *short-term stable* and *unstable*. For optimum support design strategies and to avoid premature collapse of the roof the above categorization is expected to be very helpful. The optimum design of support system calls for an appropriate knowledge of rock pressure distribution. An empirical relation has been established to calculate the expected rock load in the development heading of bord and pillar workings.

No such work has been done previously to estimate a critical value of convergence to predict roof falls. Unal (1983) has established a relation with the rock load, but for Indian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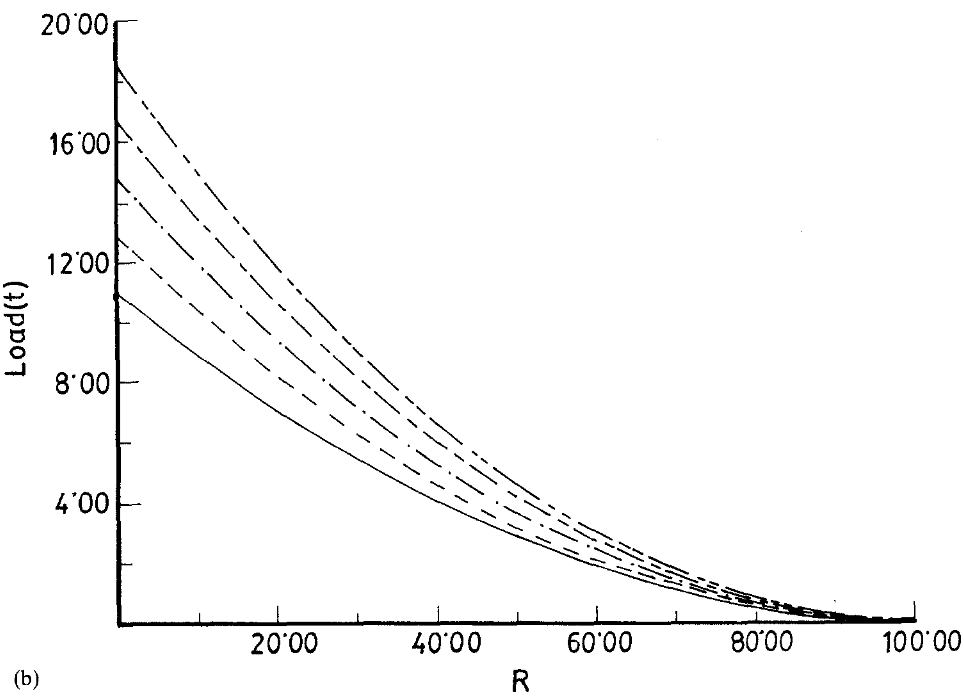
(a)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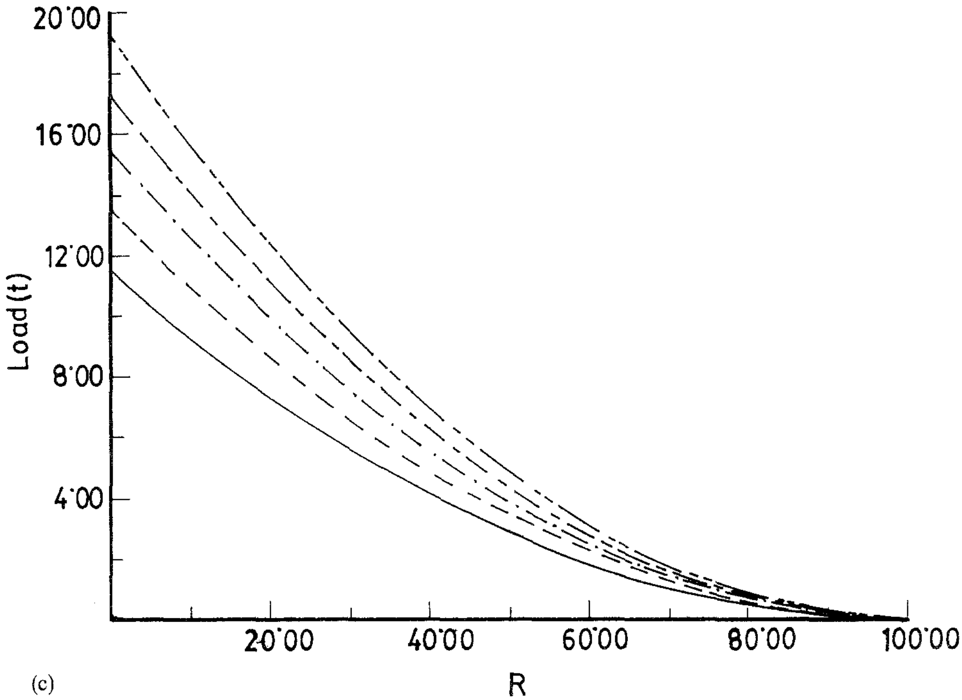
(b)

Fig. 10



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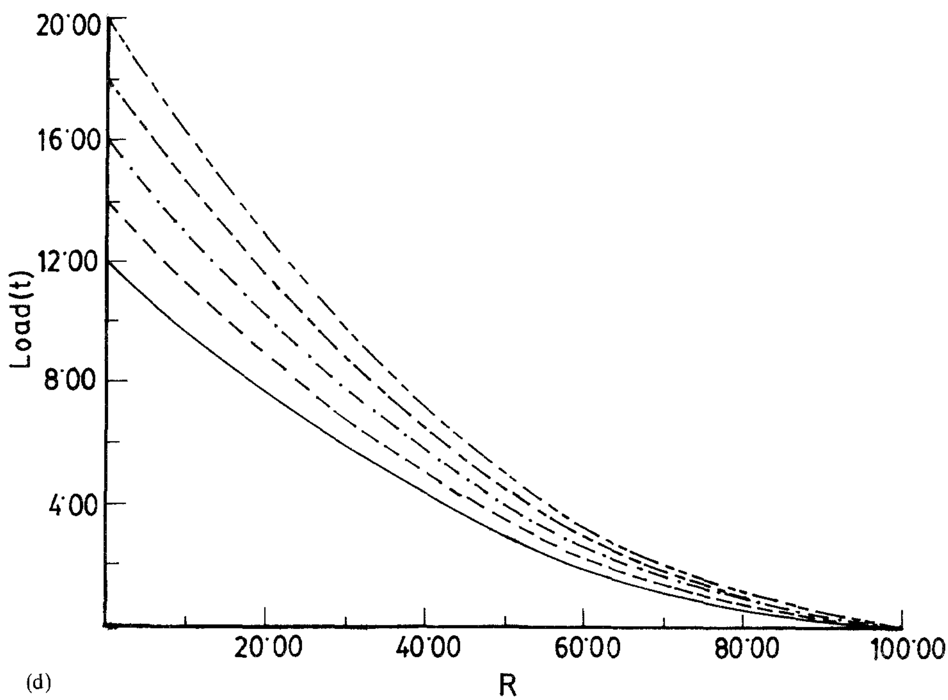

(d)

Figs 10(a–d). Rock load at different rock mass rating (R) with varying densities for a roadway width of (a) 3 m, (b) 3.6 m, (c) 4.2 m and (d) 4.8 m. $\gamma = 1.50$ (——), 1.75 (– – –), 2.00 (—·—), 2.25 (— — —), 2.50 (— – – —) tonne m^{-3}

Table 4. Estimation of rock loads (tonnes), for three types of roof rocks (gallery span = 4.2 m)

R	Sandstone ($\gamma = 2.5$ tonne m^{-3})	Shale ($\gamma = 2.2$ tonne m^{-3})	Coal ($\gamma = 1.54$ tonne m^{-3})
0	19.23	16.92	11.84
10	15.57	13.70	9.59
20	12.30	10.83	7.58
30	9.42	8.29	5.80
40	6.92	6.09	4.26
50	4.81	4.23	2.96
60	3.08	2.71	1.89
70	1.73	1.52	1.07
80	0.76	0.68	0.47
90	0.19	0.17	0.12
100	0	0	0

geomining conditions the relation was not found to be suitable. The empirical relation described in this paper is being used successfully at different coal mines in India for optimum design strategies and it is found to be very useful.

The above formulae for critical convergence and rock load is based on studies for roadway widths between 3 m and 4.8 m, rock mass ratings (R) ranging from 23 to 55 and in all the cases the working heights were between 2 m and 3 m; hence the formulae described in this paper may be most suitable for the above ranges.

Acknowledgements

The work described herein is based on an S&T project funded by Department of Coal, Government of India and supported by Central Mine Planning and Design Institute of India. Dr N.M. Raju, Dr V. Venkateswarlu of National Institute of Rock Mechanics, Kolar and Mr B.N. Mishra and Mr R. Sahu of Central Mining Research Station, Dhanbad helped us in instrumentation. Dr B. Singh, Director, Central Mining Research Station, Dhanbad, accorded permission to publish the work.

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