Critical analysis of process cycle by numerical modelling for faster development of drives in hard-rock underground mine – a case study

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To extract deep-seated metallic minerals, an underground mine needs to be developed at a faster rate to access the ore body safely. There are various techniques to break the rock but still drilling and blasting is considered the cheapest. Therefore, an effort was made to re-engineer the mine development cycle time in order to achieve high advance rate. This study highlights the requirement of numerical simulation in rock excavation for its stability and design through the re-engineered drill and blast operations. Implementation of the findings of numerical modelling and deployment of advanced drilling equipment helped reduce the total drilling time and overall cycle time by 30.70% and 15.90% respectively, in the two cases of drilling length considered, i.e. 3.4 and 4.0 m. Subsequently, in a further modified process, where the supporting activities were avoided till the third advance, there was significant improvement in the process cycle time by 43.10% for 3.4 m drilling and 39.30% for 4.0 m drilling length. We found that 15 m advance per day can be achieved by the deployment of double boom jumbo for drilling length of 3.4 m and drive size of 4.5 m × 3.0 m.

Keywords: Drill and blast, linear excavations, mine development, numerical modelling, process cycle time.

THE excavations of ore drives and drifts are common features in any metal mine. The room and pillar mining method needs a large quantity of drivages (4.5 m × 3.0 m) in the ore body to make available more stope blocks for production. Declines, ramps, loading bays, cross-cuts and drifts are the additional requirements for permanent mine construction. The vital component in a drive is the absence of initial free faces. The burn cut (parallel holes with three numbers of Reamers holes) in blast design is suitable for any large-sized of drift/drive excavation with proper explosives, initiation sequences, etc. It can give considerable amount of pull. Further, optimizing the process cycle time, drilling, charging, mucking and ground support, the time can be reduced by mechanization and other factors, such as operator skills, face conditions, etc. resulting in an re-engineered hybrid model, which is an optimum option.

It is difficult to find open literature detailing advance rates for drill and blast development projects. Twelve such case studies were found and are listed here (Table 1). The median and average advance rate for the 12 cases is 7.0 and 6.80 m/day respectively. It is important to note that the data contains both single and multiple heading development case studies.

Even though most underground development is less than 50 m^2, the possibility exists to undertake ground support in parallel with development if equipment could be designed to work within the limited space available. The same is the case for the mine discussed here, where the size of the production faces is minimum 4.5 m × 3.0 m and maximum 5.0 m × 3.0 m for permanent roadways like decline and loading bays. Based on the mining case studies (best case 8.90 m/day), 10.40 m/day could be considered a high average development rate.

Based on the benchmarking studies and by drill and blast simulations, Canadian mining industry research organization: Sudbury (CAMIROs), it has been possible to estimate a theoretical limit to advance of approximately 19 m/day. This limit assumes that it is theoretically possible to achieve the several technical developments and advances[1].

Mechanized excavation using a tunnel boring machine (TBM) is highly influenced by tunnel diameter, geological conditions, etc. TBM advance rates have been evaluated using the Norwegian University of Science and Technology (NTNU) prognosis method for different ground conditions[6]. Table 2 summarizes the results of advance rates for various rock types. On an average, TBMs can be expected to exceed drill and blast development rates (NTNU prognosis 10.40 m/day) for good (23.70 m/day), fair (17 m/day) and poor (18 m/day) ground conditions. In the case of very hard and abrasive rocks, the average advance rates were similar, 10.40 and 10 m/day for drill and TBM methods respectively. The assumed utilization factors were found to significantly impact these results.

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Table 1. Drill and blast benchmark case studies[1]

Case study	Country	Average advance rate
Common infrastructure project – PT Freeport[2]	Indonesia	9.0 m/day (270 m/month)
Craviale tunnel[3]	Italy	5.5 m/day (165 m/month)
Kidd Creek mine[4]	Canada	5.3 m/day (159 m/month)
Holt McDermott mine[4]	Canada	7.2 m/day (216 m/month)
Creighton mine[4]	Canada	5.0 m/day (150 m/month)
Brunswick mine[4]	Canada	5.8 m/day (174 m/month)
Dome mine[4]	Canada	7.4 m/day (222 m/month)
Musselwhite mine[4]	Canada	8.9 m/day (267 m/month)
Birchtree mine[4]	Canada	4.1 m/day
Stobie mine[4]	Canada	7.9 m/day (237 m/month)
Golden giant mine[4]	Canada	6.7 m/day (202 m/month)
Golden grove catalpa decline[5]	Australia	8.3 m/day (249 m/month)

Table 2. Drive development rate under different rock conditions[1]

	Rock type 1	Rock type 2	Rock type 3	Rock type 4
General description of tunnel boring machine (TBM)	Very hard and abrasive rock RMR = 90	Good RMR = 70	Fair RMR = 50	Poor RMR = 30
Penetration rate (m/h)	1.2	2.84	2.91	3.09
Advance rate (m/h)	0.6	1.42	1.0185	1.0815
Advance rate (m/day) of TBM	10.0	23.7	17.0	18.0

RMR, Rock mass rating.

Objective

After a review of faster development methods for very hard to hard-rock underground metal mines, following are the objectives of the present study:

- Determination of the theoretical limits for drill and blast advance rates of a mine in Kadapa district, Andhra Pradesh, India, which is very hard rock-type.
- To identify opportunities for improvement in drill and blast advance rates of the mine, preferably in the process cycle time.
- Identify opportunities for step change improvement achievable in underground development rates using new equipment and methods through scientific approaches like numerical modelling.

To meet these objectives, it is necessary to compare the theoretical limits for drill and blast advance rates of the mine whose uniaxial compressive strength (UCS) is in the range 350–400 MPa.

Methodology

Re-engineering opportunities in drill and blast advance of the mine in the process cycle time

(a) Drilling, blasting and ventilation along with water spraying could be done simultaneously to reduce the time.

(b) Development of blind face and rock-bolt drilling along with full column grouting could also be done simultaneously to reduce the time.

(c) Development and rock-bolt drilling along with installation of friction bolts with hydro-pump could be done simultaneously to further reduce the time.

(d) The above options followed by charging of the face with three more numbers of blaster helper lead to reduction of charging time by 45 min.

(e) The mucking time was reduced by 40% by deploying two sets of low-profile dump truck (LPDT) at the face.

(f) Drilling at the face could be operated by double boom and long round drilling such as 4.2 m.

The best re-engineered model in drill and blast advance for the mine in Kadapa district, Andhra Pradesh, India

The major process, which could be avoided in the process cycle, is drilling rock bolts and grouting in either form as discussed above. The second process could be deployment of double boom drill jumbo with increased length of round, i.e. 4.2 m. The third process that could be modified is mucking, by the deployment of two sets of LPDT. The fourth process is charging and blasting by increasing the number of blasting helpers to six. The fifth process that could be done simultaneously with face drilling is drilling of rock bolts by bolter, up to just behind the drill jumbo. All the above modifications were

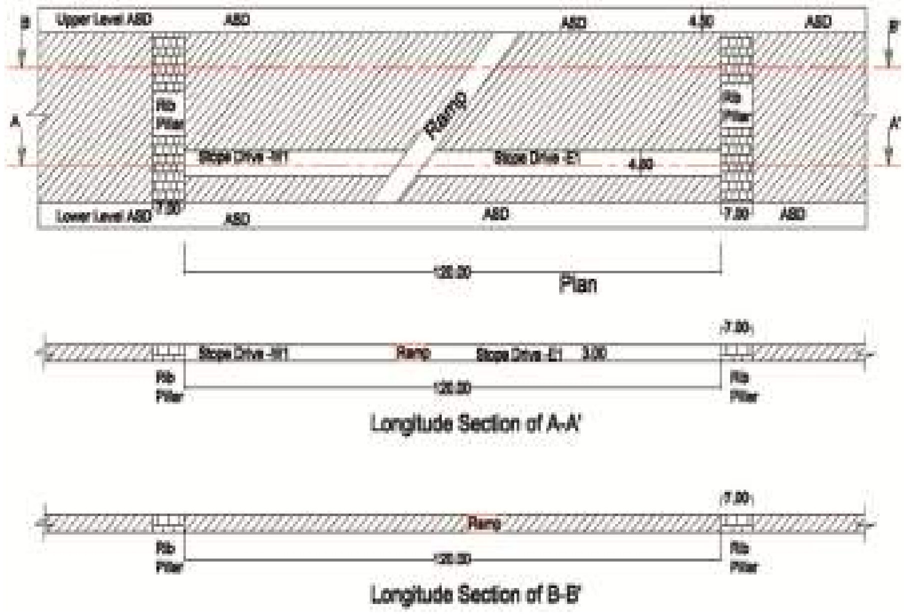

Figure 1. Plan and section of the stope block considered for numerical modelling.

Table 3. Rock properties followed in the numerical modelling

Rock mass	Tensile strength (MPa)	Young's modulus (MPa)	Poisson's ratio	Cohesion (MPa)	Friction angle (°)
FW-Lode	10	5530	0.25	10	35
HW-Lode	10	5530	0.25	8	35
Parting	10	5530	0.25	10	35
Shale	5	4320	0.29	8	35
Shale-Week	10	3110	0.30	5	42
Limestone	15	9950	0.25	12	32
Soil (surface)	0	100	0.032	0.5	30
Back fill	0	1	0.032	1	30

suggested after a detailed study in the said mines that helped achieve faster mine development advance rate.

The first (a) and fifth (e) processes mentioned above need to be examined by scientific studies through numerical modelling in a worst-case simulation and confirmed for the number of advance or rounds of blast that can be done without drilling rock bolts and grouting.

Numerical modelling

Section A–A' in Figure 1 was considered for numerical modelling.

Observations of numerical modelling (considering the effect of blasting)

In order to ascertain the stability of stope drive and the minimum advance before installing rock bolt support system, a two-dimensional numerical modelling was performed. For this, a longitudinal section of the panel was simulated (Figure 1). This modelling analysis presents the results of simulation under various stages of stope drive advance and blasting effect on the face surroundings. Actual lithology up to the surface was considered in the model. The maximum depth of mining (200 m) was also considered for the worst-case simulation. All the material properties of the various rock masses that formed the model geometry were obtained from laboratory testing of the representative samples collected from the mine. Table 3 presents results of the geo-mechanical properties tested in the laboratory. Table 4 shows the results obtained in the form of mining-induced stress, deformations and factor of safety (FoS) for various stages of excavation.

Three parameters, i.e. deformation (Figure 2 *a*–*e*), FoS (Figure 3 *a*–*e*) and induced stress (Figure 4 *a*–*e*) were examined for stability of the roof and sides of the stope drive and the first opening at ramp by the proposed blasting method and sequence of operation by numerical modelling. The proposed advance per blast achieved by the

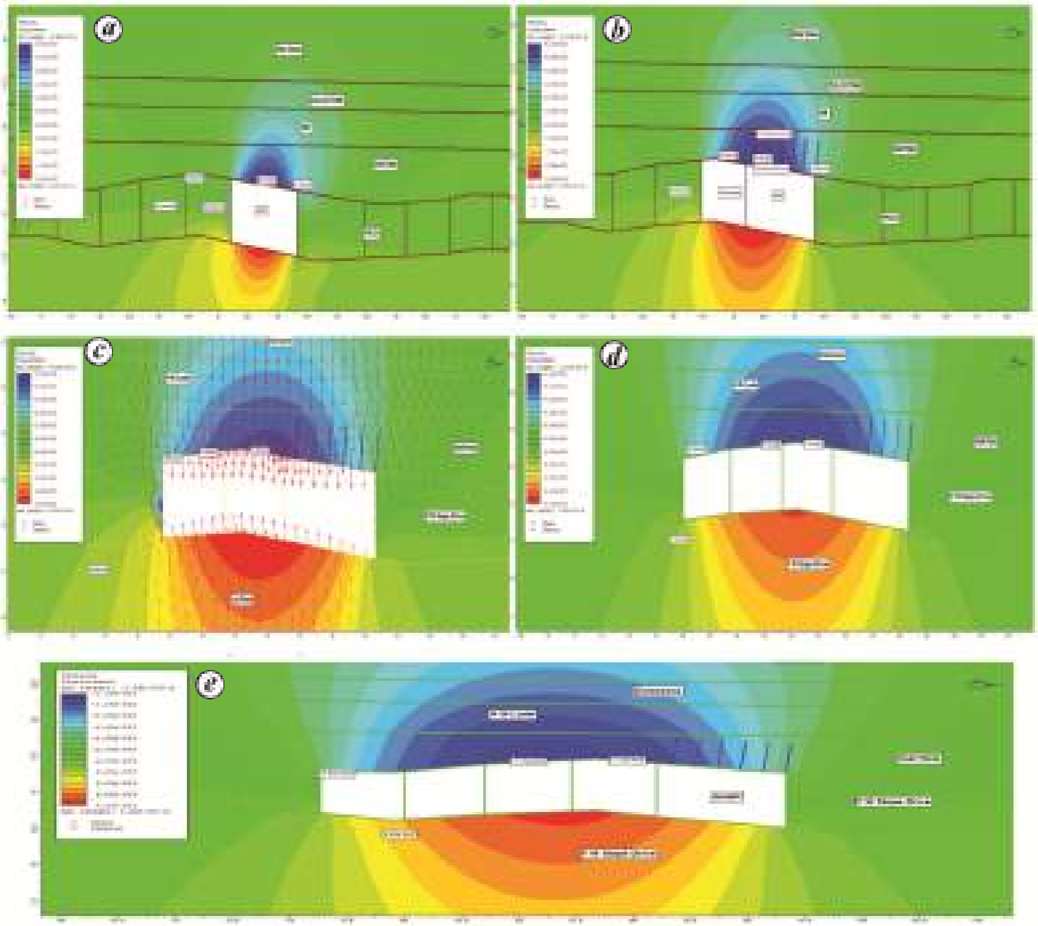

Figure 2. Deformation simulations. ***a***, Case (i): Displacement in ramp only. ***b***, Case (ii): Displacement in ramp + one advance (blast-1). ***c***, Case (iii): Displacement in ramp + two advances (blast-2). ***d***, Case (iv): Displacement in ramp + three advances (blast-3). ***e***, Case (v): Displacement in ramp + four advances (blast-4).

Table 4. Summary of model analysis results

Opening stage	Mining-induced stress (MPa)	Deformations (mm)	Factor of safety values	Remarks
Ramp only	44	3.0	1.6–5	Static
Ramp + one advance	30	4.0	1.6–5	Dynamic (considering blasting effect from this stage)
Ramp + two advances	32	8.4	2.0	Stress concentrations at the corners, yielding near the face
Ramp + three advances	33	10.01	1.89–2.21	Rock-mass yielding near the face
Ramp + four advances	34	12.0	1.28–2.53	More yielding near the surface and formation of stress relaxation arch
Ramp + more than four advances + bolts	37–38	18	1.89–4.11	Reduced yielding area and stress relaxation arch supported to contain deformations and failure

method appears to be optimum and the roof was stable during excavation of the drive until the third slice (Figures 2 *d*, 3 *d* and 4 *d*), even without the rock bolt support. However, roof strata near the face was observed to be yielding due to the effect of the blast, although the extent of yielding was limited to a few centimetres from the face.

FoS values near the face were observed to be low compared to the other places. It was observed that deformations in the roof progressively increased, particularly at the centre roof for every advance, and it reached more than 10 mm after the third advance (Figure 2 *d*). Subsequently, formation of stress relaxation arch in the roof was observed after the third advance (Figure 3 *d*), which may lead to roof failure in the absence of reinforcement in the form of rock bolts. The maximum mining induced stress observed was of the order of 34 MPa for the fourth advance (Figure 4 *e*). Based on the above results, we

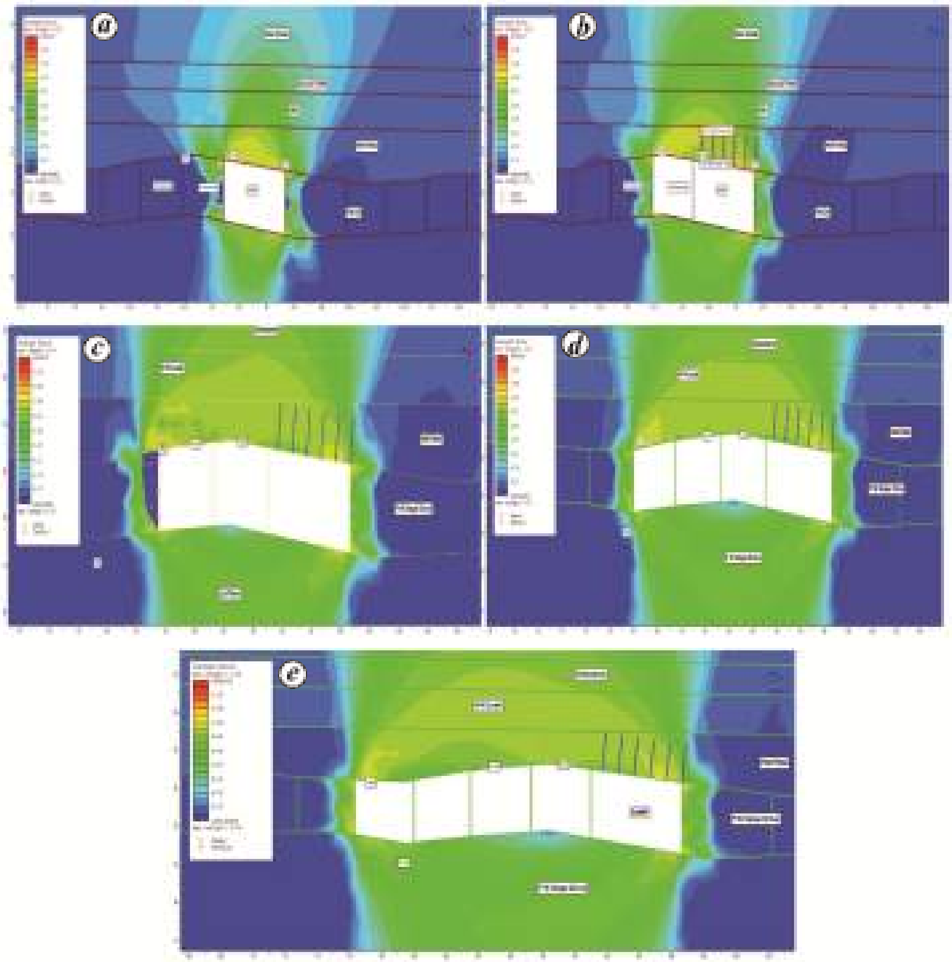

Figure 3. Factor of safety (FoS) around the opening. ***a***, Case (i): Ramp. ***b***, Case (ii): Blast-1 (advance-1). ***c***, Case (iii): Blast-2 (advance-2). ***d***, Case (iv): Blast-3 (advance-3). ***e***, Case (v): Blast-4 (advance-4).

Table 5. Cycle time for 3.4 m depth drilled with double boom jumbo (Sandvik DD422i)

	Drive size = 4.5 × 3.0 m Duration
Drilling	
Time taken for drilling 2 nos of 3.4 m hole, including positioning and collaring of hole	1 min 25 sec
Time taken for drilling 36 nos of holes	22.5 min
Time taken for drilling 1 no. of relief hole (89 mm)	5.0 min
Time taken for drilling 3 nos of relief holes	15 min
Total time for drilling	37.5 min
Charging and blasting	40 min
Re-entry/fume clearance	15 min
Water spraying and loose dressing	30 min
Mucking (1 + 2) set	90 min
Total time taken to handle 120 t by LHD	35 min
Total time taken to handle 120 t by LPDT (22.5 t capacity)	55 min
Rock-bolt drilling	
Time taken for drilling 1 no. of rock-bolt 2.0 m length	3 min
Time taken for drilling 15 nos of rock-bolt 2.0 m length	45 min
Total time for rock-bolt drilling	45 min
Grouting	
Time taken for 1 no. of rock-bolt grouting	5 min
Time taken for 15 nos rock-bolt grouting	75 min
Bottom cleaning and face preparation	
Bottom cleaning and face blow	30 min
Total cycle time	363 min

Manpower utilized for water spraying, loose dressing and grouting is the same. LPDT, Low Profile Dumper Truck.

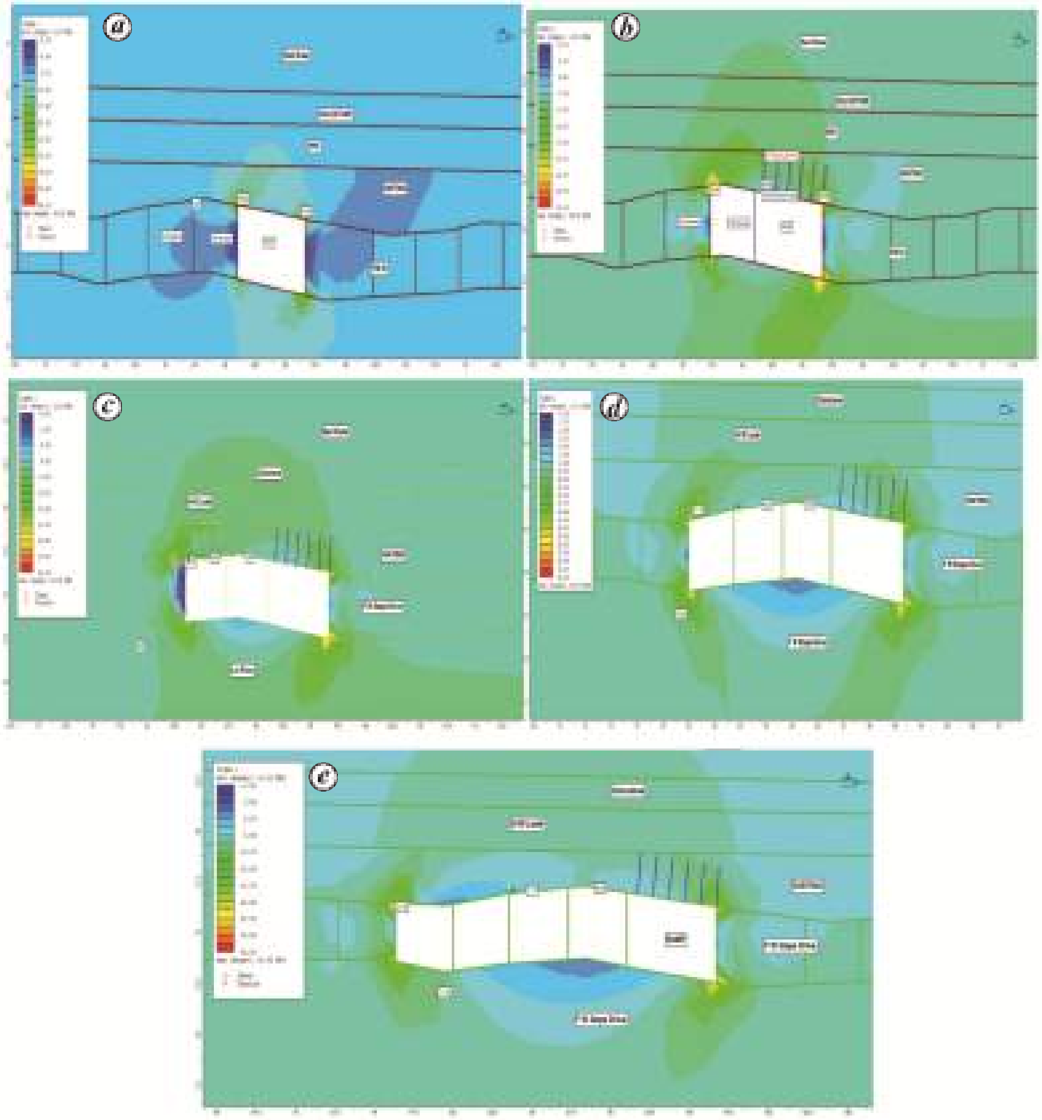

Figure 4. Mining-induced stresses around the opening. ***a***, Case (i): Mining-induced stress of sigma-1 for ramp. ***b***, Case (ii): Mining-induced stress of sigma-1 for blast-1. ***c***, Case (iii): Mining-induced stress for blast-2. ***d***, Case (iv): Mining-induced stress for blast-3. ***e***, Case (v): Mining-induced stress for blast-4.

decided to provide the rock-bolt support from the third advance to avoid roof failure (Figure 5).

Case-I (only ramp)

For better visibility, part of the model containing the representative mining scenario was zoomed in to show the condition of the pillars. Figure 4 *a* shows the mining-induced stresses and Figure 3 *a* the strength factor around pillars after all the ramps within this panel developed. It can be observed from the results that maximum stress induced due to ramp opening was of the order of 44 MPa. Deformations due to development of the ramp with rock bolt support is shown in Figure 2 *a*, which is of the order of 3.0 mm at the centre of the roof. FOS values around the ramp opening were 1.58–2.28 near the face and 5–6 on the sides even without any supports installed.

Case-II (ramp and first advance)

It is also worth mentioning here that the blasting effect has also been simulated from this case onwards. It can be observed from Figure 4 *b* that the maximum stress induced due to this opening was of the order of 30.0 MPa. The FOS values around the opening were 1.58–2.28 near the roof and face, and 5–6 on the sides. Here supports were installed only in the ramp. Also, displacements around the opening were of the order of 4.0 mm at the centre of the roof. The effect of blasting was seen to be limited near the face. Also, the rock bolts were loaded lightly to their minimum capacity.

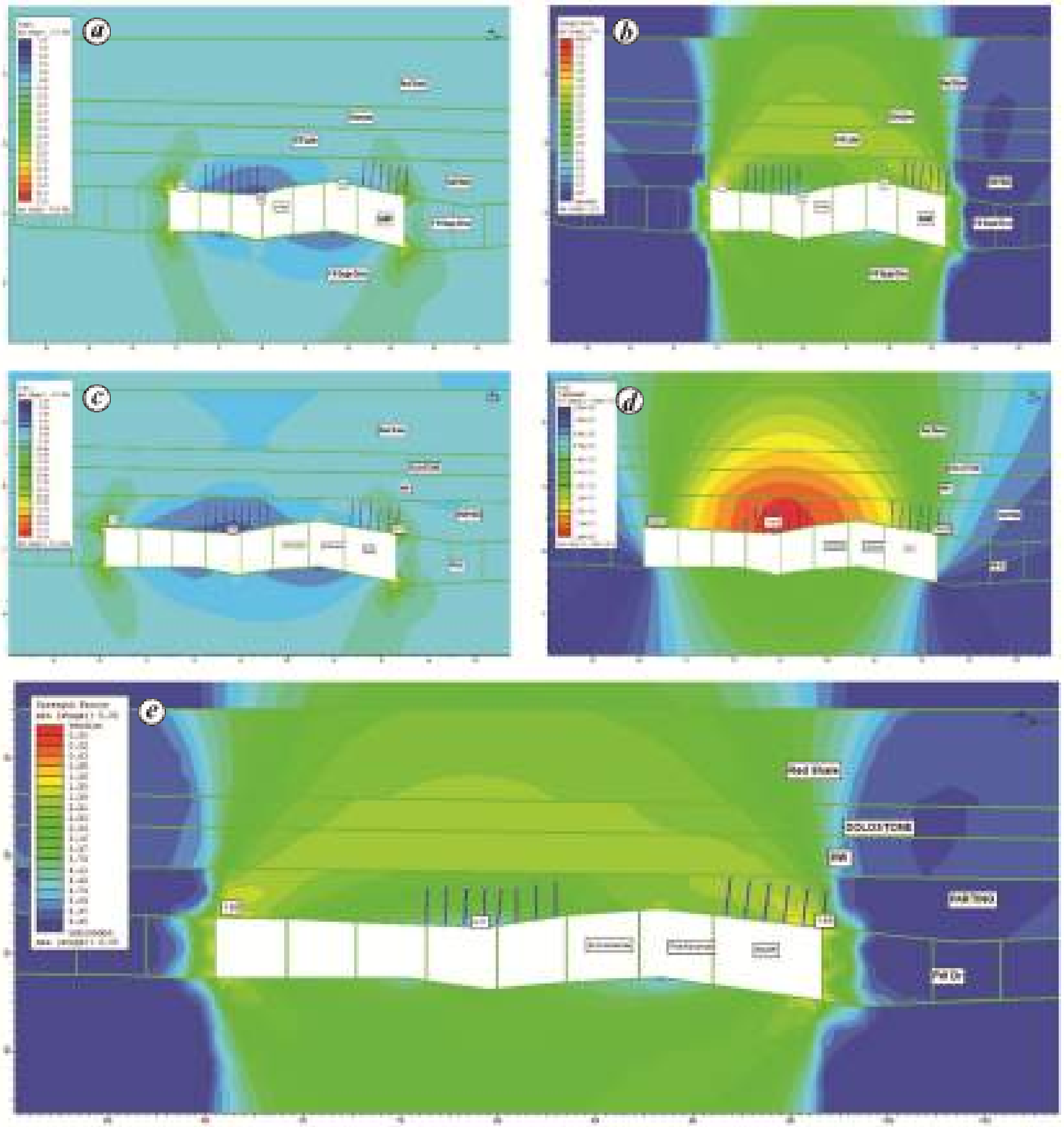

Figure 5. Final recommended mining process cycle. ***a***, Final case mining-induced stress of recommended model. ***b***, Final case FOS of recommended model. ***c***, Final recommended mining-induced stress model. ***d***, Final recommended case and its deformation. ***e***, FOS recommended process cycle for mine.

Case-III (ramp and second advance)

Figure 4 *c* shows the mining induced stresses and Figure 3 *c* the strength factor developed around the ramp and second advance of 3.6 m. It can be observed from the results that the maximum stress induced due to this opening was of the order of 32.0 MPa, which was concentrated at the corners of the opening. FOS values around the opening were less than 2.0 near the roof and face. It could also be observed that the area near the face had started yielding, although not failing. Also, displacements around the opening increased to 8.4 mm at the centre of the roof. The effect of the blasting could be seen in the form of yielding near the face.

Case-IV (ramp and third advance)

Figure 4 *d* shows the mining-induced stresses and Figure 3 *d* the strength factor developed around the ramp and third advance of 3.6 m. It can be observed from the results that the maximum stress induced due to this opening was of the order of 33.0 MPa, which is almost the same as in the previous case. This shows that the stress induced due to mining of the third slice is almost constant. The FOS values around the opening were 1.89–2.21 near the roof and face. It could also be observed that the area near to face had started yielding due to the blasting effect, although not failing. Also, displacements around opening had increased to 10.01 mm at the centre of the roof. The

Table 6. Cycle time for 4.0 m depth drilled with double boom jumbo (Sandvik DD422i)

	Drive size = 4.5 × 3.0 m Duration
Drilling	
Time taken for drilling 2 nos of 4.0 m hole, including positioning and collaring of hole	1.40 min
Time taken for drilling 36 nos of holes	25.20 min
Time taken for drilling 1 no. of relief hole (89 mm)	6.0 min
Time taken for drilling 3 nos of relief holes	18 min
Total time for drilling	45 min
Charging and blasting	50 min
Re-entry/fume clearance	15 min
Water spraying and loose dressing	45 min
Mucking (1 + 2) set	105 min
Total time taken to handle 135 t by LHD	40 min
Total time taken to handle 135 t by LPDT	65 min
Rock-bolt drilling	
Time taken for drilling 1 no. of rock-bolt 2.0 m length	3 min
Time taken for drilling 15 nos of rock-bolt 2.0 m length	45 min
Total time for rock-bolt drilling	45 min
Grouting	
Time taken for 1 no. of rock-bolt grouting	5 min
Time taken for 15 nos of rock-bolt grouting	75 min
Bottom cleaning and face preparation	
Bottom cleaning and face blow	30 min
Total cycle time	410 min (06 h 50 min)

Manpower utilized for water spraying, loose dressing and grouting is the same. LPDT, Low Profile Dumper Truck.

Table 7. Supporting process cycle time for only rock bolting and grouting in both cases of face drilling (3.4 and 4.0 m)

Supporting cycle time double boom drill jumbo (one boom drills face and other does rock bolting) for 4.5 × 3.0 m drive size	
Rock-bolt drilling	
Time taken for drilling 1 no. of rock-bolt 2.0 m length	3 min
Time taken for drilling 15 nos of rock-bolt 2.0 m length	45 min
Total time for rock-bolt drilling	45 min
Grouting	
Time taken for 1 no. of rock-bolt grouting	5 min
Time taken for 15 nos of rock-bolt grouting	75 min
Total cycle time	120 min (2 h)

Manpower utilized for water spraying, loose dressing and grouting is the same.

effect of the blasting could be seen in the form of yielding near the face only.

Case-V (ramp and fourth advance)

It can be observed from the results shown in Figure 4 *e*, that the maximum stress induced due to this opening was of the order of 34.0 MPa, just 1 MPa increase from the previous case. Figure 3 *e* shows the FOS values around the opening; which were 1.28–2.53 near the roof and face. It could also be observed that the area near to the face had started yielding due to the blasting effect, although not failing. Also, displacements around the opening had increased to almost 12 mm at the centre of the roof. The effect of the blasting could be seen in the form of yielding near the face.

Deployment of double boom jumbo to improve process cycle time of drilling: option-1

Following the above observations, a double boom jumbo, Sandvik DD422i-make was deployed, which could drill two holes at a time, thereby saving total drilling time by 30.70% and reducing the overall cycle time by 15.90% (Table 5) (details of individual process cycle time) and (analysis of results of both the cases of drilling length, i.e. 3.4 m and 4.0 m). The modified process cycle time with changes only in drilling 3.4 m and 4.0 m length is shown in and Tables 6, 7 and 8 respectively. In addition to face drilling the unit can also be used for cross-cut development and rock bolt hole drilling. An optional utility boom for bolting and utility works is available.

Table 8. Modified cycle time with the introduction of double boom Jumbo drill (Sandvik DD422i)

Type	Drive size	Drilling length/ hole (m)	Total drilling time for total holes (min)	Charging time (min)	Fume clearance (min)	Water spraying and loose dressing (min)	Mucking time (min)	Rock-bolt drilling	Grouting time	Bottom cleaning and face preparation	Advance per day	Total cycle time taken/ round (min)
Final blast pattern modified	4.5 m × 3 m	3.4	37.5	40	15	30	90	45	75	30	9.0 m @ 3.0 m advance per shift	365
	4.5 m × 3 m	4.0	45	50	15	45	105	45	75	30	10.80 m @ 3.6 m advance per shift	410

Table 9. Modified cycle time for final phase without rock-bolt drilling and grouting

Type	Drive size	Drilling length per hole (m)	Total drilling time for total holes (min)	Charging time (min)	Fume clearance (min)	Water spraying and loose dressing (min)	Mucking time (min)	Bottom cleaning and face preparation	Advance per day	Total time taken per cycle (min)
Final blast pattern modified	4.5 m × 3 m	3.4	37.5	40	15	30	90	30	15 m @ 3.0 m advance per round	250
	4.5 m × 3 m	4.0	45	50	15	45	105	30	14.40 m @ 3.60 m advance per round	290

Deployment of double boom jumbo to improve process cycle time of drilling and supporting activities: option-2

A self-propelled drilling rig named FACE MASTER 2.3 of Mine Master was introduced whose main design criteria are: optimization of technical parameters, life, operational reliability, maximum unification of assemblies and parts, ease of operation and maintenance. The machine is designed to drill blast holes and roof support holes in mine workings of nonferrous ore mines and similar environments with no hazard of explosion. The self-propelled double boom drill rig equipped with two fixed feeders provides a drilling area of 56 m^2. The working envelope allows the drilling of blast holes and roof bolt holes in working faces and necks: diameter of drilled holes is 45–76 mm, length of drilled holes is 4080 mm, maximum drilling range 8.55 m width, 6.7 m height, and rock drill-Montalbert HC 25. Diesel engine Deutz TCD2012L04-95, kW@ 2300 RPM, liquid-cooled, tramming four-wheel drive (power shift). The bolt drilling and face drilling could be simultaneously done in every third blast.

In order to develop the drive at a faster rate, the mine management decided to execute the development work in 6 h shift in a day for two months. Rest of the stopping work was performed in 8 h shift in a day with the deployment of two types of double boom jumbo, viz. Sandvik-make and Mine Master-make, for face drilling with double boom and face drilling with one boom and rock bolt drilling with another boom respectively. This has significantly improved the cycle time by 43.10% for 3.4 m drilling and 39.30% for 4.0 m drilling length. Table 9 shows results of advance per day with a pull of 3.0 and 3.6 m for 3.4 and 4.0 m drilling length respectively. It is evident that the best advance per day is 15.0 m for the improvized blast pattern with 3.4 m drilling length, which is comparable with the mechanized system of drive development.

Conclusion

Timely mine development is necessary to achieve the targeted production. Therefore, the present study was conducted in hard-rock metal mine and it was found that re-engineering opportunities in optimization of process cycle need to be considered by deployment of available technology such as numerical simulation for stability of excavation and advanced equipment for faster completion of operations. Numerical simulation showed that the proposed advance per blast in the method was optimum and stable during the mining operation until the third slice, even without rock bolting. However, it was observed that the roof strata near to the face was yielding due to the effect of the blast, although the extent of yielding was limited to a few centimetres from the face. Since the development contributes to 60% of the whole production compared to stopping in the room and pillar method of mining, the results of numerical modelling help the mine management to take judicious decisions to avoid rock bolting and supporting activities up to the third advance, thus enhancing the advance rate significantly. It was also found that deployment of double boom drill jumbo helped in reducing the total drilling time and overall cycle time by 30.70% and 15.90% respectively, in both the cases of drilling length, i.e. 3.4 and 4.0 m. Subsequently, in a further modified process, where the supporting activities were avoided till the third advance, there was significant improvement of the cycle time by 43.10% for 3.4 m drilling and 39.30% for 4.0 m drilling length respectively.

1. Stewart, P., Ramezanzadeh, A. and Knights, P., Benchmark drill and blast and mechanical excavation advance rates for underground hard rock mine development. In Australian Mining Technology Conference, 2006, pp. 45–58.
2. Barber J., Mennie, B., Poedjono, R. and Coad, G., Common infrastructure project – Development for the Future of PT Freeport Indonesia. In Proceedings of the Ninth Underground Operators' Conference, The Australasian Institute of Mining and Metallurgy, Melbourne, 2005, pp. 313–322.
3. Kalamaras, G., Carlo, A., Dinu, C., Cirvegna, G., Paolo, P. and Santucci, C., Up-to-date excavation, support, and lining solutions meet timing requirements for the first two tunnels of the 2006 winter Olympic games, Seattle, WA, United States, 2005, pp. 201–212.
4. Neumann, M., CAMIRO Safe and Rapid Development Project – Benchmarking of 12 Canadian Mine, Neumann Engineering and Mining Services, 2001.
5. http://www.downeredi.com, 2006.
6. Bruland, A., Hard-rock tunnel boring, advance rate and cutter wear. Norwegian University of Science and Technology, Trondheim, Norway, 1998.

ACKNOWLEDGEMENTS. This work is supported by a large mechanized underground metal mine of a public sector company in Kadapa district, Andhra Pradesh, which produces strategic mineral. We thank the Mine Management for support.

Received 1 April 2019; revised accepted 28 January 2020

doi: 10.18520/cs/v118/i10/1547-1556