Development of an Assessment Tool to Minimize Safe After Blast Re-Entry Time to Improve the Mining Cycle

A. D. S. Gillies, H. W. Wu & D. Shires
The University of Queensland, Brisbane, Australia

Published as: A.D.S. Gillies, H.W. Wu and D. Shires, Development of an Assessment Tool to Minimize Safe After Blast Re-Entry Time to Improve the Mining Cycle, *Proceedings, Tenth US Mine Ventilation Symposium, Anchorage,* Balkema, The Netherlands, 315-323, May 2004.

ABSTRACT:

A study has been examining the present problems resulting from inappropriate re-entry times across Australian underground development heading mining practices. It has focused on possible solutions that enable current ventilation practices with reduced quantities of fresh air without affecting re-entry times, or shorter re-entry times without increasing fresh air requirements or a combination of both. After blast re-entry times have been identified as a potential safety problems exaggerated by advanced mining technology and complicated by the modern prevalence of 12 hour shifts. Industry surveys on blasting re-entry time indicate most mines rely on a fixed time strategy to deal with the after blast re-entry mostly based on past experiences and observation. Very few monitor blast fumes for gas composition. A series of measurements of after blast fumes in various development heading arrangements has be undertaken to improve the understanding of occurrence and magnitude of blast contaminants. A development face ventilation study has been undertaken. Rigorous analysis has produced two empirical equations that can be used with some confidence to conservatively estimate re-entry times for level development headings. It was concluded that the working space volume used in mathematical modeling might be a constant for development headings of a similar configuration and environment, regardless of the relative distance from the end of the ventilation ducting to the face. Further testing is required to confirm this hypothesis.

1 INTRODUCTION

.

Ventilation systems on mines primarily have to ensure adequate environmental conditions for the workforce such as sufficient oxygen, cooling, and adequate dilution of contaminants from equipment and other sources. If rock and minerals are broken with explosives, the ventilation system must also ensure the replacement of the contaminated air in a reasonable time period before re-entry of the workforce can be permitted. An appropriate re-entry time can be determined by the quantity of contaminants released, the size of the ventilation circuit, and the quantity of fresh air used, which in turn is determined by requirements during the working shift.

If a reduction in the re-entry time is required, the only option is to introduce shorter circuits, which will then have to be ventilated in parallel. This increases total air requirements and costs, and is particularly detrimental for deep level mines where the heat generated by autocompression is already a burden on the mine cooling system. To overcome this, the use of series ventilation systems or recirculation systems has often been adopted as it can reduce fresh air requirements while maintaining stope face conditions. However, since rate of removal of blast contaminants is a function of the quantity of fresh air available, the re-entry time will increase. Therefore, the requirements for satisfactory stope face conditions, low operating costs and shorter re-entry times are interlinked in such a way that they cannot be minimized simultaneously.

The extended re-entry time for blast fumes has been offset in some mines by only blasting at the end of shift. This in itself has had a productivity benefit in mines working 8 hour shifts or less when contract miners are prepared to work through their meal period. Use of 12 hour work shifts complicates this situation. A study has been undertaken to analyze the present problems resulting from inadequate re-entry times across a variety of Australian underground mining practices. It has examined possible solutions that would enable current ventilation practices with reduced quantities of fresh air without affecting re-entry times, or shorter re-entry times without increasing fresh air requirements or a combination of both.

After blast re-entry times has been identified as one of the potential safety and health problems introduced by the advanced mining technology (Fisher, 1995). An examination of Western Australian mine safety and health records in the mid 1990s indicated that at least 20 underground miners in that state suffered incidences of "fuming" resulting in loss of at least one shift over a 12 month period. An industry survey on the blasting re-entry time of 19 mines has been undertaken (Gallo, 1995) to analyze the effects of current ventilation practices such as the use of series ventilation systems on after blast re-entry time. The mines surveyed were involved with the extraction of gold, silver and base metals with annual production rates between 180,000 and 3,000,000 metric tonnes. The results show that majority of mine surveyed (75 percent) have more than four blasts per day. This indicates that the production time loss due to an after blast re-entry delay can be significant. Most of the mines (78 percent) rely on a set time strategy to deal with after blast re-entry. This strategy is mostly based on past experiences and observations (75 percent). Of the mines surveyed 83 percent did not monitor the blast fumes. More than 40 percent of mines surveyed have indicated occurrence of incidents where safety has been compromised and more than half of the mines surveyed have agreed that further study into the after blast re-entry time would be beneficial to safety and health of miners.

Modern gas measurement instrumentation with data storage allows research to be conducted to establish safe and productive blasting post detonation re-entry times. This is an area where there is a paucity of information despite the importance to health, safety and production in metalliferous mines. Many incidents and accidents occur each year due to over exposure of personnel to blast fumes and there are significant losses in production due to poor procedures for determining appropriate re-entry times after blasting (DIRWA, 2003).

Selection of re-entry times impacts on mine productivity. Over estimation of the time period reduces the time mine labor is engaged productively; underestimation will very likely lead to safety and health effects.

A development face ventilation study has been undertaken. Results from the study can offer better work productivity and a reduction of after blast incidence while minimising mine ventilation costs. Possible solutions that would enable current ventilation practices with reduced quantities of fresh air without affecting re-entry times or shorter re-entry times without increasing fresh air requirements or a combination of both were also examined. This could lead to a significant saving in mine ventilation costs.

A study has been undertaken to examine approaches to determine suitable re-entry times to improve safety and production at a 2.2 million metric tonnes per year Australian base metal mine.

2 TESTING PROCEDURES

Site tests were undertaken to monitor and record the blast fumes generated from level development headings in the test mine. Detailed testing procedures were formulated. In summary, the procedures adopted consisted of:

- Having the instrument manufacturers calibrate portable gas instruments. Gas measurement was undertaken with Draeger Multiwarn and Miniwarn instruments. These instruments can measure gas readings for a number of gases and record data for a significant period.
- Preparation and testing of instrumentation prior to traveling to site.
- Calibration of portable instrumentation with each other and with the mine gas monitoring system
- Measurement of gas concentrations over time from development heading blasts using portable gas instrumentation.
- Measurement and recording of detailed information pertaining to each blast including airway dimensions, airflows and ventilation configurations, blast agent weights and configurations, temperatures and times.
- Downloading of data from portable instruments and the mine gas monitoring system prior to leaving site for later analysis.

3 EXPOSURE STANDARDS

Threshold limit value (TLV), time weighted average (TWA) and the short term exposure limit (STEL) are terms commonly used to describe the levels of exposure to various substances. Within this study focus is on the blast fume gases carbon monoxide (CO), nitric oxide (NO) and nitrogen dioxide (NO_2). NOHSC is a statutory body recognized in Australian that releases up to date and best practice information for the Australian industry in this area. Their current guidelines are to use a TWA of 30ppm for CO, 25ppm for NO and 3ppm for NO_2.

The STEL may have some relevance for personnel exposed to blast fumes. NOHSC (2003) state that the STEL is a 15 minute exposure and should not be exceeded. The STEL should not be exceeded at anytime during a working day even if the TWA is within the exposure standard, exposures at the STEL should not be repeated more than four times per day and there should be at least 60 minutes between successive exposures at the STEL

If for example a supervisor wishes to re-enter a development heading soon after blasting to examine the face, he may do so provided that the contaminant levels are below the STEL, the duration of the visit does not exceed 15 minutes and he does not repeat this process more than four times in one day. The same could be said for a mine worker who wishes to merely position a water jet or hose to washdown the face and muckpile soon after the blast. The danger, however, is that rarely do supervisors, let alone mine workers, have access to portable gas instruments to spot measure gas readings, or rarely do robust procedures exist to prevent personnel from exposure longer than 15 minutes. There may be conditions or situations in a mine where the STEL may be used under strict guidelines for re-entry after blasting, but this should only be limited to mine supervisors and not for the general mining workforce because of the obvious dangers.

The Australian mining industry has widely moved to the adoption of 12 hour shifts with most mines adopting rosters exceeding five days straight which has implication to applicable exposure standards. The TWA exposure standard is based on 8 hour shifts and a five day working week Great care must be taken when interpreting TWA values for shifts where workers are exposed to contaminants for longer periods. To prevent over exposure the NOHSC (2003) recommend the TWA should be calculated as follows Eq. 1.

$$TWA_{12hr} = \frac{8(24-h)TWA_{8hr}}{16h} \quad (1)$$

where h is the length of exposure, hours per day.

For a 12 hour shift the TWA_{12hr} is half that for an 8 hour shift giving exposure limits of 15ppm, 12.5ppm and 1.5ppm for CO, NO and NO_2 respectively.

In a modern Australian metalliferous mine it is unlikely that an average worker would be subject to the TWA_{8hr} (30ppm) for a 12 hour shift, let alone the de-rated and conservative TWA_{12hr} of 15ppm. This is based on the significant time in one shift an average worker may spend away from the working face in a clean environment with involvement in shift changeover, smoko/meal breaks, traveling in and out of the mine and other activities spent away from the potentially contaminated working face. This hypothesis may be simulated with the following fictitious 12 hour shift case study for a mine where blasting is carried out once per shift:

- Total of 1 hour travel time in and out of the mine, including shift changeover, with negligible exposure to CO.
- Total of 1.5 hours smoko and meal breaks, including travel to and from crib facilities, with negligible exposure to CO.
- A background CO concentration of 10ppm, to account for diesel emissions.
- Re-entry time of 30 minutes for mucking duties into a development heading based on the blast gas decay curves provided, as examples, in Figures 1 and 5. It is noted that the operator of the mucking unit would be initially subject to CO at a concentration of about 230ppm with this re-entry period.

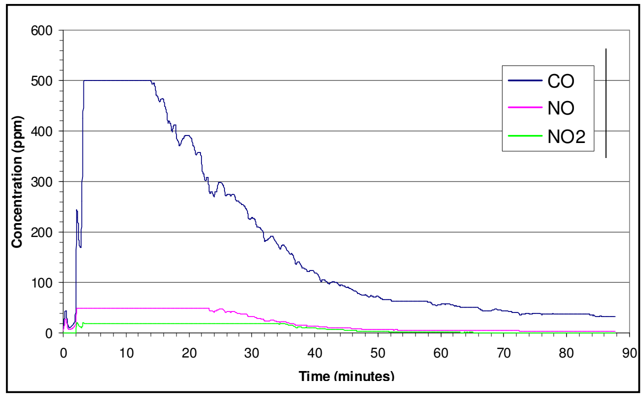

Figure 1. Blast Gas Decay for a Fictitious Case Study

The results of the simulation are provided in 15ppm and simple calculations show that the time weighted average contaminant exposure is 9600/720 = 13.3ppm. This is a figure that is less than the conservative TLV_{12hr} of 15ppm calculated previously.

Table 1 and simple calculations show that the time weighted average contaminant exposure is 9600/720 = 13.3ppm, which is less than the conservative TWA_{12hr} of 15ppm as calculated previously.

The simulation demonstrates that blast re-entry times could in fact be based on gas concentrations well in excess of either the TWA_{12hr} or TWA_{8hr}, whilst still allowing average workers to remain within the time weighted average for the 12 hour shift. The intent of this hypothesis is not for it to be used as the basis for determining re-entry times in this study, but merely to highlight the fact that the current philosophy of using the TWA_{12hr} or the TWA_{8hr} as the trigger for re-entry times may be overly conservative. This topic in itself could form the basis of an in-depth study in the future.

Table 1. Average blast exposures for a fictitious case study.

Activity	duration (mins)	duration (hours)	exposure (ppm)	time*exp (ppm.mins
Travel in-out/Shift Changeover	60	1	0	0
Smoko/Meals/Travel	90	1.5	0	0
Mucking out, 30-40mins	10	1.2	170	1700
Mucking out, 40-50mins	10		90	900
Mucking out, 50-60mins	10		60	600
Mucking out, 60-70mins	10		50	500
Mucking out, 70-80mins	10		40	400
Mucking out, 80-90mins	10		30	300
Mucking out, 90-100mins	10		20	200
General Duties	500	8.3	10	5000
	720	**12**		**9600**

Even though the TWA_{8hr} of 30ppm, 25ppm and

3ppm for CO, NO and NO_2 may be too conservative, they shall be adopted as the basis for determining re-entry times in this study.

4 DEVELOPMENT HEADING TESTS

During the one week testing program, 17 tests were conducted on development headings and 16 tests gave results that could be considered reliable. This paper principally examines results from seven tests (those in which there was reliable f ace ventilation delivered through ducting in the heading) and discusses subsequent analysis and interpretation.

4.1 *Typical Development Heading Design*

Heading types varied from drag cut strippings to full face cuts. Nominal dimensions of headings were 5.2m high x 5.0m wide. Typically, each face was drilled out with 60, 43mm diameter holes, with nominally seven lifter holes charged with Johnson's self stemming Easy Charge, nominally seven back holes charged with Johnson's Econotrim and the remainder of the face charged with ANFO. Four 100mm easers completed the cut. Tunnel incline/decline grades typically varied from 1 in 20 to 1 in 50.

4.2 *Typical Auxiliary Ventilation System*

Standard ventilation ducting used was 900mm in diameter and supplied in 20m lengths. Distances from the end of the ventilation ducting to the face ranged from 15 to 53m with an average and median of 24m and 23m respectively. The end of the ventilation ducting in all cases was neither "clipped" nor tied to assist in throwing the air further to the face for better sweeping. Ventilation ducting maintenance was generally good with minimal tears and satisfactory joins. Airflows ranged from 3.9 to 32m^3/s with an average and median of 15.4m^3/s.

4.3 *Instrument Positioning*

Placement of the gas monitoring instrument was based on affording protection more so than being the most ideal location for the testing of blast gases. The instrument needed to be protected from flyrock during the blast, water during subsequent muck pile wash down and the mucking units during load out. The instrument was typically hung from a rockbolt or low piece of steel roof mesh a considerable distance from the face and usually located in a depression in the tunnel wall. Distances from the instrument to the face ranged from 19m to 49m with an average of 37m and median of 34.5m. In any case, instrument positioning distance from the face was not seen as critical, and any increased time for blast gases to move across the instrument at times when the additional distance was large and air velocities low was seen as relatively insignificant in the overall scheme of things. In other words, instrument positioning may have affected the calculated re-entry times by up to a minute or so as a worst case.

4.4 *Test Results*

A TWA of 30ppm for carbon monoxide (CO) determines the minimum re-entry times for each and every one of the seven tests as shown in Figure 2. Nitrogen dioxide (NO_2) with a TWA of 3ppm is the next critical gas, whilst nitric oxide (NO) with a TWA of 25ppm is the gas of least concern.

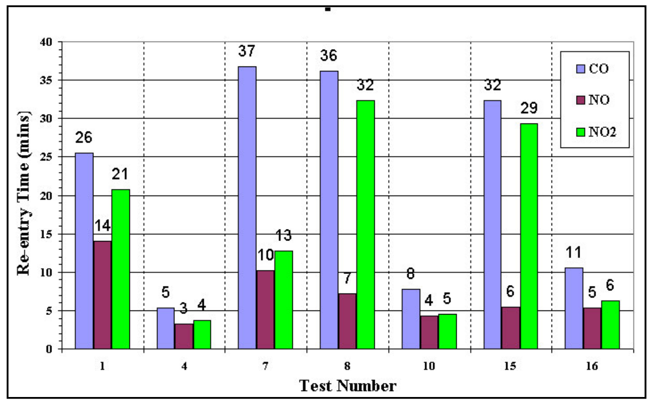

Figure 2. Re-entry times for seven development heading tests.

The results of seven tests are analyzed. These were selected as there was consistency in the operating procedure that the auxiliary fan ventilating the development heading had been switched on shortly after the blast. A mine layout for one of these tests (Test No. 4) is shown in Figure 3.

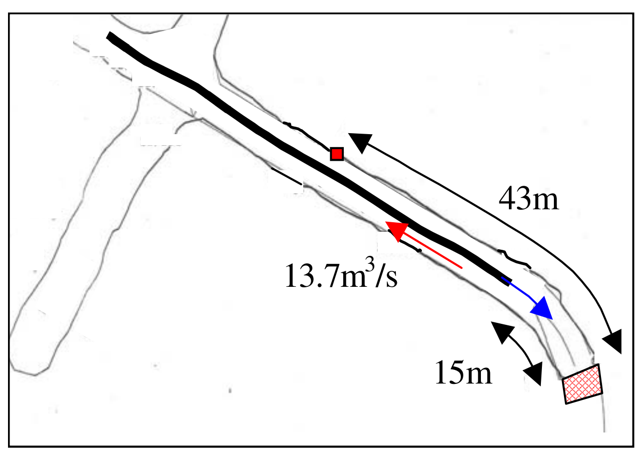

Figure 3. Example of test site layout.

The seven tests represent near-perfect gas concentration behavior following a blast in a heading with an auxiliary fan and ducting. The plots exhibit sharp rises almost immediately (~ time zero) or with mini-

minimal lag and this indicates minimal fan starting delay. Steep smooth decays represent near perfect mixing of the blast gases within the working space. An example of a plot for Test 4 is shown in Figure 4.

The gas plots reached a ceiling for a short period in many of the tests indicating that concentration was exceeding instrument limits.

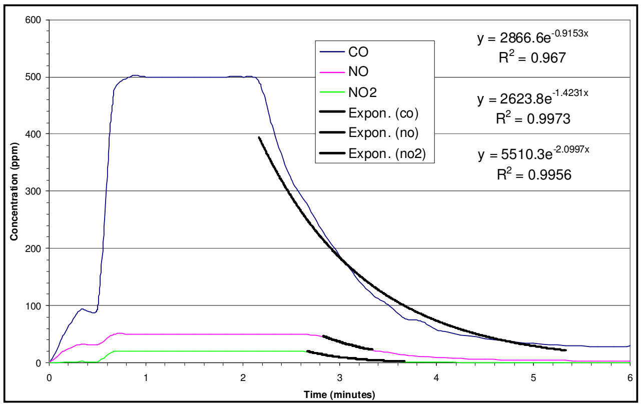

Figure 4. CO, NO and NO_2 readings versus time after blast.

4.4.1 *Calculations & Results*

From the test blast gas curves, a number of important parameters can be determined (Table 2) such as:

- volume of gas liberated, V_{CO}
- peak gas concentration, x_0
- volume of actual working space, *Y'*
- theoretical re-entry time, *Tt*
- actual re-entry time, *Ta*

Table 2. Test blast parameters.

Test	Area	T	x_{AV}	Q	V_{AIR}	V_{CO}	x_0	Y'	Tt	Ta
1	6500	26	250	15.4	24024	6.0	2150	2793	12.9	25.5
4	2360	6	393	13.7	4932	1.9	2150	902	4.7	5.3
7	5600	40	140	7.4	17760	2.5	1120	2220	18.1	36.8
8	6700	40	168	15.9	38160	6.4	3000	2131	10.3	36.2
10	3050	8	381	10.5	5040	1.9	2300	835	5.8	7.8
15	5800	34	171	17.2	35088	6.0	2120	2823	11.6	32.3
16	4000	11	364	20.7	13662	5.0	1900	2615	8.7	10.5

The peak carbon monoxide CO concentration x_0 in the working space volume can be determined by the intersection of the extrapolation of the rise and decay portions of the curves as shown for the example test in Figure 5 for Test No. 4.

The area under the extrapolated gas concentration curves can be calculated and the average CO concentration x_{AV} over the period of each test T can be determined by Eq. 2. It is noted that the area under the tail of the curve is considered to be relatively small compared to the area under the body of the curve and is therefore insignificant.

$$x_{AV} * 10^{-6} = \frac{V_{CO}}{V_{AIR}} \qquad (m^3) \qquad (4)$$

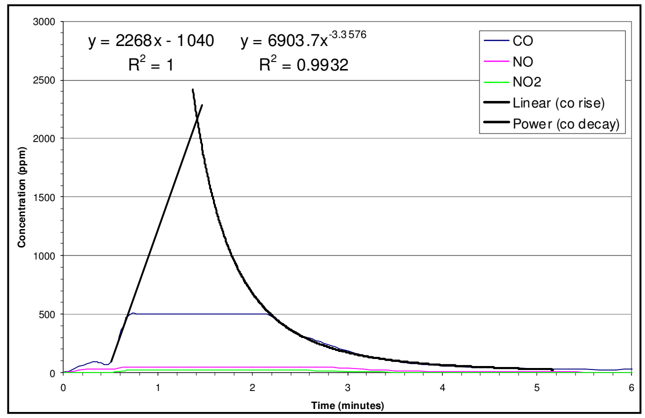

Figure 5. Extrapolated gas concentration curves.

$$x_{AV} = \frac{AreaUnderCurve}{T} \qquad (ppm) \qquad (2)$$

The volume of fresh air V_{AIR} supplied to the working space volume over the period of the test T can also be calculated Eq. 3 by knowing the fresh air flow Q.

$$V_{AIR} = Q * T * 60 \qquad (m^3) \qquad (3)$$

Knowing V_{AIR} and x_{AV}, the volume of CO (V_{CO}) produced from the blast is then able to be determined by re-arranging Eq. 4.

The volume of the working space *Y'* can then be calculated using V_{CO} and x_0 as follows

$$Y' = \frac{V_{CO}}{x_0} \qquad (m^3) \qquad (4)$$

Application of the theory of gas dilution (DeSouza et al, 1993) presents the equation

$$e^{\frac{-(Q_g+Q)(t-t_0)}{Y}} = \frac{(Q_g + B_g Q) - x(Q_g + Q)}{(Q_g + B_g Q) - x_0(Q_g + Q)} \qquad (6)$$

where:

Q_g	Gas contaminant inflow quantity	m^3/s
Q	Fresh air quantity	m^3/s
$t-t_0$	Time (τ)	minutes
Y	Volume of working space	m^3
B_g	Concentration of gas contaminant in intake air	fraction
x	Concentration of gas contaminant in the mixture	fraction
x_0	Concentration of gas contaminant in the initial air space	fraction

By taking natural logs of both sides of Eq.6 and rearranging, the time τ for dilution in the unsteady state is finite and can be determined as follows (Eq. 7):

$$\tau = \frac{Y}{Q+Q_g} \ln\left[\frac{(Q_g + B_g Q) - (Q + Q_g)x_0}{(Q_g + B_g Q) - (Q + Q_g)x}\right]$$

(7)

If it is assumed there is no gas contaminant in the intake air ($B_g = 0$) and the gas contaminant inflow is small Q_g in comparison to the fresh air quantity (Q), Eq.7 reduces to Eq.8:

$$\tau = \frac{Y}{Q} \ln\left[\frac{Q_g - Qx_0}{Q_g - Qx}\right] \qquad \text{(seconds)} \qquad (8)$$

When the limiting gas concentration x is prescribed by mining law, then x = TWA. Assuming the gas contaminant inflow Qg to be zero, can be reduced to Eq. 9 to give an estimation of the re-entry times, where the final concentration x can be taken as the 30ppm TWA for CO.

$$\tau = \frac{Y'}{Q} \ln\left[\frac{x_0}{x}\right] \qquad \text{(seconds)} \qquad (9)$$

4.4.2 *Analysis of Results*

The results presented in Table 2 are interesting. The volume of CO gas varies from 1.9 to 6.4m^3 with an average of 4.2m^3, but there appears to be two distinct groups; tests 4, 7 and 10 with a volume ranging from 1.9 to 2.5m^3 and tests 1, 8, 15 and 16 ranging from 5.0 to 6.4m^3. Obviously, some variation in the amount of fumes generated is expected with varying blast parameters as discussed by Hine et al (1985). However, most development headings at the test mine offer a fairly consistent environment with similar design and configuration; one would expect the quantity of fumes generated to be similar. Test 10, however, was a wet face and up to half the face was charged with packaged explosives and the remainder with ANFO. This reduction in powder factor may explain the differences in the quantity of fumes generated. It is possible that varying fragmentation from tests 4 and 7 resulted in more of the blast gases becoming entrapped in the muckpile. Hine et al (1985) suggested that up to 60 percent of the blast fumes may become entrapped in the muckpile and in the fissures of the host rock. The reduced volume of CO produced therefore has the affect of reducing the working space volume *Y'*.

DeSouza et al (1993) acknowledged that the gas dilution formulae represent theoretical relationships and may not represent the actual (mine environment) conditions such as gas density variations, imperfect mixing or the continuous production of contaminants from the walls or the muckpile.

The peak CO concentration in the test blast ranges from 1120 to 3000ppm with an average of about 2100ppm. The unusual peak concentrations of 1120 and 3000ppm for tests 7 and 8 could simply be attributable to errors in extrapolation of the curves.

It is noted that *Y'* is used in lieu of *Y*, because the volume of the working space (*Y'*) does not appears to correlate with the volume of tunnel beyond the end of the ventilation ducting (*Y*), contrary to Winn's (2002) assumption, as shown in Table 3.

Table 3. L_D and L_G and comparison between *Y* and *Y'*.

Test No.	L_D (m)	L_G (m)	Y (m^3)	Y' (m^3)	Difference between *Y* and *Y'* (%)
1	53	63	2914	2793	-4%
4	15	43	390	902	57%
7	23	34	658	2220	70%
8	32	49	968	2131	55%
10	21	33	579	835	31%
15	21	49	635	2823	78%
16	16	35	416	2615	84%

* Where L_D is the distance from face to the end of ventilation ducting and L_G is the distance from face to the gas instrument points.

4.4.3 *Re-entry Times*

The theoretical re-entry times (*Tt*) for each test can be compared to the actual re-entry times based on a TWA of 30ppm, as shown in Table 4. It is demonstrated that the theoretical calculations for the most part grossly underestimate the actual re-entry times *Ta*.

Table 4. Re-entry times.

Test No.	*Q/Y'* (l/s)	*Tt* (mins)	*Ta* (mins)	Difference (%)
1	0.0055	12.9	25.5	51%
4	0.0152	4.7	5.3	88%
7	0.0033	18.1	36.8	48%
8	0.0075	10.3	36.2	28%
10	0.0126	5.8	7.8	74%
15	0.0061	11.6	32.3	36%
16	0.0079	8.7	10.5	83%

There is good correlation with this data as shown in Figure 6, with an exponential relationship representing the theoretical curve as expected, and a logarithmic relationship representing the actual re-entry times when *Q/Y'* is plotted against re-entry time *T*.

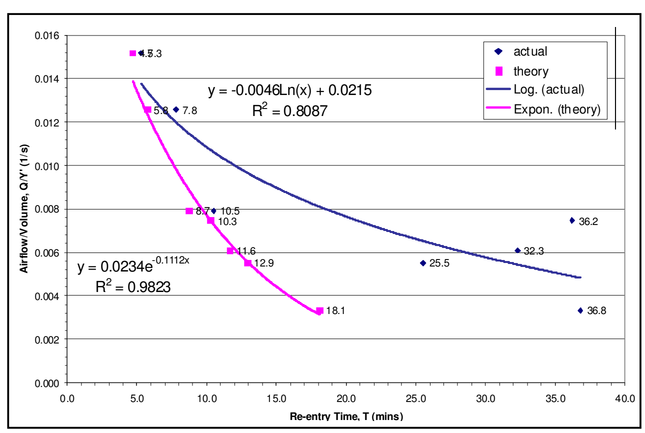

Figure 6. Re-entry Time Correlation (i)

It is shown that the theoretical and actual relationships approximate each other for high values of *Q/Y'*, and that the difference or error between the relationships increases with a decrease in *Q/Y'*. The relationship between *Q/Y'* and the actual re-entry time *T* is shown in Eq. 10.

$$Q/Y' = -0.0046 \ln(Ta) + 0.0215 \quad (\text{secs}^{-1}) \qquad (10)$$

If a conservative approach is taken and the maximum *Y'* of 2800m^3 is adopted, which should represent the maximum for a typical level development heading, then the graph may be further simplified to give a relationship between *Q* and *T* (Figure 7).

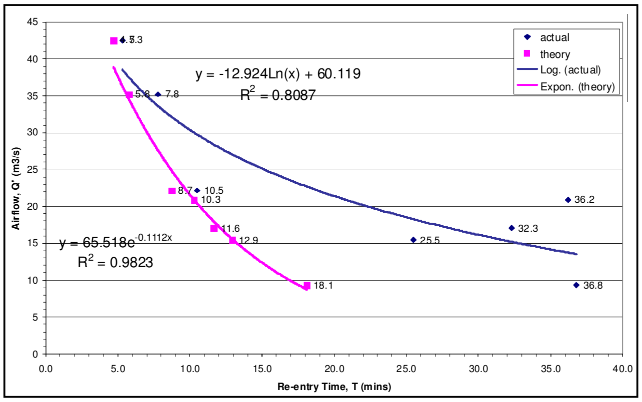

Figure 7. Re-Entry Times Correlation (ii)

The simplified relationship between *Q* and *Ta* becomes (Eq. 11).

$$Q = -12.924 \ln(Ta) + 60.119 \quad (\text{secs}^{-1}) \qquad (11)$$

Rearrangement of Eq. 11 and taking the exponential function of each side gives an equation where the actual re-entry time (*Ta*) may be approximated by the following exponential equation (Eq. 12).

$$Ta = e^{-\left(\frac{Q-60.119}{12.924}\right)} \quad (\text{mins}) \qquad (12)$$

The relationship between the theoretical and actual re-entry times can be observed as shown in Figure 8. Good correlation exists, and a power function can be used to describe the relationship.

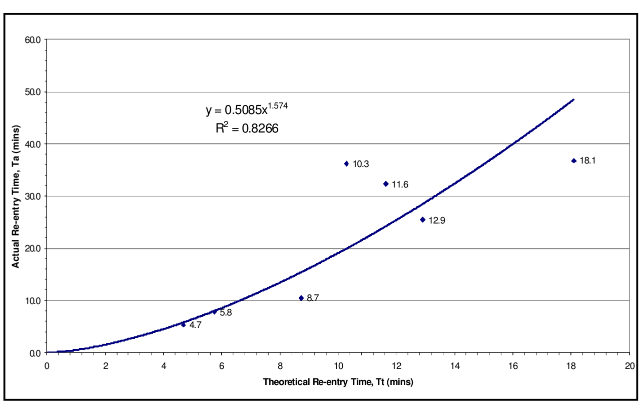

Figure 8. Relationship between *Tt* and *Ta*.

Actual re-entry times *Ta* can then be written as function of the theoretical re-entry times *Tt* as demonstrated in Eq. 12.

$$Ta = 0.5085\ Tt^{1.574} \quad (\text{mins}) \qquad (12)$$

Where the *Tt* can be calculated in minutes using the previously simplified theory of dilution equation. Eq.9 can therefore be re-written as:

$$Ta = 0.585\left[\left(\frac{Y'}{Q}\right)\ln\left(\frac{x_0}{x}\right) \div 60\right]^{1.574} \quad (\text{mins}) \qquad (13)$$

Once again, if a conservative approach is taken and the maximum working space volume (Y' = 2800m^3) and the maximum peak CO concentration (x_0 = 3000ppm) are adopted, which represents the maximum for a typical development heading, and the TWA for CO is taken as 30ppm then Eq. 13 may be simplified Eq.14 to give an approximation of *Ta*.

$$Ta = 2383.7\ Q^{-1.574} \quad (\text{mins}) \qquad (14)$$

Two empirical equations for estimating *Ta* have been derived, both expressed as a function of *Q*. These can be used to conservatively approximate re-entry times for typical level development headings; Eq.11 and Eq.14. Figure 9 shows that reasonable correlation exists between these equations.

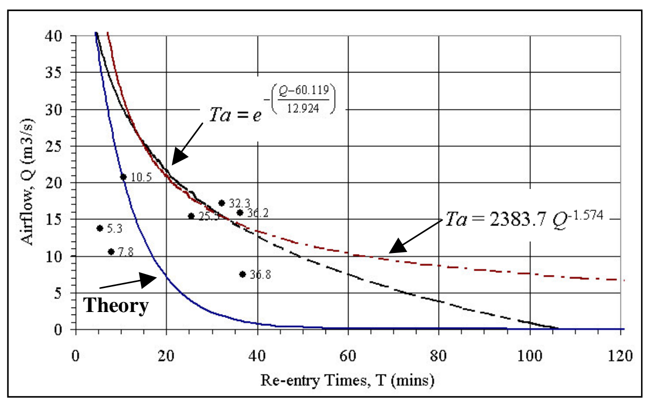

Figure 9. Correlation of re-entry equations.

Additional testing data is required to firm up both equations beyond about 40 minutes.

The plot of actual re-entry times on Figure 9 demonstrates that both equations offer a reasonably conservative approach to estimation re-entry times. Of the seven tests, there are only two test results that sit marginally above the curves, whilst all others fit comfortably under.

The above analysis demonstrates that either equation would be reasonably capable of safely estimating re-entry times for typical level development headings for airflows exceeding about $8m^3/s$. Whilst these equations give a conservative approach, they should provide reasonable confidence to estimate safe re-entry times whilst maximizing productivity by ensuring headings can be re-entered as soon as gas concentrations reach safe levels.

For these equations to be effective however, good ventilation practices must be upheld in ensuring the ventilation ducting is maintained in reasonable condition and the end of the ventilation ducting kept to within 15m from the face to maximize the sweeping effect across the face.

5 REGULATOR TESTS

The test mine currently has in place a system of remotely controlled regulators. During blasting all level regulators are opened to prevent inadvertent air blast damage to the regulator doors. All regulators remain open until such time as the blast fumes have cleared from all the levels. The doors are then returned to their pre-set levels to allow work to resume on the working levels. This in an inefficient practice and leads to longer re-entry times on the lower levels. This is because the primary ventilation airflow follows the path of least impedance leading to an oversupply of air on the upper levels. This leaves lower levels with insufficient air resulting in delays for diluting and clearing of blast fumes.

A more efficient practice may be to open the doors fully on those levels where blasting takes place whilst closing the doors (or leaving them at their normal setting) on those levels where there is no blasting. This way, blast fumes should be cleared quickly and re-entry times significantly reduced on the blasting levels with this approach.

A complication would arise when blast fumes from single blasts filter through several levels via ore passes, stopes, or rises. Through use of the “contaminants” feature of a ventilation network simulation program it should be possible to roughly predict the flow of blast gases through the mine. This would allow the Ventilation Engineer to set the regulator doors appropriately on each level to give the optimum clearing of blast fumes. Once the gas levels from the test mine’s gas monitoring system indicate that the appropriate TWA has been reached the regulator doors could be returned to their pre-set levels.

6 FURTHER RESEARCH

Further research needs to be conducted at suitable mines to confirm the empirical hypotheses developed. It is suggested that detailed testing focus on one or two critical path development headings within a mine such as a priority high-speed access to a new mine area. There are a number of major advantages in doing repeated tests on a small population of critical path headings:

- The testing environment remains relatively constant
- The heading design and configuration remains the relatively constant
- The auxiliary fans are usually switched on without delay after the blast for critical path headings.

It is suggested that all possible parameters be recorded for every blast to try and better understand abnormalities or inconsistencies in test results if they occur. Parameters to record should include:

- Wet and dry bulb temperatures and atmospheric pressure
- General design and configuration of the heading (tunnel dimensions, grade, type of cut etc)
- Number and depth of holes in each face, and the exact quantities of each type of explosive used including any stemming
- Ventilation flows (airway dimensions and air velocities) and a detailed description of the ventilation ducting condition, relative distances, orientation and end condition
- General description of the testing environment such as the presence or absence of water, geology, fragmentation etc (these can affect blast gas quantities). Photographs may also be appropriate.
- Times of critical events (blast initiated, fan-on, instrument-on, etc)

In summary, the testing regime should focus on a discrete part of the overall project where the time schedule is paramount, the operating environment and design is consistent and where the quality and accuracy of the tests results can be assured.

7 CONCLUSIONS

The aim of this study is to examine approaches to determine suitable re-entry time to improve safety and production at mines. The literature review has highlighted a number of important aspects of previous research, namely mathematical modeling of blast fume decay (DeSouza et al, 1993), exposure standards (McPherson, 1993) and test work on blasting fumes (Hine et al, 1985) and determination of re-entry times by Winn (2002). A simulated case study has demonstrated that blast re-entry times could probably be based on concentrations well in excess of the conservative TWA_{8hr} and TWA_{12hr}. The TWA_{8hr} has however been adopted for determining re-entry times in this study.

Most Australian mines currently base their re-entry times on the experience of their workforce. In general they have not performed testing or attempted to apply any other means to determine safe and productive re-entry times. This approach presents potential for the overexposure of personnel and losses in productivity due to inappropriate methods for determining re-entry times.

Portable gas monitoring equipment has been used to capture the behavior of blast gases in auxiliary ventilated level development headings. An analysis of the development end test results have shown that carbon monoxide is the critical gas in terms of defining minimum re-entry times. The development end test results for auxiliary ventilated headings show that fume dilution appears to be best described by a power relationship, as opposed to the classic exponential relationship for flow-through ventilation systems such as through the regulator doors (or though a stope), and for naturally ventilated workings. This new phenomenon has not been concluded by previous research and only with further testing can this relationship be confirmed.

Rigorous analysis has produced two empirical equations that can be used with some confidence to conservatively estimate re-entry times for level development headings:

$$Ta = e^{-\left(\frac{Q-60.119}{12.924}\right)} \qquad \text{(mins) or}$$

$$Ta = 2383.7\ Q^{-1.574} \qquad \text{(mins)}$$

Both equations are based on a function of Q and give similar results for values of Q greater than about 8 m^3/s. It was concluded that the working space volume Y' used in mathematical modeling might be a constant for development headings of a similar configuration and environment, regardless of the relative distance from the end of the ventilation ducting to the face. Further testing is required to confirm this hypothesis.

The study results have been achieved through undertaking of the following stages.

- An extensive literature review with particular emphasis on mathematical modeling of blast fume decay, exposure standards and recent test work on blasting fumes and re-entry times
- An analysis of appropriate exposure standards and the selection of suitable threshold limit values for the determination of re-entry times
- Design of underground test procedures to measure post detonation fume products in level development headings at a test mine.
- Measurement of post detonation fume products, airflows and other blast parameters such as design, configuration and the operating environment in level development headings.
- An analysis of test results and mathematical models of blast fume decay to allow an empirical relationship to be established that can be used to determine suitable re-entry times.
- Recommendations on future research on this topic

REFERENCES

Department of Industry and Resources Western Australia (DIRWA), 2003. Safety Bulletin – Re-entry After Blasting, March. At http://notesweb.dme.wa.gov.au/.

De Souza, E.M., and Katsabanis, P.D., 1993. On the Prediction of Blasting Toxic Fumes and Dilution Ventilation, *Mining Science and Technology*, 13: 223-235.

Gallo, N., 1995. *Leakage Perfromance of ACME ACFAB^{TM} Auxiliary Ventilation Ducting*, Undergraduate Thesis (unpublished), University of Queensland, Brisbane.

Haase, H.H., 1991. The Potential of Blast Contaminant Control for Reducing Re-Entry Times and Ventilation Costs, *Journal of the Mine Ventilation Society of South Africa*, 44: 86–89.

Hine, M., and Jones, I.O., 1985. The Analysis of Post Detonation Fumes in an Underground Operating Environment – A Case Study, *Proceedings 2nd U.S. Mine Vent Symp*., Nevada-Reno 23-25 September 1985 (Ed: P Mousset-Jones) pp 689–698. Rotterdam/Boston: Balkema.

McPherson, M.J, 1993. Chapter 11 – Gases in Subsurface Openings, *Subsurface Ventilation and Environmental Engineering*, London: Chapman & Hall.

National Occupational Health and Safety Commission (NOHSC), 2003. http://www.nohsc.gov.au. June.

Winn, D, 2002. *Blast Fumes and Re-entry Times in Underground Metalliferous Mines*, Undergraduate Thesis (unpublished), University of Queensland, Brisbane.