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Coal Mine Explosion Suppression Using Active On-Board Suppression Systems - The South African Experience

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ABSTRACT; Coal mining operations in South Africa commenced in 1874. With the introduction of continuous miners (CM) in the early 1970s to the South African coal mining industry, the number of ignitions and explosions related to frictional ignitions has increased. After the explosion at Middelbult Colliery, South Africa in May 1993, which claimed 53 lives, the South African coal mining industry and the Safety In Mines Research Advisory Committee (SIMRAC) united forces to establish a surface facility to develop and test on-board flame suppression systems for CM and roadheaders (RD) to enhance the safety of South African coal mine workers in collieries. The first test that was conducted in this newly constructed test tunnel at the Kloppersbos Research Facility, CSIR was in July 1995. From July 1995 to December 1997, 42 tests have been conducted using the facility and have focussed on on-board active ignition suppression systems for CM. Since flame propagation speed is an extremely important parameter, the CSIR-Miningtek, the operators of the test facility, made the results of the test programme available for re-analysis and it is this analysis of flame speeds, with and without the application of the suppression system that is reported in this paper.

1 INTRODUCTION

The G. P. Badenhorst Research Facility, which is owned and operated by the CSIR-Miningtek, is situated 40 km north of Pretoria, where in 1987 a 200-meter long circular explosion gallery was completed (Cook, 1995). This gallery has so far been used to test different types of barriers for stopping flame propagation in coal dust explosions. In response to the need for enhanced precautionary measures to safeguard mine workers in collieries from the consequences of methane ignitions in a heading, the coal mine industry expressed the desire for the development and testing of an active on-board suppression system (du Plessis and Bryden, 1997). To serve this purpose, a new 20 m long rectangular shape test tunnel was constructed in 1995.

This facility has been used to develop and test on-board, active suppression systems with a particular view to determining the exposure of CM operators close to the coal face to methane flames. In other words, the flame must be extinguished before it reaches the machine operator's position. This work was conducted by CSIR-Miningtek and funded by contracts with SIMRAC and the system manufacturers, CENTROCEN. The way to determine the effectiveness of the flame suppression system is to note the reduction or increase in the flame speed. The lower the flame speed, the more effective the flame suppression system. Results from this test work have been made available for further analysis (Gene, 2000) and it is this analysis of flame speeds, with and without the application of the suppression system that is reported in this paper.

The type of suppression system used in the tests is of a proprietary nature and, as such, no details can be made available.

2 DESCRIPTION OF THE TEST TUNNEL

The short test tunnel simulates conditions that could be encountered at the face of a bord and pillar heading in an underground coal mine.

The test tunnel is 20 m long, 7 m wide, with a variable height which can be set at heights of between 2 m and 6 m in increments of 0,5 m. It has a rectangular shape closed at one end. The test tunnel is equipped with sensors (pressure, flame and temperature) to measure the pressure generated by the explosion, to delect the rate of the flame travel and to determine temperature increases especially in the vicinity of the CM operator's position; a data acquisition system to computerise the test output; a methane-mixing and measuring system as well as an ignition source to ignite the methane/air mixture; and a video camera for the visual recording of the event (Figure 1. Figure 2 and Figure 3).

The tests are carried out on a scale of 1:1 i.e. at full size. Some of the tests were conducted on full-face tunnels while others were conducted with a shoulder in position as shown in Figure 1, to more accurately simulate the underground condition. Earlier tests were done using an actual CM machine, which was on loan from a mine. However, due to production requirements at the mine, this machine was taken back and was replaced by a model of equivalent geometry constructed of steel.

According to the test protocol, provision has been made to simulate the conditions in a heading being mined by a CM after the first lift or part of the first lift has been completed through the addition of a shoulder towards the front of the tunnel as shown in Figure I. Because the CM is about 3,2 m wide, it cuts the heading in two lifts. This creates the shoulder and this shoulder will be able to simulate a cut of up to a depth of 6 m for all the seam heights. The heading can be simulated at the start or end of the lift and can be done without the shoulder to give a full heading width of 7 m. This is similar to a test being conducted in a full heading as would be the case in the testing of roadheaders.

According to the dimensions of the continuous mining machines, square frames near the closed end of the test tunnel allow the attachment of a plastic membrane thus forming a chamber into which the air-methane mixture is pumped (Figure 2). The position of the membrane varies depending on whether a shoulder is in position or not. If the shoulder is absent, the membrane is located 5 m from the closed end of the tunnel. If the shoulder is present, the distance varies from 5 m to 7 m according the test to be conducted.

Tine specifications of the sensors, data acquisition system and methane mixing in the test tunnel are given in the protocol for testing procedures in the SIMRAC Project Report (du Plessis and Bryden, 1997). There are 76 flame sensors, one dynamic pressure sensor, one static pressure sensor and one temperature sensor inside the tunnel (Figure 3).

By measuring the time of activation of the individual sensors, the speed of the flame advance can be obtained as well as the profile of the final positions reached by the (lame front, ft should be noted that the system has a distance sensitivity of one meter. A glass cover is placed over each sensor on the tunnel wall to provide protection. These glass protection covers are cleaned and inspected before every test to ensure that the correct flame intensity will be recorded. When each of the four sensors at 1-meter intervals is activated, a digital output is generated. This will indicate if the flame has passed that point or which side of the tunnel the flame has passed. The positions of the 76 flame sensors inside the test tunnel are shown in Figure 4.

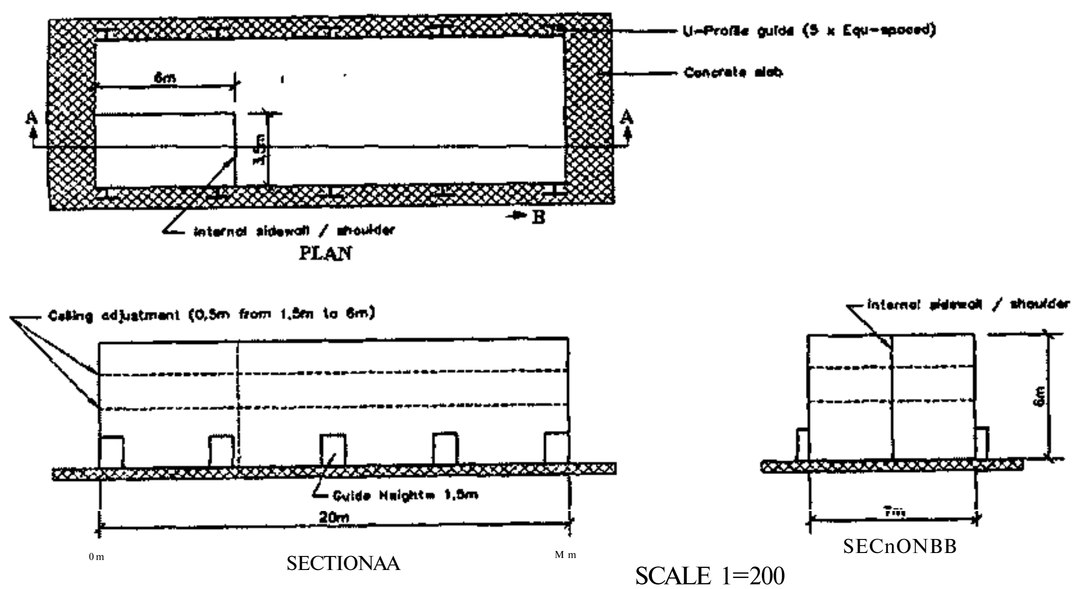


Figure I Test tunnel (du Plessis and Bryden. 1997)

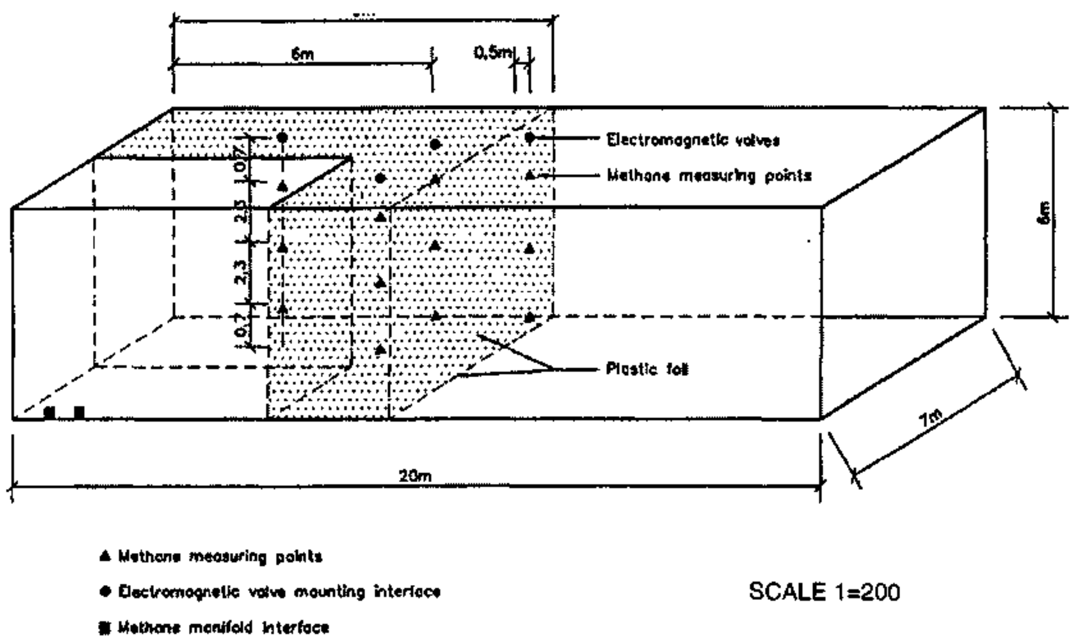
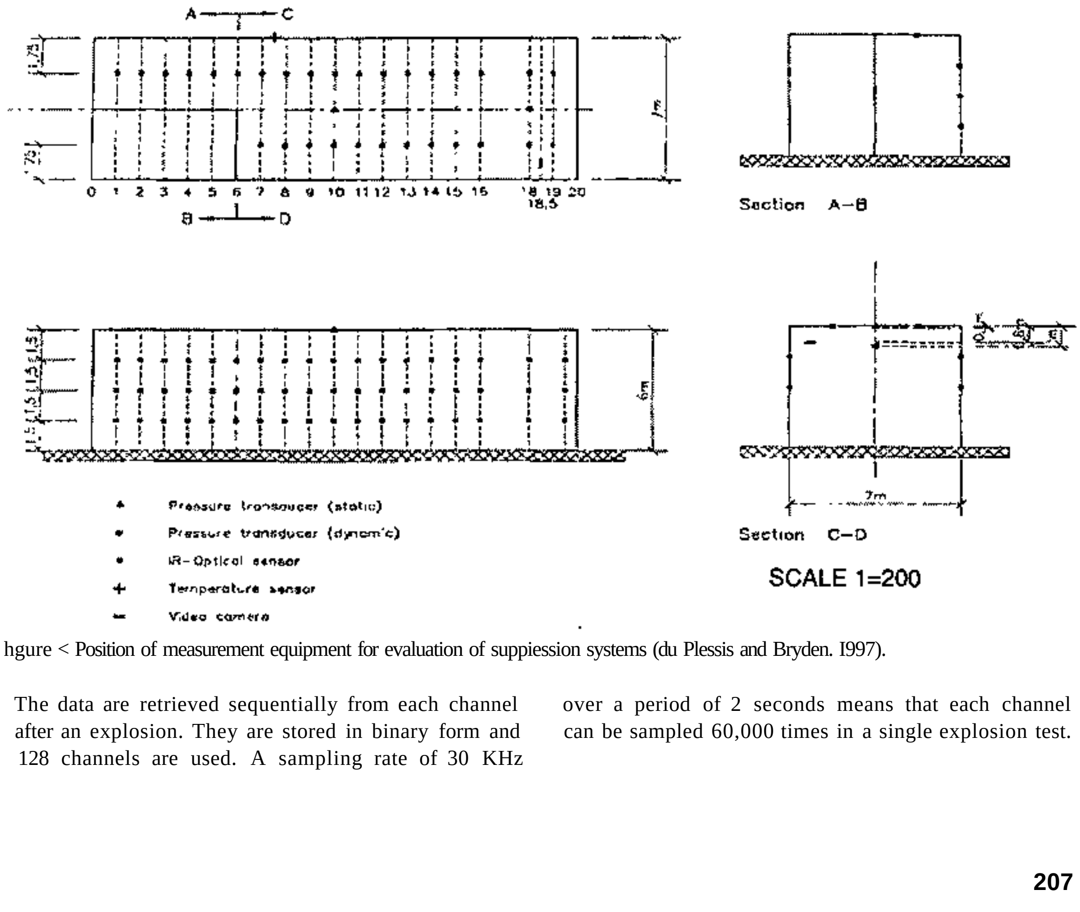


Figure 2. CH_4/air mixture measuring points and electromagnetic valve mounting (du Plessis and Bryden. 1997).



hgure < Position of measurement equipment for evaluation of suppiession systems (du Plessis and Bryden. I997).

The data are retrieved sequentially from each channel after an explosion. They are stored in binary form and 128 channels are used. A sampling rate of 30 KHz over a period of 2 seconds means that each channel can be sampled 60,000 times in a single explosion test.

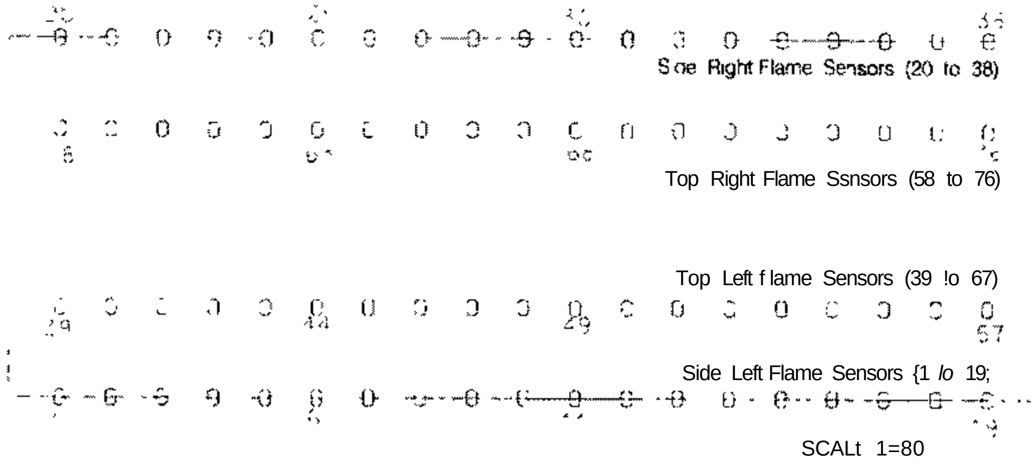


Figure 4. Position of the flame sensors

3 EXPERIMENTAL PROCEDURE

The tests conducted can be categorised into two different groups:

- tests conducted without a suppression system
- tests conducted with a suppression system

The tests conducted without a suppression system were aimed not only at determining the extent of the flame, the flame speed and the value of the dynamic pressure and the temperature increase inside the tunnel, but also to calibrate the test tunnel equipment and prepare the tunnel for active suppression system tests.

According to the testing protocol, when suppression tests were conducted, the ignition source was positioned between the drum and the face so that it was in the sighting shadow of the sensors of the suppression system. The methane/air mixture was ignited by means of a fuse cap (200 joule) or a chemical detonator. The ignition source and the data acquisition system were activated simultaneously, thereby allowing the controlled capture of the explosion data. An in-tunnel video camera captured the visual material. The visual material and the data acquisition system output were used to determine the extent of the tlame, the values recorded by the dynamic and static pressure sensors as well as the temperature sensor.

Different methane/air volumes and concentrations for a CM, CM model and finally CM mounted on-board suppression systems were used. During the tests conducted, the roof height of the test tunnel was set at 2,5 m and during the shoulder tests, the depth of shoulder was 2 m and its width 3,5 m. Plastic membranes were used to create an explosive methane/air mixture in a chamber covering the head of the machine. Methane/air concentrations of 7,5 to 12 % were used. The volume of the mixture depends on the height of the seam being simulated, the position of the membrane and the required methane/air concentration.

4 EXPERIMENTAL RESULTS

There were 76 (4x19) flame sensors inside the test tunnel. Two sets of flame sensors in linear array (19 each) were located on the sides of the tunnel, while the other two sets were on the roof (total of 4 sets of flame sensors, Figure 4). Figure 5 shows how the flame arrival time at a specific flame sensor was determined. The channel numbers from 1 to 76 correspond with the number of the tlame sensors, e.g. channel 24 corresponds with tlame sensor number 24. Similar readings were obtained for the 76 flame sensors to obtain the exact flame arrival time. Figure 6 combines all flame sensor-reading results for one test and demonstrate the flame propagation in seconds. The maximum time for which data can be recorded is 2 seconds.

From a research point of view, one of the most important parameters to study is flame speed. However, depending on the method of calculation, different results may be obtained. From an initial study of the data, it was apparent that the early stages of ignition, where the interaction between the initiator and the methane/air mixture takes place, contribute lo the degree of experimental error. While it was important to calculate the flame speeds and arrival time from a fixed datum ($t_{(}i$=0), the results were also calculated from the time the tlame passes through the membrane position (t_6) The equations used for these four calculations are

1 Flame $speed_1 = V_1 = [(1/(t_1 - t_0) + 1/(t_2\ t_1) + 1/(t_{19}\ t_{18})) / 19]$ (m/s)

2 Flame $speed_2 = V_2 = [19 / (t_{19} - t_0)]$ (m/s)

3 Flame $speed_3 = V_3 = [(1/(t_6 - t_0) + 1/(t_7 - t_6) + 1/(t_{19} - t_{18})) / 14]$ (m/s)

4 Flame $speed_4 = V_4 = [13 / (t_{19} - t_6)]$ (m/s)

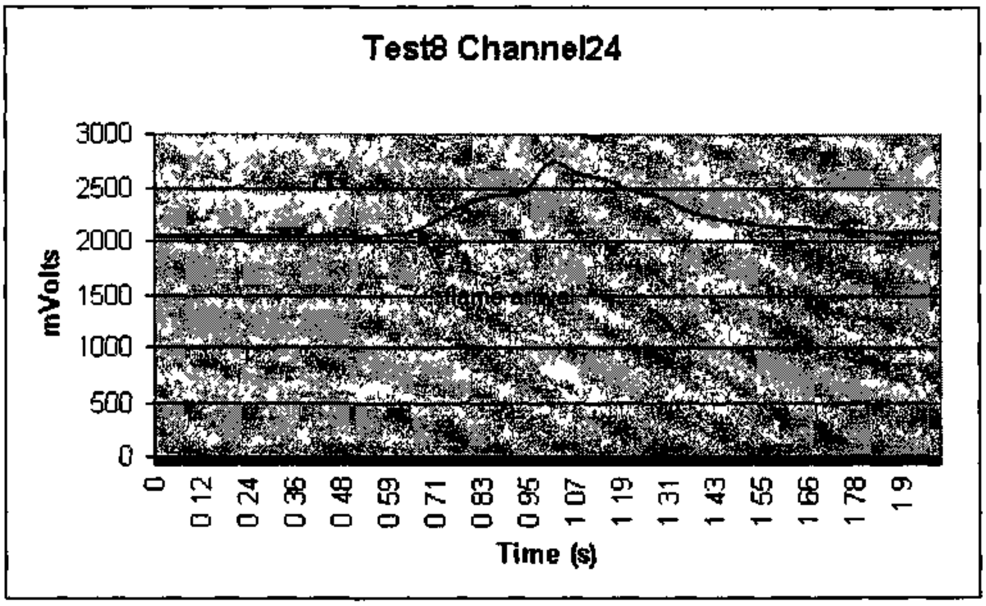


Figure S Flame anı val time

In the first method of calculating the flame speed (V|) the distance between two consecutive flame sensors which is 1 m, is divided by the difference between the two consecutive flame sensor readings The sum ot these values is then divided by 19 where 19 was the distance between the first sensor (ti) and the last sensor (tio) This formula was applied to all 4 sets ot flame sensors (as described above) The average results obtained from the tour linear arrays ot flame sensors were then added together and divided by 4 to calculate the total average flame speed through the test tunnel In the same way, the aveiage results obtained for each set of flame sensors when calculating with the second, third and forth tormulae weie also added together and divided by toui to obtain the total average flame speed

In the second method the flame speed (V2) was calculated by dividing 19 into the flame arrival time at the last flame sensor In the thud and fourth method the distance between the tace ot the tunnel and the membrane position was ıgnoıed Depending on the test conducted, the membrane was positioned at 5 m and 7 m respectively from the face The flame speed calculation tormulae (V? and V_4) were used accoıdıngly When the membiane was positioned 5 m tiom the tace, the formula tor the flame speed (VO was [(l/(tf-t„) + l/(t_7-tft) + l/(t|9-tis)) / 14] and when the membrane was positioned 7 m from the face, the toimula tor the flame speed *(V\)* was [(l/(tx-tn) + l/fo-ts) + l/Uw-tix)) / 12] where 12 was the distance between the first sensor (t_s) and the last sensor dm)

When comparing the flame speed results it can be seen from the flame sensor readings, explosion videos and the test graphs (Figure 3) that it takes up to 200 milliseconds tor an explosion to develop During this time, the explosion develops inside the membrane, the methane/air mixture burns, and thereafter the methane explosion starts propagating from the membrane position onwards From this point ot view the flame speed V_4 = [13 / (t|.j - U] would be the most realistic approach to calculate the flame speed When using V_4 the distance between the face of the tunnel and the membrane position was ignored and in some tests, the active suppression system successfully stopped flame propagation, within the membrane As a result, some ot the results show no flame speed inside the test tunnel In this case, the formula V? was used instead to calculate the flame speeds where applicable, even though the accuracy ot the result is in doubt. In the Tables ot results that follow, the flame speed quoted has been calculated by method 4, i.e. (V_4)

As discussed earlier, the 42 tests that were conducted between July 1995 and December 1999 in the 20 m tunnel at the Kloppersbos Rescaich Facility can be categorised in two different ways:

- tests conducted without a suppression system
- tests conducted with a suppression system

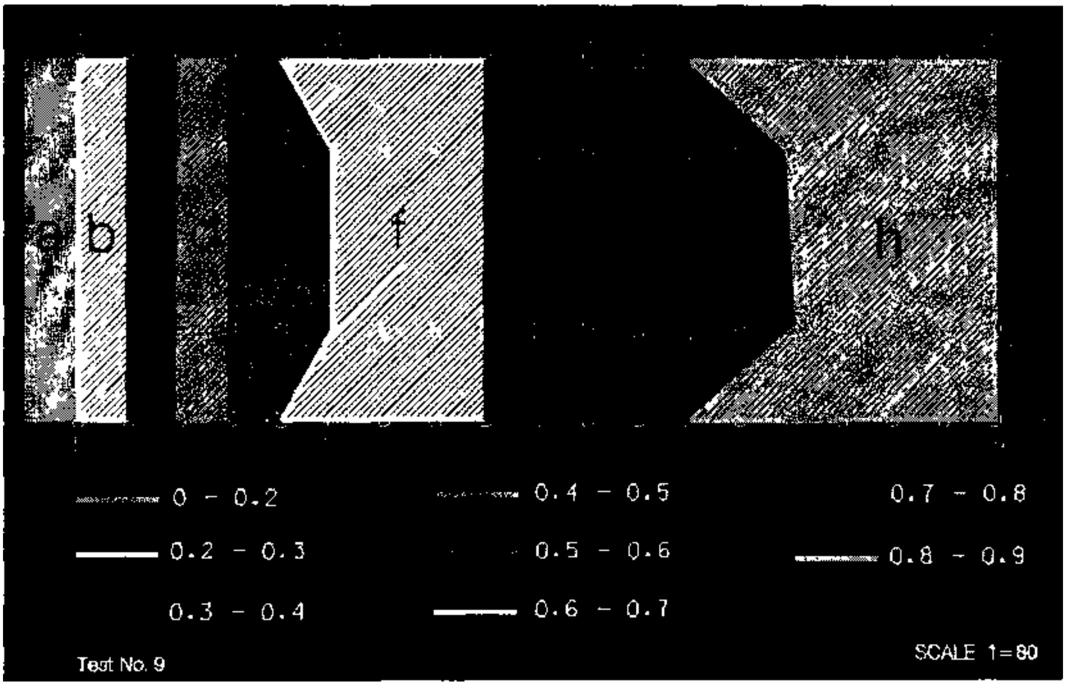
5 TESTS CONDUCTED WITHOUT A SUPPRESSION SYSTEM

There were tour ways to conduct tests without a suppiession system, namely:

- empty tunnel tests
- tests with a CM in place
- tests with a CM model in place
- tests with a CM model in place and with the shouldei in position

5.1 *Empty Tunnel Tests*

The empty tunnel tests results are shown in Table 1. Tests S to 12 lall into this category. Flammable gas mixtuies were generated using the mixing and monitoring procedures described by du Plessıs and Bryden (1997). Two concentrations were used; 9% and 12% and the volume ot the mixture was kept at 87,5 m throughout this series of tests Flame speed was found to be independent of the change in the methane concentration from 9% to 12%. The flame propagated throughout the test tunnel, which can be seen in Figure 6. The average flame speed was 45,2 m/s when 9% methane/air concentration was used and it was 44,9 m/s when 12% methane/air concentration was used. The highest flame speed was 53,4 m/s, which was recorded in test 12.



Fieuie 6 Flame piopjgjtion inside the lest tunnel The shading denotes the amval of the flame

5 2 *Tests with a Continuous Miner m Place*

Only two tests were conducted with a Joy 14 CM 6 piesent inside the lest tunnel: These were test 5 and 6 Flame speeds of 118,9 m/s and 69,2 m/s were recorded respectively The results of the tests conducted with the presence ot a CM are shown in Table 1 While usint; the same volume ot mixture that was used in the empty tunnel tests, only 9% methane/air concentration was used. Because the presence of the CM inside the test tunnel created an obstruction and reduced the cross-sectional area of the tunnel, the flame propagated more quickly. The 70% difference between the flame speeds could have been caused by experimental error, however, compared to empty tunnel tests, the flame speed increased by more than 100%. The average flame speed was 94,1 m/s.

Tiible I Flame speed without suppiesMon system

Empty Tan nel			Tes!I with ;n CM			Test with a CM Model			CM Model with a Should'er		
No	(%)	S (m/s)	No	(7<)	S (m/s)	No	(</<>)	S (m/s)	No	(%)	S (m/s)
8	9	30,3	5	9	118,9	13	9	68.4	28	7.7	55,1
9	9	51.9	6	9	69.2	15	7.5	28.1	37	7.8	117,5
10	12	41.7				16	9	70.1	38	7.8	101.9
11	12	48.1				17	7,5	35,8	39	7,8	118,2
12	9	53,4				18	7.5	33,9	40	7.8	109,4
						19	9	78,2	41	7.8	106.8
						20	9	79.2	42	7,8	109.6
						21	9	68.5			

No- Test No, (%): CH4/ Air %, S: Flame Speed. Volume CHj/Air = 87.5 m'

5.2 Tests with a Continuous Miner Model in Place

Eight tests were conducted with the presence of a CM model inside the test tunnel, the results of which are given in Table 1. Even though two methane/air concentrations were used, 7,5% and 9% respectively, the volume of the mixture was again kept at 87,5 m\ the same as for the two previous series of tests conducted.

In this category, the cutting head ot the CM model was positioned at one of three positions: on the floor, in the middle of the tunnel or at the ceiling (roof position). In tests 13, 19, 20 and 21, the cutter was on the floor. In tests 15 and 16, the cutter was in the middle of the tunnel, while in tests 17 and 18; the cutter was at the roof. The position of the cutter did not influence the flame speed.

One significant result from the tests conducted was that when comparing the results of tests conducted with 7,5% methane/air concentration and 9% methane/air concentration, it can be seen that the flame speed more than doubled. The reason for this was the change of the methane/air concentration from 7,5% to 9%. The average flame speed was 32,6 m/s when 7,5% methane/air concentration was used and it was 72,9 m/s when 9% methane/air concentration was used.

When comparing the results of tests conducted with the presence of a CM and CM model inside the test tunnel using a methane-air concentration of 9%, the average flame speed was 94,1 m/s when the CM was present inside the test tunnel, and 72,9 m/s when the CM model was present. Despite the fact that the average llame speed was 94,1 m/s when the CM was present inside the tunnel (with considerable variation in the results ot the two tests), the flame speed of test 6 was 69,2 m/s. This comparison shows that the CM model can be used as a replacement for the CM.

5.3 *Tests with a CM Model with Shoulder in Position*

Seven tests were conducted with both the CM model and shoulder in position as illustrated in Table I. Two methane/air concentrations were used: 7,7% and 7,8% and the volume of the mixture was increased to 105 m' throughout this series of tests. In this category, the cutter of the CM model was positioned in the middle of the tunnel.

Adding a shoulder inside the tunnel and increasin« the methane/air volume from 87.5 m to 105 m^1 affected the flame speed. 7,8% methane/air concentration was used for tests 37 to 42 and the average flame speed was 110,6 m/s. Only one test (test 28) was conducted with a 7,7%< methane/air concentration which resulted in a 55,1 m/s flame speed. This difference in flame speed of more than 100% between test 28 and the average flame speed of the rest of the tests in this category was probably due to an experimental error.

5.4 *Discussion of Results of the Tests Conducted without ci Suppression System*

A summary of the results of the tests conducted without a suppression system can be seen in Table 2. When comparing the average flame speed between tests with a CM model inside the test tunnel (9% methane/air concentration) and tests with a CM model with shoulder in position (7,8% methane/air concentration), the flame speed increased from 72,9 m/s to 110,6 m/s. This is almost a 52% increase in the average flame speed. The reason for this difference was as a result of the decrease in the cross-sectional area caused by the presence of the shoulder.

Although the flame speeds with a 7,8% methane/air concentration (CM model with a shoulder) was 40% faster than those at 9% (CM model without a shoulder), it can still be reasonably concluded that the tests conducted without a suppression system resulted in the most violent explosions when the methane/air concentration was 9% while weak explosions occurred when the methane/air concentration was 7.5%. The reasoning behind this conclusion is because of the absence of the shoulder in the tests conducted with a 9% methane/air concentration, thus the 40% difference in flame speed. Unfortunately no test with a 9% methane/air concentration with the shoulder in position were conducted to make a more accurate comparison.

Table 2. Average flame speed wiihoul suppression system.

Empty Tunnel		*Test with a CM*		*Test with a CM Model*		*CM Model with a Shoulder*	
(%)	S (m/s)	**(%)**	S (m/s)	**(%)**	S (m/s)	**(%)**	S (m/s)
9	45,2	9	94,1	7,5	32,6	7,7	55,1
12	44,9			9	72,9	7,8	110,6

TESTS CONDUCTED WITH THE SUPPRESSION SYSTEM

Seventeen tests have been conducted with an active on-board suppression system present inside the test tunnel. Test 0 was the first and only test conducted using a local on-board suppression system. For the other sixteen tests, an international system (Centrocen / DMT Explo-Stop System) was used. The Centrocen / DMT Explo-Stop System is a German based system which proved to be very effective in suppressing flame propagation. The tests conducted with the suppression system can be categorised in three different areas:

- on-board suppression tests with a CM
- on-board suppression tests with a CM model (full face)
- on-board suppression tests with a CM model with the shoulder in position

According to the test protocol for tests conducted under this category, the machine operator's position was 8 m from the face of the test tunnel.

6.1 *On-board Suppression Tests with a CM*

Only three tests were conducted with a CM (Joy 14 CM 6). Test 0 was the first test conducted using the local on-board suppression system with the shoulder in position. A 9% methane/air concentration was used and the volume of the mixture was kept at 87,5 m' for this test. This test caused severe damage to the test tunnel with a failure of the suppression system to stop the flame propagation. The fastest flame speed recorded was 189,9 m/s (Table 3). The flame propagated throughout the test tunnel. This vast disparity in the flame speed was caused by two factors: the 9% methane/air concentration and the presence of the shoulder.

Table 3 On-board suppression tesls with a CM and a CM model (full-face lests) results

Test with a CM	*Test with a CM Model (Full Face Test)*

No	(%)	S (m/s)	L	P	No	(%)	S (m/s)	L	P
0	9	189,9	>I9	Roofs&Sides	22	9	11,9	4	TLF
3	7.5	19,6	2	TLF	23	9	9,1	4	TLF
4	9	7,6	3	TLF	24	9	13,9	4	TLF and TRF
					25	9	17,7	4	TLF
					26	9	14,5	4	TLF
					27	9	14,4	4	TLF

TLF; Top left flame; TRF: Top right flame; SLF: Side left flame; SRF: Side right flame; L: Flame length: P: Flame position; Volume *CHJAu* = 87,5 m

The other two tests (test 3 and 4) in this category were full-face tests and the international system (Centrocen / DMT Explo-Stop System) was used. Two methane/air concentrations were used, 7,5% and 9% and the volume of the mixture was again kept at 87,5 m^1 throughout the tests. Both tests were successful and the flame stopped propagating at 2 m and 3 m respectively long before it could reach the operator's position. The highest flame speed was 19,6 m/s or about 10% of that in Test 0. The results of the on-board active suppression system tests with a CM machine are shown in Table 3. In Table 3, flame positions e.g. TLF (Top Left Flame), TRF (Top Right Flame), SLF (Side Left Flame) and SRF (Side Right Flame) indicates whether the flame has been detected on the sides and/or the roof of the tunnel.

6.2 *On-board Suppression Tests with a CM Model (Full- Face Tests)*

Six tests were conducted with a CM model (simulation of a Joy 14 CM 9). While using the same volume mixture that was used in the on-board suppression tests with a CM, only 9% methane/air concentration was used. A Centrocen / DMT Explo-Stop System successfully stopped flame propagation at 4 m in all six tests. The average flame speed was 14,2 m/s. The on-board suppression tests with a CM model (full-face tests) results are shown in Table 3.

6.3 *On-board Suppression Tests with a CM Model with the Shoulder in Position*

Seven tests were conducted with a suppression system on board the CM model and shoulder in position, the results of which are shown in Table 4. Two methane/air concentrations were used; 9% and 12%, and the volume of the mixture was increased to 105 m throughout this series of tests. In all the tests, the flame propagated beyond the operator's position as prescribed by the test protocol except test 36.

Tests 29 to 32 where 9% methane/air concentrations was used were partially successful and the flame propagated up to 3 m beyond the operator's position with an average flame speed of 23,8 m/s. In test 33 the flame propagated up to 19 m with a flame speed of 33,1 m/s while in test 34 the on-board suppression system failed to operate and the flame propagated throughout the test tunnel with a speed of 38,1 m/s. The overall average flame speed with *9%* methane/air concentrations was 25,7 m/s. Test 36 was a repeat of test 34 where the flame propagation stopped within 8 m.

6.4 *Discussion of Results of the Tests Conducted with the Suppression System*

Both the on-board suppression tests with a CM and with a CM model were successful and the flame propagation ceased before reaching the operator's position, except in test 0 which was a failure. All the tests that succeeded were full-face tests while the failure was a shoulder test.

The results of the on-board suppression tests with a CM model with the shoulder in position were all unsuccessful except for test 36. The reason for the failure was the presence of the shoulder in position as well as an increase of the methane/air volume from 87,5 m^3 to 105 m\ Therefore we can once again conclude that the most violent explosions occurred when shoulder tests were conducted.

Table 4. On-board suppression tesis results with a CM model with the shoulder in position.

Test No.	*CHJAir (%)*	*Flame Speed (m/s)*	*Flame Length (m)*	*Flame Position*

29	9	22.8	10	TRF
30	9	23,8	10	TRF
31	9	26.8	9	TRFandSRF
32	9	21,8	11	TRF
33	9	33,1	19	TRF
34	12	38.1	> 19	Roof and sides
36	12	4,6	8	TRF

Volume CHVAir = 105.0m*

When we compare the average flame speeds of the shoulder tests conducted with a suppression system and without a suppression system, the average flame speed of the tests with the suppression system where *9%* methane/air concentrations was used was 25,7 m/s while tests conducted without a suppression system with a 7,8% methane/air concentralions was 110,6 m/s, the results of which are given in Table 5. From these results we can conclude that even though there were failures with the shoulder tests, the suppression system slill reduced the flame speed by up to 76,8%.

Table 5 Average flame speed with suppression system

Test with d *CM*		*Test with « CM Model*		*CM Model >with a Shoulder*	
(%)	S (m/s)	(%)	S (m/s)	(%)	S (m/s)
7.5	19,6	9	14.2	9	25.7
9	7.6			12	21,4

7 CONCLUSIONS

It can be concluded from the tests conducted with an on-board suppression system that the Centrocen / DMT Explo-Stop System successfully stopped flame propagation inside the test tunnel. Despite a few failures, this system has a potential of significantly reducing the risk of harm to CM and RH operators involved in underground methane ignitions.

As expected, the most violent explosions occurred when the methane/air concentration was 9%. This was, in general, also the concentration that resulted in the highest flame speed. The presence of an actual CM or full-size models of a CM resulted in a very significant reduction in the tunnel cross-section and a consequent increase in flame speed. In fact, on the occasion of the first lest, the failure of the suppression system on a Joy 14 CM 6 resulted in massive damage to the test tunnel when 87,5 m of 9% methane/air mixture was ignited.

In South Africa, the coal mining operations are highly mechanized with more than 175 continuous miner machines in use. Even though the risk of a coal mine explosion can never be reduced to zero by a single line of defence, the Centrocen / DMT Explo-Stop System active on-board suppression system has the potential of stopping explosions and could be deployed in high risk areas to reduce the possibility of a coal mine explosion.

ACKNOWLEDGEMENTS

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Appraisal of Self-Contained Self Rescuers for Use in Australian Coalmines

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ABSTRACT: A laboratory and field trials were undertaken at lour active underground coal mines to evaluate the performances of Self-Contained Self Rescuers (SCSRs) in the normal Australian underground coal mine conditions. The results showed that the oxygen run out lime of SCRSs could be predicted using the body weight, average heart rate and exercise rating of the subjects. Additional factors that influenced oxygen consumption rate include speed of travel, intensity of work and the individual habits such as smoking, drinking.

INTRODUCTION

The self-contained self-rescuers (SCSRs) are breathing apparatus that make oxygen available to a wearer tor a nominal period of time. There are various types of SCSRs on the market currently deployed at underground coal mines. A study was undertaken to evaluate the performances of the units in the normal Australian underground coal mine conditions. The study involved a laboratory and field trials at four active underground coal mines using 37 volunteers. The aims of the laboratory/field trials are:

• To gather data on the duration of SCSRs, escape times, distances travelled and average heart rates to develop methodology to predict how much oxygen is actually needed by an individual to escape from a mine.

• To evaluate the influence of personal factors such as age, weight, physical fitness and previous experience on the duration of SCSRs.

• To evaluate the effect of environmental factors on the duration of SCSRs..

• To develop a system to assist collieries with practical escape planning which accounts for relevant issues pertaining to SCSRs. • To identify training issues relating to SCSRs.

• To develop a system to assist collieries with practical escape planning which accounts for relevant issues pertaining to SCSRs.

2 LABORATORY TREADMILL TESTS

The experimental set-up is shown in Figure I. Each subject breathed through a mouthpiece which was connected to an SCSR in a closed circuit using the flexible tubes. Both the 02 and C02 concentrations in the inhaled and exhaled air were monitored and recorded by Sensormedics Respiratory Gas Analysis System (SRGAS) at a sampling rate of one minute interval. In addition, the inhaled and the exhaled volume of air at one minute interval (minute ventilation) were also recorded. Six volunteers, aged between 21 and 53, were selected to walk on the fiat treadmill at 5 kın/lir wearing the SCSR. Heart rates of each individual monitored continuously throughout the tests. The Polar Vantage NVTM was set to sample average heart rate over 15 second interval during the test. The pressure relief valve installed on the breathing bag was used to release excess oxygen to the atmosphere. The test procedure consisted of:

1. The test subject donned the chest mounted heart rate monitor and the wristwatch receiver.
2. A functionality test was carried out on the heart rale monitoring system.
3. The test subject donned a mouthpiece and nose clip-.
4. The mouthpiece was connected to an activated SCSR.
5. The level treadmill was adjusted to 5 km/h
6. The subject walked on the treadmill until the SCSR exhausted its oxygen supply.

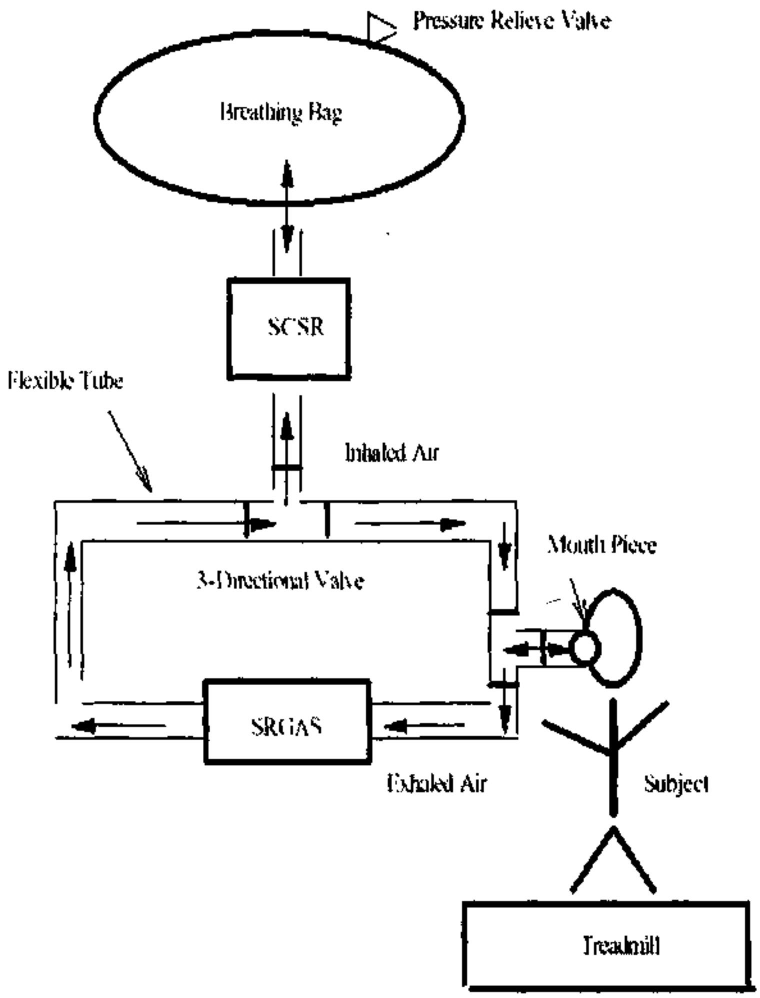


Figure 1. Experimental set-up

The criteria to terminate the test were:

- The breathing bag was totally collapsed.
- The subject's desire to terminate the trial.
- The subject showed evidence of excessive strain, for instance, the heart rate greater than 85% of cardiac reserve
- The inhaled C02 exceeded 3%.

3 TREADMILL RESULTS

3.1 Heart Rates

Dynamic heart rates of one of the six subjects walking on the treadmill are depicted in Figure 2.

In general, the heart rates of each subject increased rapidly during the first five minutes, and then stabilised at steady rate.

These effects corresponded to the natural exercise progression from standing on a treadmill, to walking and finally exercising.

The heart rates for the subject W2 was relatively stable while the other five subjects showed a steady increase of heart rates with elapsed time. This continuous increase of heart rate during exercising is called cardiovascular drift and caused by following factors:

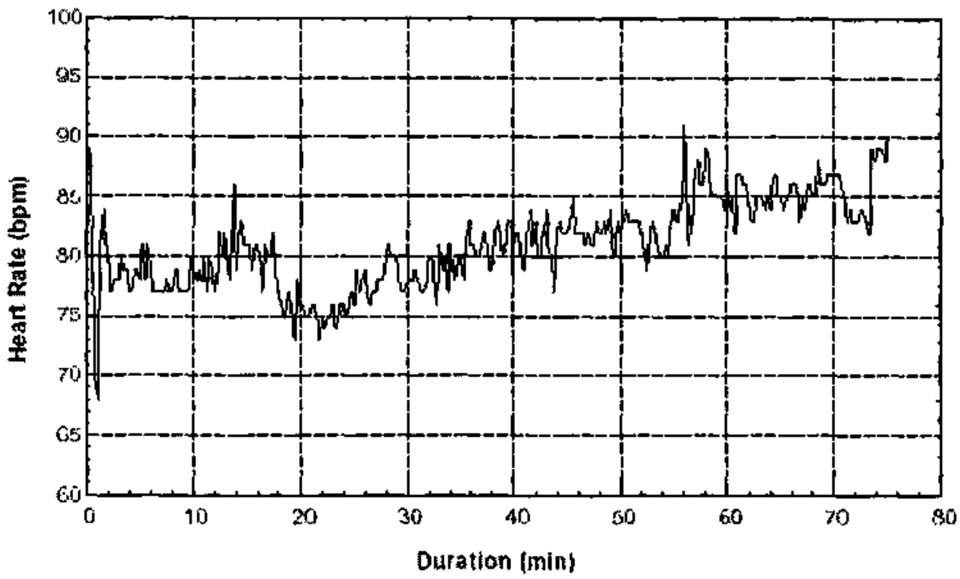


Figure 2. Heart Rate versus duration of a subject

- increase in body temperature during the exercise,
- a noticeable amount of body fluid was gradually lost,
- increase in body temperature during the exercise,

Table 1 summarises the treadmill results. All subjects were stopped when the unit ran out of oxygen. However, in case of the subject W5, the experiment was terminated due to the concentration of inhaled C02 continuously exceeding 3% prior to the breathing bag completely collapsing.

3.2 *Gas Analysis*

The inhaled and useable oxygen was calculated from the SRGAS recording sheet.

In a closed circuit, the exhaled air is made up of 02 and C02. The consumption of oxygen (V02) can be defined by the relationship

VO: = (Volume of Inhaled Air) * (Percent of Oj in Inhaled Air) % (Volume of Exhaled Air) * (Percent of O: in Exhaled Air)

Similarly, the amount of exhaled carbon dioxide (VCOi) generated reduces to the relationship

$VC0_2$ = (Volume of Inhaled Air) • (Percent of CO? in Inhaled Air) + (Volume of Exhaled Air) * (Percent of COi in Exhaled Air)

Since both gases were sampled at one minute interval, the amount of O2 and CO2 used by each subject is obtained by direct summations of unit time rates. The results summarised in Table 1 suggest that each subject consumed between 85 and 126 litres of oxygen with an average of 99.3 litres. From practical point of view, the 99.3 litres of useable amount of oxygen via SCSR supports the manufacturer's 100 litres per unit.

4 FIELD SIMULATED TRIALS

The objectives of the underground investigations were to gather in-mine data on escape times, istances and heart rates and to develop a technique to predict how much oxygen was actually needed for an average miner to escape from an underground mine. Field simulated escape trials were conducted at three collieries located in New South Wales and one in Queensland. The mines were selected to represent variety of conditions that are normally encountered in the actual escapeways of Australian coal mines. The escape routes were selected by the mine and based on the following requirements:

I. A distance that requires walking an excess of one hour.

Table 1 Summaıy ot treadmill lests

Subject ID	Age	Weight	Heart Ratet			Observed Time[1]	Observed VOi	Observed	Observed[1] XCOi	Useable Oiygen
			Minimum	Average	Maximum		(Indirect)'	(direct» '		
	<vrs)	(kR>	(bpm)	(bprm	(bpm)	(mfn)	flmin)	amin)	(Lirtn)	(liters)
Wj	43	**S3**	6S	K1.1	**91**	75.0	1.33	1.04	0.95	N'A
W,2	53	81	88	110.9	120	78.0	1.28	1.61	1.48	125.9
W?	21	82	121	128 8	139	68.0	1.47	1.28	1.18	SS.4
W*	48	70	80	97.4	105	67.0	1.49	1.27	1.17	84.9
Ws	44	**90**	102	135.0	149	65.0	1.54	1.46	134	94.6
W,	32	66	94	122.4	132	107.0	0.93	0.97	0.90	102.8
Minimum	2J.0	66.0	6S.0	81.1	**91.0**	65.0	0.93	0.97	0.90	84.9
Mean	**40.2**	**78.7**	**92.2**	**112.6**	**122.7**	**76.7**	**1.34**	**1J7**	**1.17**	*993*
Median	43.5	81.5	91.0	116.7	126.0	71.5	1.40	1.27	1.18	94.6
Maximum	53.0	90.0	121.0	135.0	149.0	107.0	1.54	1.61	1.48	125.9

Observed lime irom commencement of breathinu oxjgen via the SCSR to the cessation of the tea when the breathing bag is fully collapsed. Tins is the *Oi* •nun-out" time via an SCSR.

[2] Tlie indirect oxygen eonaimptian was chained by dhiding 100 liters divided by the observed time.

•' The direct oxygen consumption was calculated uüing Equalkn (4).

[4] The observed carben dioxide consumption *was* calculated using Equation (4)

2. An established escapeway to represent a typical underground escape route conditions.

Attempts were made to ensure that the profiles of the volunteers represented those of the current workforce in Australian underground coal mines. All the volunteers were medically screened and physically examined before the field trials.

During the simulated escape trials, the 37 volunteers walked along the escape route at their respective mines on Day 1 carrying the MSA Portal Pack SCSRs on the belts. The walking was repeated on Day 2 with the same subjects wearing the units. The tasks performed on the second day (Day 2) simulate the emergency situations at a typical mine.

All the subjects had been trained in the donning and use of the SCSR units prior to the field trials. The walking paces on both days were kept at a constant rate of about 3 km/hr. The influence of walking pace on the oxygen consumption was not investigated in this study. The heart rates of each subject was recorded by a Polar Vantage NV'V'.

At the end of Day 2 each SCSR's breathing bag was monitored to enable the oxygen "run out" time to be determined. Normally, the end of trials could be defined as the point of complete collapse of the breathing bag, often accompanied by an increase in breathing resistance and light headache. The latter was most likely caused by the inhaled air with a high concentration ot CO: in excess of 3%. The corresponding time from starting to breathe oxygen via the SCSR to the termination was observed, which is the so-called 02 "run-out" time.

After completing the field simulated trials, the subjects were asked to till out questionnaires designed to assess both the performance and the comfort of the SCSRs.

5 SIMULATED RESULTS

5. / *Age and Weight Profiles of Subjects*

81% ol all the subjects are between 30 and 49 years old The minimum and maximum ages of the sub|ccts aie 27 and 57 years old with an overall aveiage age of 40.7 years old. The age distribution ol the selected volunteers reflected the profiles of the current worktorce employed in New South Wales and Queensland.

87% subjects have a weight between 70 and 99 kg. The minimum and maximum weight of all the subjects are 66 and 130 kg with an average of 85.5 kg. The weight distribution of the selected volunteers represented the profiles of the current workforce employed in New South Wales and Queensland.

5.2 Performance and Comfort of the SCSRs

All the 37 volunteers were asked to complete questionnaire designed to assess the performance and comfort of the SCSRs. The following conclusions could be drawn from the questionnaire.

- All the subjects felt they could don and wear the SCSRs in an emergency.
- 95% felt the SCSRs would protect them of oxygen deficient or toxic atmospheres.
- 50% felt the SCSRs uncomfortable to wear.
- 56% would not wear the unit during the a normal shift
- 89% found the nose clips uncontestable.
- 56% felt the temperature of the inhaled air was comfortable and 44% found the temperature tolerable.
- 44%; found the breathing resistance to be comfortable with 56% tolerable.

5.3 *Oxygen "Riin-Out"Ttime*

The oxygen "run-out" time distributions for the 37 subjects ate shown in Figure 3.

It is noticed that the majority (81%) of subjects have an oxygen "run-out" lime between 50 and 69 minutes, few (11%) over 70 minutes and the very few (8%) less than 50 minutes.

5.4 Predicting Oxygen Consumption

Various studies in USA (Bernard et al, I979) have linked the oxygen consumption (VO2) to average heart rate (HR) and the body weight (W) as per the following equations:

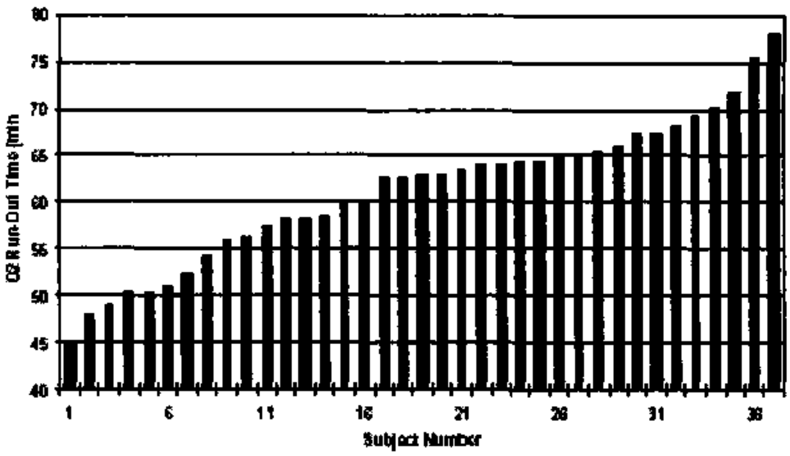
PSU Model $VO2 = (HR - 66) / 36$ (I)

Foster Model $VO2 = 0.24HR - 1.54$ (2)

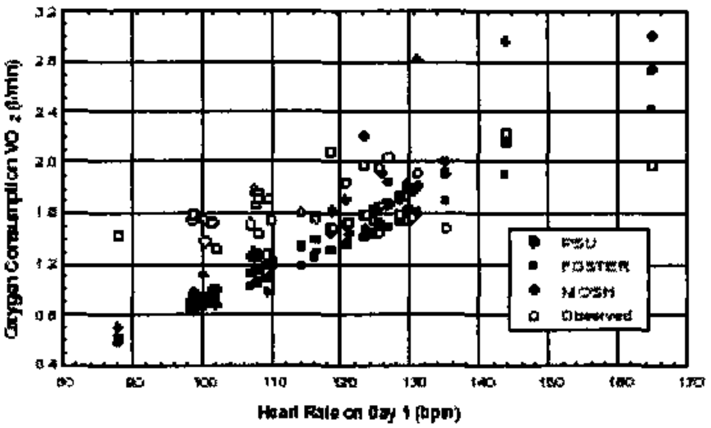
NIOSH Model $VO: = W(HR - 61.25) / 3230$ (3)

Each of the above relationships was based on statistical analysis of extensive laboratory work overseas. Using Day I average heart rates and the weights of the 37 subjects, the average oxygen consumption rate of each subject was estimated. The results with the three models are presented in Figure 4.

The three models generally underestimated the average VO: for average heart rates under 120 bpm and inconsistent at heart rates above 120 bpm. There is a strong possibility that the above models were developed under conditions which were dilTerent from the field simulated trials.



Fıguıe 3 Distnbution ot Oxygen 'Run-Out times



Fıguıe 4 Pıedıcted vs obseıved VO»

The fundamental question addressed heie is whether there is a relationship between VO: and average heait rate, age weight, smoking habit, dunking habit, late ot exercising and othei factors Theie was a tendency tor low values ot heart ıates and weights to be associated with low values ot VO: Subjects with low (poor) exercise ratings were associated with high VO:

The associations between VO: and other measured vanables could be described as follows

- strong association ot VO2 with body weight,
- modeiate association ot VO: with exercise ıatıng,
- weak association ot VO:with cigarette and alcohol consumption,
- very weak association ot VO: with age

It was observed that the heart rates on the two days for the same person were not the same, but with 0 79 coefficient ot correlation Both the compiled medical data and the simulated escape field data were statistically analysed to produce the following model

VO: = (3 25W + HR) / 250 + 0 023 (4)

Equation 4 relates oxygen consumption (VO:) with average heart rate (HR) and body mass (W) is based upon a repıesentatıve group ot Australian male underground coal miners This model appears to be of better pıedıctıve value than the previous models, therelore a better estimate ot oxygen consumption and hence predicted oxygen "ıun out' time. The predictive model given by Equation (4) is recommended for use by collieries because of its simplicity.

Equation (5) below relates oxygen consumption (VO2) with average heart rate (HR), body mass (W) and exercise rating (ER) of an individual and is based upon a representative group of Australian male underground coal miners.

$$V02=(7.39\ W + .74\ HR - 11.35\ ER)/500 + 0.26 \quad (5)$$

Equation (5) is presently of academic interest and requires standardisation with an inter- and intraobserver reliability study before its utility can be established. The method would require the assistance of a health professional experienced in assessing the quantity and quality of an individual's exercise history. The method is based upon the subject's weekly exercise habits, intensity of exercise and scored on a scale of 1 to 10. It may be prone to errors generated, by both the subject and the observer.

6 CONCLUDING REMARKS

The MSA Portal Pack SCSR produces about 140 liters of oxygen but only about 100 liters oxygen is used by each wearer.

A fit person may use less oxygen from SCSR in comparison to a less fit person. This implies that more unused oxygen is released to the atmosphere by a fit person.

Using results from field simulated escape trials, a linear model has been developed to predict oxygen run-out times of self-contained self rescuers. The field results clearly demonstrate that for a fixed work rate, oxygen consumption by an individual depends upon the following personal factors:

- Body weight has a major influence.
- Physical fitness has some influence. Physically fit people produce oxygen in excess to requirements. This excess is lost through the relief valve and although oxygen consumption is less, this only has a slight effect on oxygen run-out time.
- Age may have slight influence on oxygen consumption depending on the level of physical fitness and the degree of cardiac strain.
- Experience in wearing the breathing apparatus appeared to have no influence.

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A Method for Continual Monitoring the Natural Ventilating Pressure

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ABSTRACT: A new information on continual monitoring the natural ventilating pressure on the basis of temperature measuring is here submitted. Thermometers of mine air are placed in downcast and upcast shafts while their measured levels are transmitted to the control computer. The values of barometric and fan static pressures are simultaneously transmitted to the same computer where all the input data are processed by means of computer algorithm. In such a way obtained results can be continually used at any time.

I INTRODUCTION

Natural ventilation pressure (NVP) is the very important consequence of mining thermal energy. Especially it becomes significant in cases, where ventilation is running with combination of main fan (MF) and natural ventilation, and where the colliers are very deep.

Deep mines in the Czech Republic can be found in geographical region where daily and seasonal ranges ot surface air increase the mechanism of natural ventilating effect. The authors have known many ways how to analyse NVP, however for economic ventilation controls, continuous monitoring of running MF and NVP values are necessary.

Correct selection for solving possibility is minimized, so the balance U-tube can be used in downcast and upcast shafts as well as a set of transmitting thermometers where only dry temperatures are measured.

2 DESIGN THEORY

Mine Darkov has been situated within the Ostrava-Karvinâ coalfield with the most significant Czech hard coal deposits, see Figure 3. Mine Darkov had performed the preparation and equipment installation in terms of total main ventilating reconstruction. The aim was to work-out the computer algorithm of continual monitoring NVP in the local mine conditions. This method was chosen on the basis of skip or cage operating experience in the downcast (DC) and upcast (UC) shafts. However, the balance U-tube method has, been very questionable and unreliable.

On the other hand the wetness in UC is often linked to water droplets that are not suitable for wet temperature measuring. Therefore the simplest formulation of enhancement pressure directly to bottom of shafts from measured dry temperature was chosen.

The determination of NVP has been used in hydrostatic method, so-called mean density method (McPherson 1993)

$$NVP = g.H.(\rho_{md} - \rho_{mu}) \qquad [Pa] \tag{1}$$

where:

ρ_{md} is the average density in the DC
ρ_{mu} is the average density in the UC
H is the vertical depth of shafts

The average densities are calculated by means of simplified scheme of Darkov Mine ventilation system, see Figure 1.

$$\rho_{md} = \frac{\sum_{i=1}^{i=5} \rho_i . a_i}{H} \qquad [kg.m^{-3}] \tag{2}$$

$$\rho_{mu} = \frac{\sum_{i=6}^{i=13} \rho_i . a_i}{H} \qquad [kg.m^{-3}] \tag{3}$$

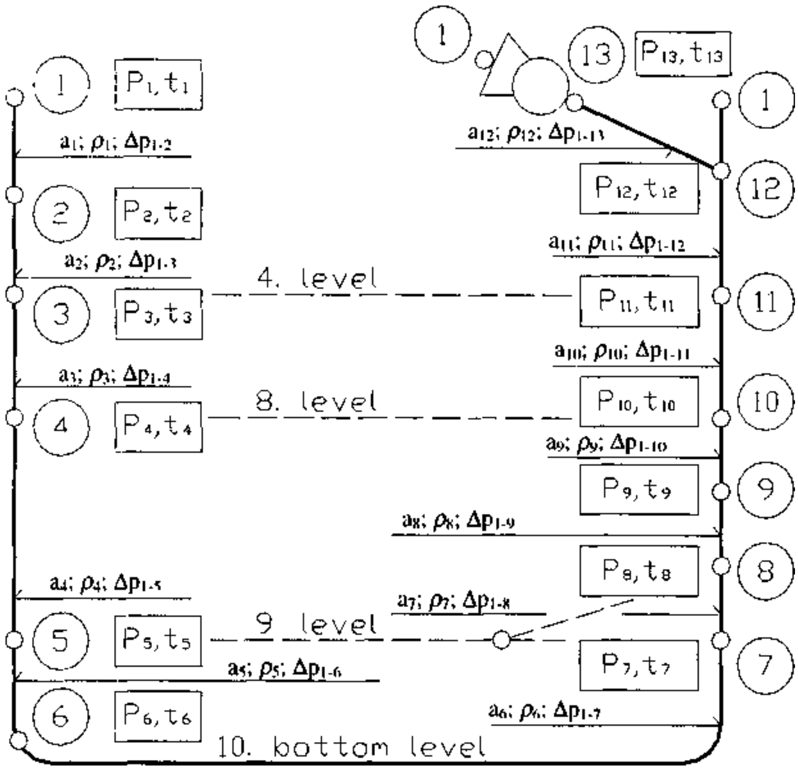

Figure 1 Darkov Mine Ventilation system

where:
pi to ps are middle densities in the sections of DC
ai to as are vertical lengths in the sections of DC
p_6, to pn are middle densities in the sections of workings and UC and under the same conditions the pressure in point No 2 is as follows (further for dry air in section 1 between points No I and No 2 is valid):

$$P_2 = b.\exp\left[\frac{0{,}03416.a_1}{0.5(546.3+t_1+t_2)}\right] \quad \text{[Pa]} \qquad (4)$$

ar, to au are vertical lengths in the sections of workings and UC

$H = \sum_{i=1}^{i=5} a_i = \sum_{j=6}^{j=13} a_j$ = 886 m, is a vertical depth of both shafts in Mine Darkov conditions.

$$\rho_1 = \frac{0.00348.0.5[b+(P_2-\Delta p_{1-2})]}{0.5(273.15+t_1+273.15+t_2)} \quad [\text{kg.m}^{-3}] \qquad (5)$$

where for equation (4) and (5):
11 and t: are temperatures in points No 1 and No 2
b is the barometric pressure
Pi is the pressure in point No 2
Api-: is the pressure difference between points No I and No 2

Further the calculation will be led in equations for densities pi to pn in the sections ai to au, see Figure 1.

Therefore, formulas (01), (02) and (03) give the NVP mine action. It means that the amount of NVP depends on continuous measurement of temperatures li to lu, barometric pressure b, MF airflow Q_v and pressure difference Δp_{13} from suction part of the main fan.

The control screen shows all monitoring data including fan operating point (FOP), characteristic curves of MF and activated characteristic resistance of colliery with NVP, see Figure 2 and Figure 6. In other words the FOP is determined by MF airflow (Qv = 145 m'\s") and pressure difference Api.i = 2138 Pa), both are measured continually.

The system of operating point (SOP) is identified by equation,

$$Q_v = \sqrt{\frac{1}{R}\left[\Delta p_{13} - (\pm NVP)\right]} \qquad [m^3.s^{-1}] \qquad (6)$$

where R is the resistance of colliery.

In the expression (06) it is apparent that the "V" apex of the activated parabola is - for the constituent effect - situated in the minus part of vertical pressure different coordinate and vice versa.

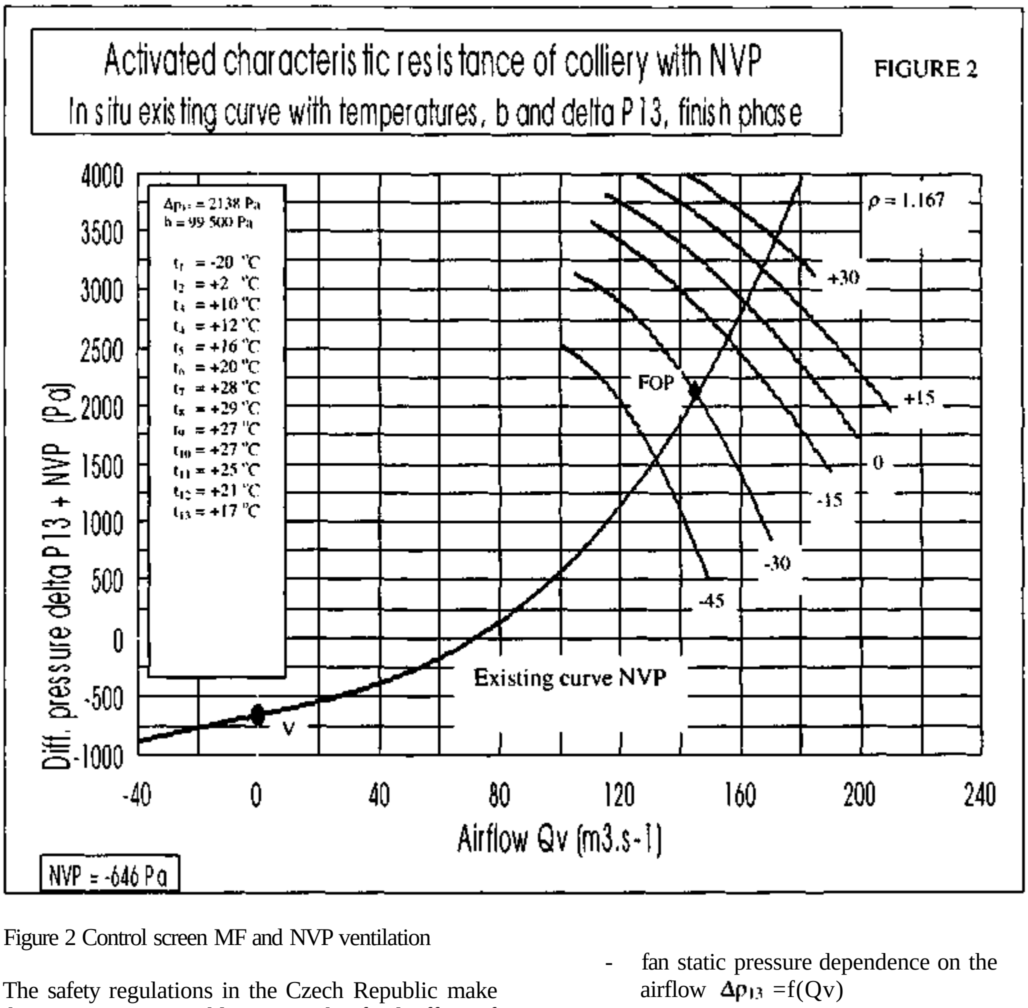

Figure 2 Control screen MF and NVP ventilation

The safety regulations in the Czech Republic make the MF measuring obligatory. The final effect of the measurement is the output in three forms of fan characteristic curves (Kopâfiek. 2001), as follows

- fan total pressure dependence on the airflow $\Delta p_{tv} = f(Q_v)$
- fan static pressure dependence on the airflow $\Delta p_{13} = f(Qv)$

  fan motor power input dependence on the airflow Pimi = f(Qv)

Now the dependence $\Delta p_{13} = f\ (Q_v)$ becomes the base on control screen MF and NVP ventilation.

see Figure 2 and Figure 6. The set of Darkov Mine MF regulation curves take the form of equation.

$$\Delta p_{13} = A_0 + A_1.Q_v + A_2.(Q_v)^2 \qquad [Pa] \qquad (7)$$

where Ao, Ai and A_2 are multinomial coefficients of the second rate.

It is important to say that the set of equation (7) is valid for standard density p_0 = 1,2 kg.m''^{3}. On the' contrary, the density in the point No 13 is as follows:

$$\rho_{13} = \frac{0.00348.(b - \Delta p_{13})}{273.15 + t_{13}} \quad [kg.m^{-3}] \qquad (8)$$

Therefore it is useful to modify the MF equation (7) into a new form:

$$\Delta p_{13} = \frac{\rho_{13}}{\rho_0}\left[A_0 + A_1 Q_v + A_2 (Q_v)^2\right] \; [Pa] \qquad (9)$$

It means that on the basis of monitoring p^ by equation (8) the MF curves (7) becomes the so-called "floating", in accordance with the resistance of colliery by equation (6).

The photos from Darkov Mine plant - from underground and surface place - are shown:

Figure 3 Overview of Darkov Mine

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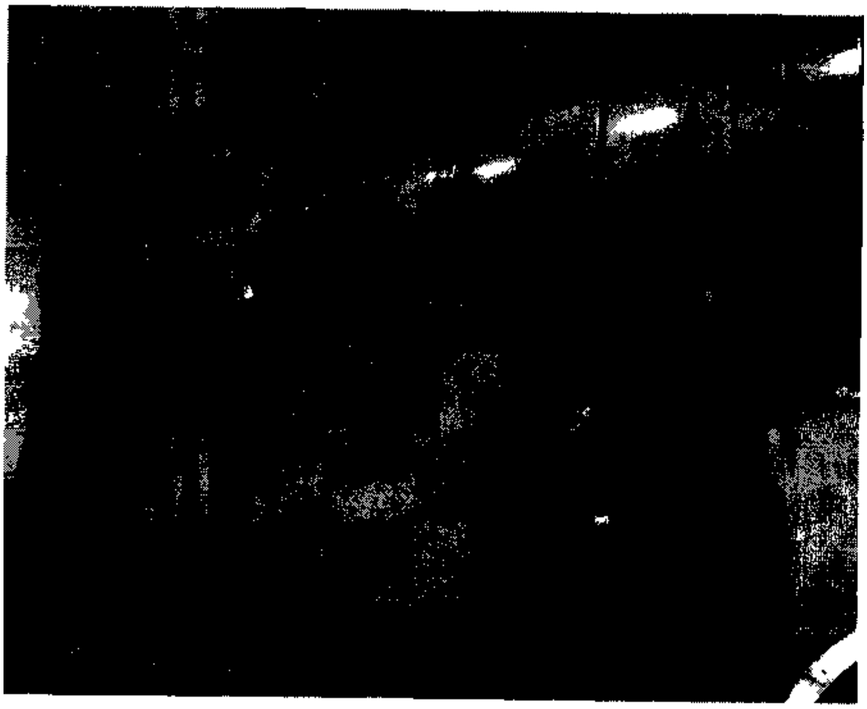
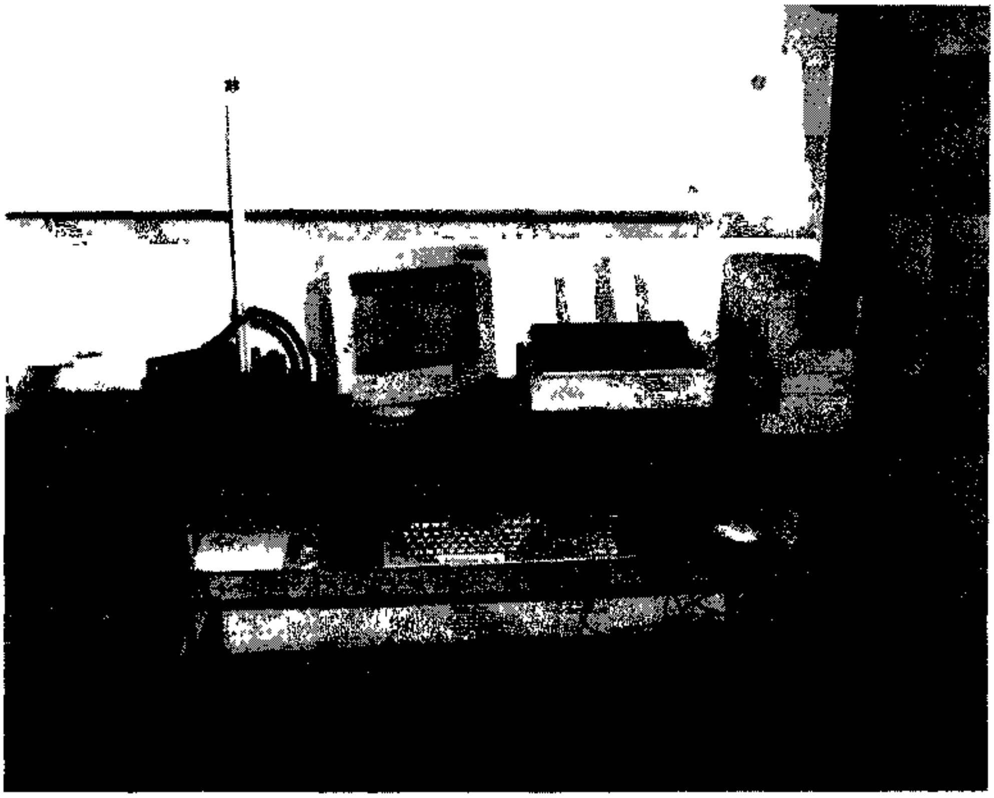


Figure 4 Overview of Darkov Main Fan axial type ARA-2-^550

Figure 5 Theimometei point No 6 honzontal 886 m the bottom of upcast shaft



Fıguıe 6 Control PC of MF and NVP in Manager Ventilation office

3 CONCLUSIONS

The authors have shown one ot the methods ot continually determined natuial ventilating pressuie and main fan running It is important to say that operating results are very hopeful tor economic return ot investment lor future At present it is possible to assign saved energy of main plant ventilation There are simple and reliable opeıatıons led fiom control loom ot Ventilation Manager, see Figure 6

ACKNOWLEDGEMENT

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Estimation of Fugitive Dust Impacts of Open-Pit Mines on Local Air Quality - A Case Study: Bellavista Gold Mine, Costa Rica

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ABSTRACT: Fugitive dust impacts of Bellavista open-pit gold mine located in Costa Rica was investigated. The fugitive dust emissions from the gold mine were identified and then quantified using emission factors. The impacts of the fugitive dust emissions were estimated by means of FDM air quality dispersion model developed by the USEPA. TSP and PM|„ concentration estimates obtained from the FDM model were compared with the Costa Rican and World Bank air quality standards. The fugitive dust impacts from the gold mine were found to have relatively insignificant impact in a confined area centered around the mine site.

I INTRODUCTION

Dust is a generic term that describes particulate matters suspended in the atmosphere. It does not make any distinction between the size, shape, and chemical composition of the particulate matters. Fugitive dust refers to dust that is derived from non-point sources and does not have an easily defined source. It is formed when particulate matters become airborne due to turbulent action of the wind, mechanical disturbance of fine materials, or release of gaseous emissions laden with particulate matters in an unconfined manner.

There are various dust generating activities and processes in the mining industry. Sources such as material storage silos, process boilers and heaters, onsite power generators etc. are the major point sources in the mining industry. The exhaust gases emanating from the mining equipment and other vehicles operated within the mine site are mobile sources of particulate matters. But the most predominant sources of particulates in the mining industry are the fugitive dust sources. Fugitive sources comprise earth moving and material handling operations, crushing, screening, milling, blasting and drilling, haul roads, and wind erosion of exposed surfaces. Fugitive sources are more difficult to control as compared to point sources.

Mine dust can cause serious nuisance and aesthetic deterioration in the surrounding environment and communities. Fortunately due to relatively large particulate matter sizes associated with the mining emissions and the relatively short release height of the pollutants, such negative impacts are usually confined in relatively small areas. Within these areas of impact, fugitive dust may result in damage to the vegetation and agriculture. The deposited particulate matter may block the plant leaf stomata hence inhibit gas exchange, or smother the plant leaf surfaces reducing photosynthesis levels (Environment Australia, 1998).

Besides the impacts on vegetation, health effects of particulates on mine personnel and public may also be significant. The inhalable fraction of dust (i.e. PM|o, particulate matters of aerodynamic diameters less than I0 |im) passes through the nose and mouth, and is easily deposited in the trachea and bronchial section of the lungs. Respirable dust (i.e. PM_{25}, particulate matters less than 2.5 u.m diameter) penetrates unciliated airways in human lungs, and lodges in the alveolar region (ISO, 1995). Depending on the chemical and physical characteristics of the particulate matters, there may be significant health effects. Dust containing heavy metals, certain silica and asbestos forms are known to have increased adverse health effects.

It is important to identify and quantify the fugitive dust emissions and impacts associated with mining operations early in the planning stage. Such an approach to mine fugitive dust emissions may help control the dust emissions and improve environmental performance of the proposed mine. Emission and air dispersion models are widely used to estimate the fugitive dust emissions and impacts.

• The current study focuses on the investigation of fugitive dust impacts of an open-pit gold mine in Costa Rica. The fugitive dust emissions from the proposed mine were estimated by means of emission factors. The subsequent atmospheric dispersion of particulate matter was investigated by means of a numerical model, FDM, developed by the US Environmental Protection AgencyTUSEPA).

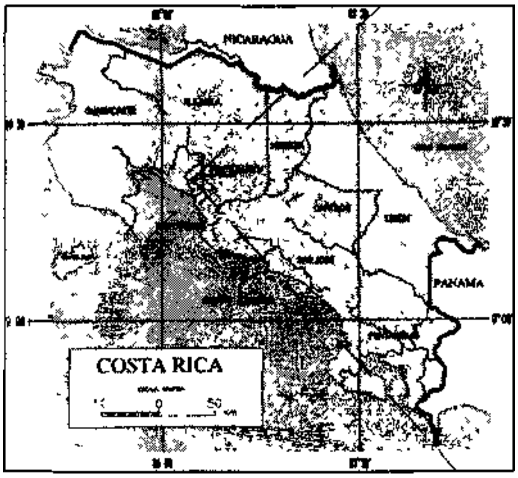
2 PROJECT DESCRIPTION

Bellavista gold mine is situated in the Puntarenas Province of Costa Rica (Fig. 1). The mine site is located approximately 3 km northeast of the town of Miramar. The fugitive dust impacts of the mine are of significance for the residents of Miramar and the forested area northeast of the mine site.

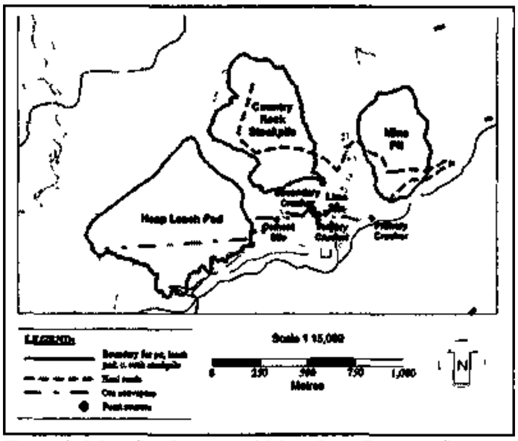
Bellavista gold mine has total mineable reserves ot 11.2 million tonnes with an estimated project life ot 10 years. Ore production will be approximately 1.6 million tonnes per year at its maximum capacity. Gold ore will be mined by conventional open-pit methods utilizing mid-size earth moving equipment. Approximately 2.2 million tonnes of waste rock per yeai will be excavated during the open-pit mining.

Blasting will be conducted daily in the mine pit. The broken waste rock and ore will be hauled using 50-tonne capacity haul trucks. The waste rock will be hauled to a stockpile area located approximately one km northwest of the open-pit (Fig.2). The ore will be hauled to the stockpile on the south end of the pit, from where it will be loaded into primary crushers.

Transport of the ore between the crushers will be accomplished via conveyors. Lime will be added to the ore before the secondary crusher to prevent crusher plug-ups and screen blinding. Following the tertiary crushers, ore will be placed into either low-grade or high-grade ore bins. From the ore bin, high-grade ore will be transferred to the milling plant for further size reduction. Low-grade ore will be transferred to the agglomerator where high-grade ore and cement will be added to form stable agglomerates. The ore agglomerates will be transported to the leach pad via overland conveyors feeding a line of standard portable "grasshopper" conveyors to a portable long-leg conveyor and a radial stacking conveyor



Figuie !• Location of the Proposed Bellavista Gold Mine



Figuie-2. Mine Site Layout and Major Dust Emission Sources

Cyanide solution pumped from the carbon adsorption-desorption-recovery (ADR) facility will be applied onto the leach pad. The collected pregnant leach solution will be sent to the ADR facility to recover precious metals from the pregnant solution. The gold in the pregnant solution will be removed by carbon in adsorption columns. The loaded carbon columns will be stripped of its gold content using a heated caustic-cyanide solution. The caustic-cyanide solution will be heated using diesel-fired hot water boi'x The gold will then be plated out in an electrowinning cell. The gold plating from the cathodes of the electrowinning cell will be refined into doré using diesel-fired pot furnace. The spent carbon from the columns will be regenerated in a diesel-fired reactivation furnace. Power requirements of the mine will be met by seven 1000 kW diesel-fired generators.

3 DUST EMISSIONS AND CONTROLS

Particulate matter emissions will take place during the construction, operation, and closure phases of the mine project. The emissions will occur in varying particulate size ranges. The two size ranges considered in this study are the inhalable (PM_{10}) and total suspended particulate (TSP) matters. TSP roughly covers the range of particles with aerodynamic diameters less than 45 urn. TSP inherently encompasses the inhalable and respirable tractions, as well as the coarser fractions. The coarse fraction of TSP has more of an aesthetic significance than human health significance. The respirable fraction ($PM_{2.5}$) of particulate matter was not considered in this study since mine dust mostly consists of the coarse fractions.

3.1 Construction Phase Emission Sources

During the construction phase of the project, which will last about a year, particulate matter will be released from the point, mobile, and fugitive sources. In general, the nature and spatial orientation of the emissions during the construction phase will be similar to the operational phase emissions. The construction activity and pollutant emission rates will be less than the full-scale mining operation rates. Therefore, construction phase emissions were not assessed in detail. It is conservatively assumed that construction phase air emissions and related impacts would be at most equivalent to the operational phase emissions and impacts.

3.2 Operation Phase Emission Sources

The emission sources during the mine operation will be from point, mobile, and fugitive sources. The major point sources will be:

- Power generators;
- Desorption column boiler;
- Doré furnace;
- Carbon regeneration furnace;
- Cement and lime silos;

The major fugitive dust sources will be:

- Primary, secondary, tertiary crushers/screens;
- Haul and various mine roads;
- Transfer points on the ore conveyors.
- Drilling and blasting in the mine pit;
- Materials handling (loading/unloading, bulldozing etc. in/on open-pit, waste rock stockpile, leach pad, coarse ore stockpile, and various other material aggregates);
- Wind erosion of the exposed surfaces (i.e. leach pad, coarse ore stockpile, waste rock stockpile, etc.)

Major mobile sources of air contaminants during the mine operation will be several 50-tonne haul trucks, front-end loaders, bulldozers, backhoes, utility trucks and pickups.

3.3 Closure Phase Emission Sources

The emission sources during the closure phase will be reclamation activities. The plant buildings will be demolished and the mine site will be graded and revegetated. The particulate matter emissions will be due to surface activities and mobile sources. The closure phase emissions will be relatively insignificant and will have short duration.

3.4 Emission Control Measures

The proposed mine will implement several mitigation measures to reduce air emissions. The proposed emission control measures and expected control efficiencies are listed in Table 1. Dust from the haul roads, one of the major sources of fugitive dust, will be controlled using water trucks. During the dry periods, the roads will be kept wet to reduce particulate matter emissions. The pollution controls listed in Table 1 achieve efficiencies as high as 99.5%, when fully implemented.

Table 1 : Air Pollution Control Measures.

Souice	Control	Efficiency
Crushes s	High pressure waler sprays	90.0 %
Haul loads	Dust suppression using water	90.0 %
Conveyors	Covers	85.0 %
Silos	Dust collector	99.5 %

3.5 Emission Rate Estimates

Particulate matter emission rates for the operation phase were estimated using emission factors published by the US EPA (IJSEPA, 1995a). Emission factors are pollutant quantity estimates that relate pollutant generation rate to mine activity levels (e.g. grams of pollutant per tonnes of material transferred or produced). But for some fugitive dust sources, such as material handling operations (e.g. bulldozing, unloading etc.) or wind erosion such a direct relation may prove difficult to establish. For such fugitive sources, meteorological conditions (e.g. wind speed, precipitation) and physical properties of materials (e.g. silt and moisture content) are important and needs to be addressed to estimate emissions.

Annual operation phase fugitive dust emission estimates for the Bellavista gold mine are given in Table 2. These values reflect the emissions resulting after the pollution controls listed in Table 1 are implemented. The overall mine TSP emissions are estimated to be 114 tonnes/year. PMm emissions (62 tonnes) constitute 54% of TSP emissions. Fugitive souices constitute 82% and 66% of overall mine TSP and $PM_{|0}$ emissions, respectively.

Table *2:* Annual Fugitive Dust Emission Estimates.

Source	Type'	PM|„ (kg/year)	TSP (kg/year)
Cement Silo	P	10	10
Lime Silo	P	2	2
Combustion Sources	P	9.215	9.215
Crushers & Screens	F	7.524	19.627
Conveyors	F	8.750	21.876
Haul Roads	F	6,005	16.681
Pit Operations	F	1.621	6,516
Blasting & Drilling	F	13.188	16.924
Waste Rock Dump Opeiallons	F	492	2,078
Leach Pad Opei allons	F	118	663
Ore Stockpile Operations	F	608	2.410
Wind Erosion	F	2.219	6,163
Mobile Sources	M	11,794	11.794
Piiint Sources Total		*9.226*	*9.227*
Fugitive Source. Total*		*40.525*	*92,9M*
Overall Mine Emissions		*61,545*	*11.1959*

P- Point source; F- Fugitive source; M: Mobile source.

The particulate matter emissions due to blasting and drilling to be conducted in the open-pit constitute 15-20% of the overall mine emissions. Although, these emissions constitute a major portion of the overall emissions, the amount that would be dispersed into the surrounding environment will be less due lo retention of the open-pit. To account for pit retention effects, Equation 1 is used to estimate the escape fraction of the fugitive dust generated within the pit confines (USEPA, 1995b):

$$\varepsilon_i = \frac{1}{\left(1+\frac{V_g}{\alpha \cdot U_r}\right)} \quad (1)$$

where;

U, = approach wind speed at 10 m (m/s)
V_g = gravitational settling velocity (m/s)
a = proportionality constant (0.029)
e, = escape fraction for size class i

The particulate matter emission rates presented in Table 2 were estimated by assuming the maximum possible mine activity rates. In this manner, worst-case emission rates were obtained. The mine operational schedules (e.g. holidays, shift hours, heavy rain days), as well as the meteorological conditions (i.e. wind speed, precipitation) were taken into account in estimating the emission rates. The particulate matter emission rates were obtained from the uncontrolled emission rates by applying the control efficiencies listed in Table 1. The uncontrolled overall mine TSP and PM|<, emissions are estimated to be 494 tonnes and 206 tonnes per year, respectively.

4 REGIONAL METEOROLOGY

Detailed meteorological information about the mine area is necessary to estimate particulate matter emission rates and atmospheric dispersion of the emissions. Air quality dispersion models usually require hourly meteorological data covering at least one-year period. In this study, data for the following parameters were used:

- Wind speed,
- Wind direction,
- Ambient temperature,
- Rainfall amount,
- Atmospheric stability class.

Wind speed, wind direction, ambient temperature, and rainfall amounts are measured parameters, whereas, atmospheric stability class is a derived parameter. Atmospheric stability is a measure of the dispersal potential of the atmosphere. There are various methods for estimating the atmospheric stability. Meteorological Processor for Regulatory Models (MPRM). developed by the USEPA (USEPA, 1996) was used to process measured hourly meteorological data to estimate the atmospheric stability.

Daily rainfall amount and number of days with rainfall are required for particulate matter emission estimation. Days with rainfall amount greater than 0.25 mm are considered to be wet days where fugitive dust generation is zero. For fugitive dust generation, a threshold wind speed criterion of 5.4 m/s is also applicable. Time periods with wind speeds higher than this criterion have higher fugitive dust generation potential.

The meteorological data required for emission and dispersion estimates was obtained from a newly established on-site meteorological station. The on-site station did not a have complete year's worth of data, and therefore, was augmented by the Puntarenas Station 18 km southwest of the mine site.

The predominant wind directions at the mine site are north-northeast (NNE) and northeast (NE). These directions are in agreement with the local topography. Stream channels for the Rio Rastra and the Rio Ciruelas lie in a southwest (SW) to northeast (NE) alignment. Annual average wind speed is 3.0 m/s. The frequency of hourly wind speeds exceeding 5.4 m/s is approximately 16%. The combined occurrence frequency of wind speeds higher than 5.4 m/s and daily precipitation amounts less than 0.25 mm is 12%. This indicates that most of the high wind speed events coincide with the dry periods. Such conditions are conducive to high fugitive dust generation.

The average monthly rainfall is given in Figure 3. The rainfall pattern observed at the mine site is defined by the influence of the Inter-tropical Convergence Zone (ITCZ), northeastern trade winds, and polar air masses. During the dry season observed between the months of December and April, the ITCZ is located south of Costa Rica and the polar air masses dominate the region. The polar air masses lose their moisture content while ascending the Tilaran Mountain Range on the Caribbean side. While descending the Tilaran Mountain Range on the Pacific side the polar air masses become cool and dry. The dry season ends when the ITCZ returns to 10° North latitude. The average annual rainfall for the region is approximately 2950 mm (Herrera, 1998). The highest rainfall occurs in the months of August, September and October.

5 AIR QUALITY REGULATIONS

Costa Rican general health law prohibits all actions, practices or operations that deteriorate the natural environment or alter the composition or intrinsic characteristics of its basic elements, especially air.

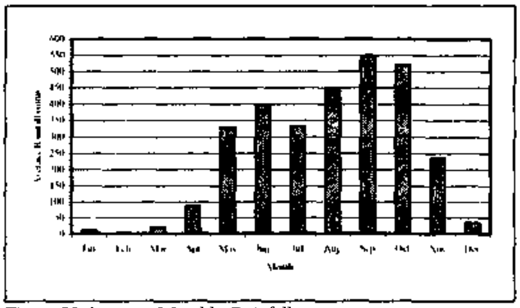


Figure-.V Average Monthly Rainfall

At present, few Costa Rican ambient air quality standards are available. In the absence of Costa Rican standards, international World Bank standards are used to evaluate the impacts of the proposed mining operation. The Costa Rican and World Bank ambient particulate matter standards are given in Table 3.

Table 3' Ambient Particulate Matter Standaids.

Souice	PM„, (lig/m')	TSP (Mg/m[1])	Averaging Penod
Costa Rican	85	150	Annual
Woild Bank	50	80	Annual
Woild Bank	150	100	Daily

6 ESTIMATION OF FUGITIVE DUST IMPACTS

In order to determine the impacts from a facility on local air quality, an air dispersion modeling study has to be conducted. This study is accomplished by use of mathematical models that relate emission rates to ambient air concentrations. Air quality models use meteorological data, emission rate and source data, and topographical and surface cover data to estimate ground-level pollutant concentrations. For the current study, impacts of the particulate matter emissions from the mine operations were estimated using Fugitive Dust Model (FDM) developed by the US EPA (USEPA, 1992). The pollutant concentrations estimated using the air dispersion model are compared with the established national/international air quality standards.

6. / *Description of the FDM*

FDM is a computerized air quality model specifically designed for computing concentration and deposition impacts from fugitive dust sources. The sources may be point, line, or area sources. Each source type may be treated as virtual volume source with an initial vertical mixed extent. The model has not been designed to compute the impacts of buoyant point sources, thus it has no plume-rise algorithm. Particulate matter emissions for each pollutant source are apportioned into a series of size classes. A gravitational settling velocity and a deposition velocity are calculated by FDM for each size class. Particulate matter concentrations and depositions are then computed at receptor locations.

Particulate matter mass depletion is achieved by the process of dry deposition. FDM accounts for deposition through two parameters: the gravitational settling velocity and the deposition velocity. As the name implies, the gravitational settling velocity accounts for removal of particulate matter from the atmosphere due to gravity. Since only the larger particles have sufficient mass to overcome turbulent eddies, this mechanism is significant only for the larger size ranges (e.g. particles greater than 30 (im). The deposition velocity accounts for removal of part.cles by all methods, including turbulent motion, which brings the particulate matters into contact with a surface and allows it to be removed by impaction or adsorption at the surface. It is known that for smaller particles the deposition velocity is significantly different from the gravitational settling velocity, while for large particles they are roughly the same. In the FDM, the emission rate is divided into a number of particles size classes. Each of the classes has a unique gravitational settling velocity and deposition velocity. The method used by the model to compute the gravitational settling velocities and deposition velocities is modeled after the work of Sehmel and Hodgson (1978). Key variables to the method are the roughness, height and the friction velocity.

6.2 Modeling Methodology

Emission sources discussed in Table 2 were modeled as point, area, and line sources. The modeled sources are also presented in Figure 2. Lime silo, ceiT,ent silo, crushers, and ore conveyor transfer points were modeled as point sources. Haul roads were modeled as line sources, made up of 30 individual segments. The emissions occurring within the boundaries of the mine pit, country rock stockpile, heap leach pad, and coarse ore stockpile were modeled as area sources. All of the pollutant sources were modeled as virtual volume sources with emissions distributed equally within the first 10 m above the ground surface. This height is consistent with the general mine operations where emissions are initially distributed vertically due to turbulence caused by surface irregularities present at the mine sites.

The particulate matter emissions from each source were varied hourly. The emission rates were adjusted according to shift hours, workdays, rainy days, and high wind speed events. Each emission source was represented by 8760 hours (24 hours x 365 days) of emission rate. The particulate matter emissions were segregated into two major particle size class categories, namely PM|(> and TSP (PM_{45}). Additional particle size classes (i.e. 2.5, 5, 15 and 30

|iin) were included in the settling and deposition velocity calculations, but their concentrations were not estimated since there are no established international standards.

The project site has a complex topography with elevations ranging from 500 to 1100 meters. In order to account for the complex topography and thick vegetative cover, a surface roughness length of 2 m was used to represent the modeling area.

7 MODELING RESULTS

7. / *Annual Concentrations*

The annual average ground-level concentrations resulting from the FDM runs were plotted as isopleths. The isopleths for PM_m and TSP concentrations are presented in Figures 4 and 5, respectively.

The maximum annual average ground-level PM|„ concentration due to mine operations was estimated as 12.7 pg/nr^1. This maximum concentration was located immediately southeast of the waste rock stockpile (Fig.4). The influence of the PM|,i emissions from the mine was very limited in terms of area coverage. The majority of the PM_m impacts occurred within the mine property. The concentration increase within the direct area of influence due to proposed mine operations was less than 5 lig/m^1. The increase in the indirect area of influence (and the town of Miramar) was less than I |ig/nr\ All the estimated PM m concentrations are much lower than the Costa Rican (85 pg/nr^1) and World Bank (50 pg/nr^1) annual standards (Table 3).

The maximum annual average ground-level TSP concentration due to mine operations was estimated as 22.2 pg/nr^1. The maximum TSP concentration was in the same location as the maximum PM|(I concentration. The concentration increase due to proposed mine operations within the direct area of influence was less than 10 pg/nr and within the indirect area of influence was less than I pg/nr^1. All the estimated TSP concentrations are much lower than the Costa Rican (150 Hg/m^3) and World Bank (80 pg/nr^1) annual standards (Table 3).

7.2 *Daily Concentrations*

The predicted daily ground-level concentrations within one year were ranked according to their numerical values without giving consideration to the date and location. The top-50 concentrations from this ranking are presented in Figure 6. The highest predicted daily concentrations were 76.4 pg/nr and 44.7 pg/nr^1 for TSP and PM,,,, respectively. The locations of these maximum daily concentrations coincided with the location of the maximum annual concentrations. Both the predicted daily TSP and PMio concentrations are well below the World Bank standards.

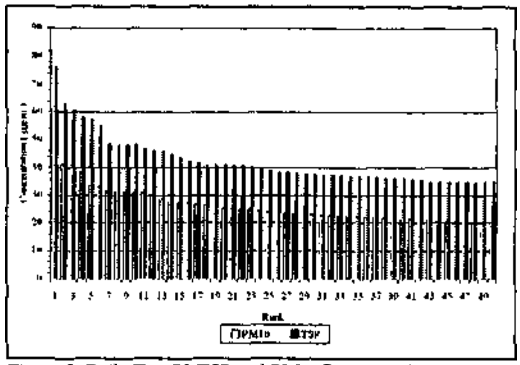


Figure 6: Daily Top-50 TSP and PM,,, Concentrations.

8 CONCLUSIONS

Based on the predicted particulate matter eoncentrationa, it can be concluded that most of the impacts will be within the mine site area. These impacts will not be very significant and will meet the Costa Rican and the World Bank ambient air quality standards. The fugitive dust impact to the direct area of influence and the indirect area of influence will be insignificant. The annual incremental PMio and TSP concentrations due to mine operations will be less than 1 pg/m'^{1} in the town of Miramar and the northeastern forest area.

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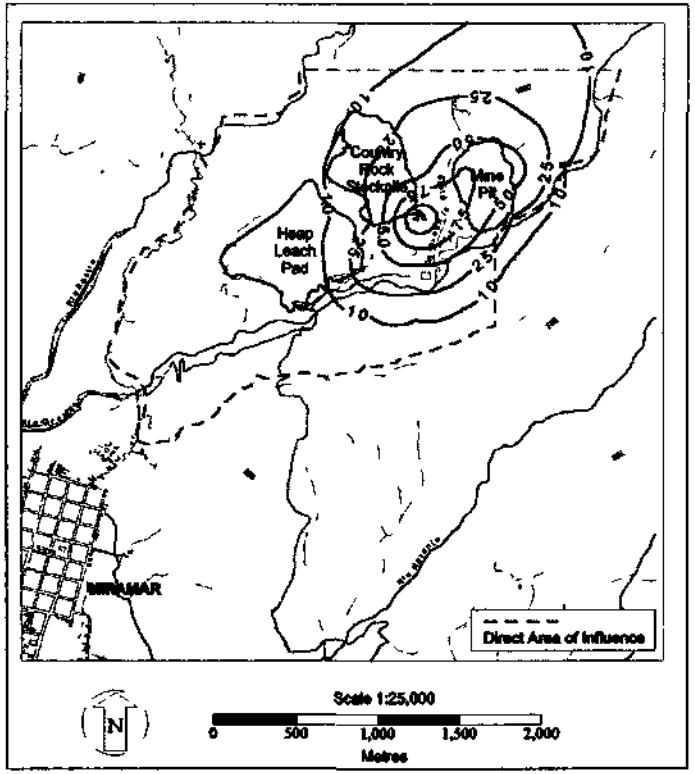

Figure 4 Estimated Annual Average PM_{10} Concentrations ($\mu g/m^3$)

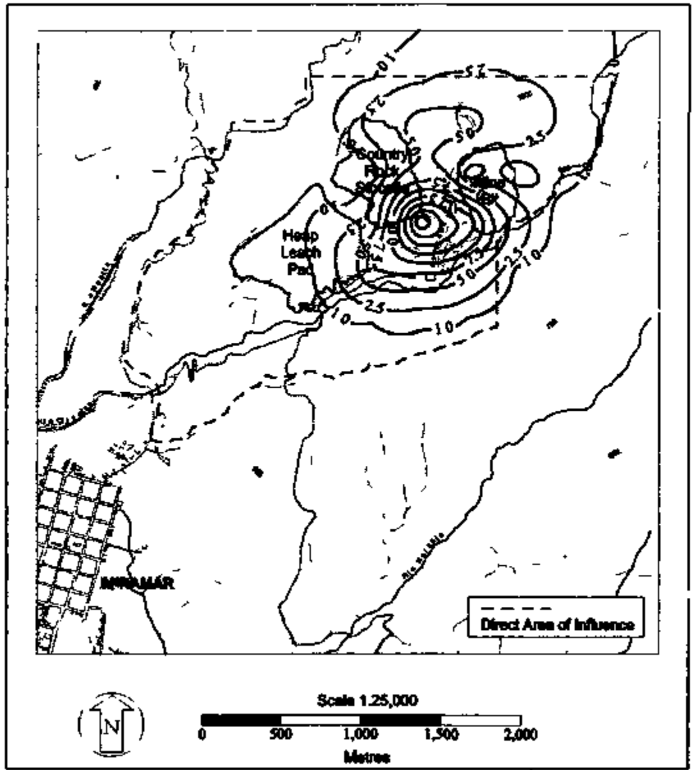

Figure 5 Estimated Annual Average TSP Concentrations ($\mu g/m^3$)

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Rockburst and Fall of Ground Investigations in Deep Level Gold Mines: South African Example

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ABSTRACT: South Africa's mineral industry is supported by an extensive and diversified resource base. It has a third or more of the world's reserves of alumino-silicates, chromium, gold, manganese, platinum-group metals vanadium and vermiculate etc. This large reserve base allows it plays an important role in the world in respect of the production and exports of many minerals and also some processed mineral products. Rock related accidents have been the major problem in the South African mining industry for many years. According to the latest South African statistics, 50 % of the accidents are rockfall and rockburst related (rock-related). Although three has been a steady a decline in accidents in the industry, the rock-related accident rates are still far from being satisfactory. The author conducted considerable number of rock-related accident investigations and inquires in various deep level gold mines, and the conclusions of the accident investigations and inquiries revealed that most of the accidents occurred as a result of lack of support units in the working face area, poor mining practice, lack of hazard identification, poorly designed mining and support layouts and lack of strata control training for workers. The main objectives of this study are firstly to describe the structure and importance of the mining industry for the South African economy and secondly to introduce and determine the causes of the rock-related fatalities in the South African gold mines.

I INTRODUCTION

South Africa's mineral industry is an important contributor to the country's economy. South Africa's total primary mineral sales revenue in 2001 was $13 billion, and the major foreign earnings in 2001, remained the platinum group metals (32,70 %) overtaking gold for the first time, followed closely by gold (31.90 %) and coal (18,90 %•)• Mining contributes around 7,5% directly to GDP and an estimated 15% through associated multiple effects.

2 STRUCTURE OF THE SOUTH AFRICAN MINING INDUSTRY

For more than a century. South Africa's mineral industry, largely supported by gold, diamond, coal and platinum production, has made an important contribution to the national economy. Furthermore, it has provided the impetus for the development of an extensive and efficient physical infrastructure and has also contributed greatly to the establishment of the country's secondary industry.

The mineral industry is a well-establishment and resourceful sector of the economy, and has a high degree of technical expertise and the ability to mobilize capital for new development. Table 1 and table 2 indicate that the mining sector is recognized worldwide as a leading and reliable supplier of a large variety of minerals and mineral products of a consistently high quality. In 2001, some 55 different minerals were produced from 775 mines and quarries, of which 69 produced gold, 83 coal, platinum 18 and 58 diamonds. Mineral commodities were exported to 92 countries. According to the 2001 statistics, 407,000 people arc employed in the mining industry.

The government's economic policies arc based on principles of private enterprise and free-market mechanism. The system has enabled the mineral induslry lo develop without undue state influence, thereby allowing market forces to dictate the pattern of its development. The Department of Minerals and Energy Affairs (DMEA) is responsible for the administration of the Minerals Act, 1991 and Mine Health and Safety Act, 1996, which regulate the prospecting for, and optimal exploration, processing and utilization of minerals, safety and health in the mining industry etc. respectively.

The DMEA's mission is to provide for effectual governance of the minerals and energy industries for economic growth and development thereby improving the quality of life of the propel of South Africa.

The Chamber of Mines of South Africa is voluntary membership, private sector employers' organization founded in 1889, three years after gold was discovered on the Witwatersrand. The chamber is an association of mining finance companies and mines operating in the gold, coal, diamond, platinum, asbestos, lead, iron ore, antimony and copper mining sector.

The National Union of Mine Workers (NUM) is the biggest union in South Africa, which was formed in December 1982. The NUM is the largest recognized collective bargaining agent representing workers in the mining and electrical energy sectors.

Table I: South Africa's mineral industry s ıcserves. 2(XX)

Commodity	World reserve °/t	**World ranking**
Alummo-silicate	37.4	1
Chrome ore	68,3	1
Gold	35.0	1
Manganese	80.0	1
Coal	10.6	5
Phosphate	7.2	3
Platinum	55.7	1

Table V South Africa's role m world mineral exports, 2000

Commodity	World production %	**World ranking**
Alumino-sihcate	49.8	1
Chrome ore	28,9	1
Coal	11.9	3
Manganese	28.5	1
Vanadium	69.0	1
Zirconium	32.3	2

3 MINE HEALTH AND SAFETY ACT, 1996 (Act No 29 of 1996)

The Act is dedicated solely to health and safety within the mining industry, which was not the case with the amended Minerals Act. The main objectives of the Act are;

- to provide for employee participation in the matters of health and safety;
- to protect health and safety of persons at mines;
- to provide effective monitoring of health and safety conditions at mines;
- to provide for investigations and inquiries to improve health and safety at mines; and
- to require employers and employees to identify hazards and eliminate, control and minimize the risks relating to health and safety at mines.

4 ACCIDENTS IN SOUTH AFRICAN MINES

The South African mining industry has for the past years been trying to reduce the expenditure on all major and minor cost in an attempt to either keep profits up to acceptable levels, or in many cases, to prevent the mine from making a loss. A big pressure in the filed of heath and safety has also been experienced for unions, employers and state to see a significant reduction in the fatality and injury frequency rates. The type of accidents in the South African mining industry both fatality and injury are indicated in the table 3. The safety situation is still far from being satisfactory as per Table 4.

Table 3: Type of accidents in South African mines, 2000

Type of accident	**Fatality**	**Injury**	Total accidents
Rock related	142	1265	1296
Machinery	12	229	239
Transport	45	920	960
Electricity	6	46	48
Fires	0	4	4
Explosives	3	47	48
Heat sickness	2	16	18
Gas/fumes	18	31	38
Conveyance	4	32	34
General	53	2138	2166
Rate	0.72	11.92	
Total	285	4728	4851

General: miscellaneous, occupational diseases, diving sickness, inundation, struck by objects, slipping, falling etc.

Table 4: Total accidents in South African mines. 2001

	1997	1998	1999	2000	2001
FATAL	424	366	315	285	301
INJURY	7100	6056	5488	4728	4722

Table 5- Total injuries & fatalities in specific mines. 2001

Year	1999	1999	2000	2000	2001	2001
	I	F	1	F	1	F
Gold	4202	21.3	3549	173	3372	192
Coal	207	28	213	31	171	17
Pt	765	39	638	46	795	50
Otheı	314	35	332	35	422	27
Total	5488	315	4728	285	4760	301

Table 6: All mines rock-ielated accidents. 2000

	1996	1997	1998	1999	2000
Fatal	247	192	181	137	142
Injury	2184	2012	1819	1517	1265

Table 5 also indicates the fact that most of the accidents occur in the gold mines. As it can be seen in the table 6 most important issue in the mining industry is the *rock-related* accidents such as roekburst and falls of ground.

In this research, most attention will be focused on the South African gold mines, which not only has higher total casualties than other hard rock mines, but also experiences more severe hazards, notably a high incidence of roekburst in the deep level gold mines.

4.1 South African Gold Mining Industry

Gold is synonymous with South Africa. Approximately 31% of the world's gold has been mined in the country over the past decade. Today, the Goldfields from a discontinues arc, 430 km long, stretching the Gauleng, the North-West, the Mpumalanga and the Free State provinces. In 2001, 395 t of gold (at 5-6 g/t average grade) was produced by primary gold mines, tailings re-treatment operations and as a by-product of the production of the other metals.

Gold mining in South Africa, from its humble beginnings in the first recorded mine in Eesterling in the Northern Province in 1871 to its pre-eminence as the largest gold mining industry in the world, has played a significant role in the economic development of the country over the past 120 years. Through gold mining, many towns and cities have come into being. Notable example is Johannesburg. Much of the infraslruclural development of roads, electricity generation, water reticulation, telecommunications, housing and the development of industry to provide the inputs to the gold mining industry have resulted directly from gold mining.

Most gold mining companies exploit more than one reef-vein in the Witwatersrand Supergroup. Further exploration, although at a reduced level, is expected to ensure that recent production levels are maintained for at least the near future. Precise age estimation in the Witwatersrand Basm is difficult since the rocks were deposited by sedimentation approximately 2700 million years ago. before the age of fossils. Experts believe that a great inland sea existed in what is now the Highveld and the Free State plains. Successive layers of conglomerate containing pebbles and gold were washed down into the sea and spread over the bottom by wave action. The gold particulars subsequently settled in successive layers of pebbles along the shoreline of this sea which layer sitted up.

4.2 Mining gold in South Africa

South Africa's thin but extensive gold reefs often lie several kilometers beneath the earth's surface and usually slope through the ground at up to 25°. The country's gold mining industry has to sink the deepest mine shafts in the world sometimes close to 4 km in-depth in order to that miners can reach and extract these reefs. The mining method in the deep gold mines is the 'longwall' mining method and the mining operation is carried out in hard brittle quartzitic rock, often at extreme depth. The great bulk of this rock mass behaves clastically, but stress concentrations around the excavations cause stable, as well as sometimes unstable, fracturing to take place. The vertical component of the virgin stress m South African mines tends to increase with depth approximately 0,027h(MPa)-rock density 2,75 t/m^3 in each meter. The rock temperature can reach up to 50 C° and this requires use of ice and refrigeration facilities.

Today, with the tremendous pressure on profit margins in the gold mining industry, which is mining steadily declining grades at ever-greater depths, there is more, emphasis on mechanization then ever before. Among the many aspects of mechanization, which are the focus of on-going research, are technologies like trackless mining, backfill, non-explosive breaking and hydropower.

On average, only 5 ppm of every ton of ore mined are actually gold. It is therefore necessary to separate the precious metal from the more than 100 million tons ore milled each year in South Africa. The Carbon-in pulp (CIP) method, which is increasingly widely used, makes use of the tremendous physical affinity "activated" carbon has for gold, which it readily attracts to its surface in cyanide solution. After smelting which takes place on individual mines, bullion bars containing about 85% gold are then taken to the Rand Refinery near Johannesburg and processed to either 99,5% purity or 99,9% purity to meet specialized demands from certain industries.

Despite the fact that the gold industry's contribution to mining and the economy has declined, it remains a vital sector in the South African economy as it can bee seen in the table 7. Gold mines provided 185,000 direct job opportunities in 2001. Moreover, through its linkages in the domestic economy, about 220,000 additional jobs are also maintained in the rest of the economy. The direct contribution in terms of salaries and wages amounted to $1,7 billion during 2000. South African gold production is pioneer in the world and will still remain as world-class gold producer in 2T[l] century.

Table 7- South African gold output. 2002

Year	Fine Gold- ton
1998	464
1999	449
2000	427
2001	393
2002	395

4.3 Accidents in South African Gold Mines

In 2001, 192 fatalities occurred in gold mines in which 50 % of these fatalities were rock related as per table 8. Accident statistics in South African gold mines cannot be compared with other countries' due to the following reasons:

1 - Most of the gold mines are currently working in 2500 km below surface.

2- Deep mines subjected to very high states of rock stress and cause seismicity and rockbursting.

3- In all deep mines, the exposed rockwall s are highly fractured.

4- The heavy faulting encountered in many deep gold mines and generates strata control problems and seismicity.

5- Gold mines in South Africa are the largest employees in the industry and employ more than 185,000 workers.

Table X. Rock lelated fatalities in the South African gold mines. 2000

FATALITIES	YEAR			
	97	98	99	00
GRAVITY	73	81	68	57
ROCKBURST	SO	62	41	48
TOTAL	153	143	109	105

In South Africa rock-related accidents are classified in two groups. Firstly *gravity accidents,* which accidents occur as a result of mainly unsupported of a rock or portion of fractured rock fall in the working environment. Secondly, *rockburst* accidents, which occur, as a result of stress-strain build up in the rock face or geological discontinuities such as fault or dyke and sometimes cause fatality and/or damage to underground workings.

In gold mines rockfall and rockburst accidents represent the most important cause of all fatal accidents. The important finding of the investigations revealed that 72% of all rock related accidents were seismic, and 28% of them were gravity related accidents. In 25 accidents, total death toll was 38. All gravity related accidents could have been prevented if the support had been installed prior to the accidents. Accident investigations and inquiries also revealed that most of the damage mechanism of the seismic related accidents could alsobeen minimized if the designed of the mining layout and support had been adhered by the production staff. Other important finding of these investigations was that most of the seismic related fatalities occurred between the support units due to seismic shake down.

5 ACCIDENT INVESTIGATIONS IN MINES

J. / *Accidents to be reported & investigated*

When an accident causes the immediate death of any person(s) as a result of rockfall or rockburst, the place where the accident occurred cannot, without the consent of the inspector of the DMEA, be disturbed or altered before such place has been inspected by an inspector, unless such disturbances or alteration is unavoidable to prevent further accidents, to remove corpses and injured persons or to rescue persons from danger, or unless the discontinuance of work at such place would seriously impede the working of the mine or works: provided that should an inspector assigned by the Chief Inspector fail to attend within three days after

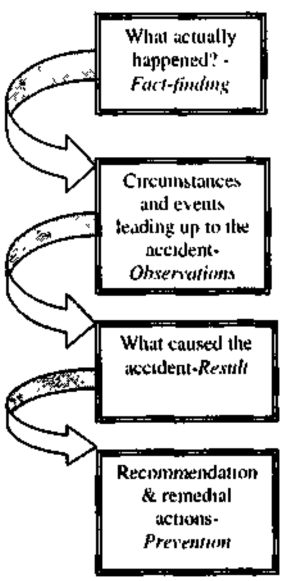
N»,

X

notice of the accident has been given, work may be resumed at the working place concerned.

In terms of the Mine Health and Safety Act, 1996 (Act No. 29 of 1996) the Chief Inspector of Mines of the DMEA instructs an inspector to investigate any accident or occurrence at a mine that results in any death, serious injury, any occurrence, practice or condition concerning safety of person(s) or any actual or suspected contravention of, or fail to comply with, any provision of the Act.

Method of Accident Investigation



6 SEISMICITY AND ROCKBURST IN SOUTH AFRICAN MINES

The term "rockburst" first received official recognition in 1924, with appointment by Government of the terms of reference were "to investigate and report upon the occurrence and control of rockburst in mines and the safety measures to be adopted to prevent accidents and loss of life resulting therefore".

Rockburst can be defined as "seismic event which involves brief, violent movements of the rock mass and which causes fatality and noticeable damage to an excavation."

Mining excavations induce elastic and then inelastic deformation within the surrounding rock mass. The elastic strain energy accumulated in a portion of the rock mass may be gradually unloaded due to the passage of mining, or it may be released gradually or suddenly during the process of inelastic deformation. Therefore, seismic event is a "sudden inelastic deformation (release of the strain energy stored in the rock mass) within given volume of rocks i.e. seismic source that radiates detectable seismic waves."

Table 9: Rock ıelated accidents, death and injuries in all mines. 2000

	Accidents	Killed	Injured
Giavity	993	94	943
Rockburst	303	43	322
TOTAL	1296	142	1265

Rockburst has been a matter of concerned on the South African Mines especially in deep level gold mines for many years. Whilst the total number of injuries and fatalities has been dropping steadily, the rates have remained essentially constant for many years. Table 9 shows that the number of fatalities resulting from rockburst is 30% of all rock-related accidents. Table 8 indicates that 105 rock-related fatalities took place in gold mines, which is almost 75% of all rock-related fatalities that took place in all South African mines. Table 8 also shows that 46% of all rock-related accidents that took place in gold mines were rockburst related. Mines is South Africa are planning to extract ore at depth of 4.5 km's and deeper in the next 10 years, it is clear that serious steps are needed to minimize the risks implicit in mining at a great depth. In essence the safety of the underground workers is paramount.

6.1 Seismic monitoring in gold mines

Most of the seismicities in the South Africa mining region is mining induced. Most of the seismic events are categorized as being face driven, geological driven (local) and regional driven. That is why most of the African deep level gold mines are equipped with seismic network system for warning, prevention and design purposes.

The recognition of the hazards posed by seismicity can be quantified through seismic observation i.e. experience over time in a particular environment. That is by having seismic information from particular environment for a period of time. The hazard of large events associated with major geological discontinuities can also be inferred by knowledge of the structure and the mine layout. It can be recognized without having had prior seismic information, which then implies that it can be estimated even before mining starts in an area i.e. 'non-monitor' recognition of hazard substantial research investment is being made in South Africa to understand the physical processes, the development and evaluation ol early warning concept

6 2 Seismic emission and rockburst control in gold mines

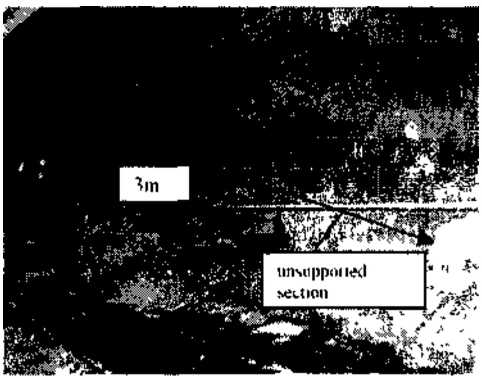
Mines must take all reasonable procedures and techniques adopted to prevent 01 reduce seismic emissions. These can be achieved by implementing stabilizing or bracket pillais, backfilling, proper mining configuration and sequencing, limitation of excess shear stress (ESS) on geological feature, mining of dykes, face shapes, limitation of energy telease rate (ERR), face advance rate, remnant iemüval, mining of dykes, mining away from structures, hydraulic props, seismic monitoring for prediction or any other preventive procedures.

7 CASE EXAMPLES

Case 1 Rockburst damage to tunnel

A seismic event with magnitude ML= 1,5 caused 2 fatalities and considerable damage to a tunnel 2,000 m below surface as per photo 1. The tunnel concerned was located some approximately 20 m beneath stabilizing pillar which was being mined out at the time of the accident. The tunnel was shotcreted and rockbolted in the some section of the tunnel, and the crew was in the process of drilling additional rockbolts prior to the accident. During inspection at the scene of the accident, approximately lm oi sidewall ejection along 15 m of the tunnel was evident 15 previously installed old rockbolts were sheaied as result dynamic loading.

A key factor controlling the distribution of the damage was the type ot support in place at the time of the accident. It was observed that some section of the tunnel had been left unsupported (mesh & lacing + lockbolts) tor a long time, and pillar extraction process in a highly stressed area above the tunnel was in progress, and this allowed further deterioration and stiess build up aiound the tunnel.



Figuie I Sidewall ejection as aiesult ot lockbuist

Case 2.Fall ofsround-support/backfill

The stope was located some 2,500 m below surface when the falls of ground accident occurred as per photo 2. 3 workers were struck by falls of ground during stope face examination prior to the accident.

It was observed that the stope panel concerned had already been backfilled in conjunction with pre-strC'Sable timber elongates approximately 6m from the face. There was no any other support unit within 6,5 m distance at the time of the accident. The fall out thickness was 1,5 m and extended along the 30m along the longwall face. Most evident issue was that the permanent support distance-backfill was far beyond the mine's standard, and the deceases accessed an unsupported stope face. It was also clearly evident that the backfill bags had not been pioperly placed, and left gaps between the roof and bag interfaces.

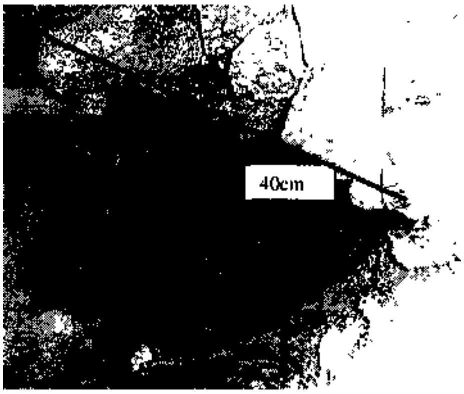
Figuie 2 Fall ot giound in a stope

Case 3. Rockburst - minute layout

3 in-slope members were fatally injured almost 3,000m below surface when the seismic event of magnitude ML=3 on the Richter scale occurred as per photo 3. The epicentre of the event was located about 20 m ahead of the panel. The stope panel was part of a longwall being mined on breast, and approaching a scismically active fault at the time of the accident. The original sloping width of the panel was 1,5 m prior to the accident, and approximately 1 m of dynamic closure approximately 4,5 m from the face was measured after the accident. The critical question to be answered in this case: what wast the cause ol the intense damage mechanism?

After having many inspections and studies, a slip type of event was determined, most probably high stress-strain build up on the major fault concerned.

In this case there was no in-situ bracket pillar had been left along the active dyke. The mining of the panel was not being mined and supported (no consideration for backfill & hydraulic props) as per rock mechanics engineer's recommendations.



Figuie 3. Complète closure in the slope

Case 4. Shaft pillar extraction

The extraction of a shaft pillar late in the life of a shaft demands the most stringent planning, since stress and safety conditions in the pilar are now at their worst. The exercise in fact be thought of as the removal of a large and probably seismically hazardous remnant, containing, important installations whose well being usually needs to be preserved.

A seismic event with a local magnitude of ML=4,1 on the Richter scale occurred in a shaft pillar extraction area approximately 2000m below surface as per Figure 4.

The tunnel was heavily supported by means of longanchors, rockstuds and mesh and lacing system. Damage to this service excavation was associated with massive footwall heave from the north side, and bulking of approximately 1,5m of the top corner of the south sidewall, and some bulging of the mesh and lace support was also evident. The area of damage associated with small dyke, and approximately 25m below the stope panel.

In this case shaft pillar was being mined out in seismically active 2 major faults, with potential to unclamp a structure while mining close-by. At the time of the incident much mining was taking place towards shaft, which increased the risk of damage to the shaft. Mining the shaft pillar area significantly increased the size of the excess shear stress-ESS areas, with associated increase in risk of slip on these structures.

Figure 4 Rockburst damage mechanism in a deep level tunnel

The steel guides in the shaft have buckled into shaft compartment as per Figure 5. Large slabs of the concrete shaft lining had been broken off and fallen into shaft. In the hoist chamber the chassis of large electric motor pinned into the foot was broken on all four corners.

Figure 5 Veitical closure of about 0.5m caused severe damage to the shaft steel work

8 CONCLUSIONS

Over the years, the South African had rock mining industry sustained, and continues to sustain, a high level of rock related accidents, and the resulting rate of human casualties (injuries and fatalities) has been, in world-wide terms, unacceptably high. Of these accidents-2000, 48% of all accidents have been rod,-related, that is, the result of rockfalls or rockbursts. In gold mines rockfall and rockburst accidents also represent the most important cause of all fatalities. In 2000. 61% of all fatalities took place in gold mines of which 82% of all accidents that occurred in gold mines were rock-related.

The author conducted and analysed many rock-related accident investigations in order to determine the actual causes. The investigations and the inquires revealed that most of the fatalities and seismic damage mechanism occurred in the deep level gold mines could have been minimized if the fallowings were in order;

1. Installation of the support as per mine's standards;
2. Lack of support installation and areal coverage (backfill etc.),
3. Non-adherence to designed mining layout & sequences-poor mining practice,
4. Lack of rock-related hazard identification & training,
5. Appreciation to seismic monitoring and prevention in mines.

The amount of seismicity and rock-related fatalities in mines can be reduced if the fallowings are taken in to account:

- By keeping or introducing *backfill support* in all deep level gold mines in terms of strata control, regional support, environmental control, ERR control etc.
- *Rock engineering* services can make great contributions to the rock-related safety of the mine. All rock mechanics personnel should be legally appointed so that they can speak and express themselves in management language in order to get their rightful recognition.
- Rock-related *risk management* should be integral management system of the mine to reduce accidents.
- Rock-related accidents can be reduced significantly by *training* the workers in strata control and support issues.
- In-stope face support system must be closed to the face as possible and the support should have a sufficient *areal coverage*.
- Adherence to *layout design* and extraction principals of the mine's rock mechanics department.
- The mining of remnants & pillars should be avoided.
- A proper methodology needs to be developed in order to mine out highly stress pillars safely.
- The concepts and tools for *seismic prediction* in South African mine has been developed, and the results are appreciated and valid in the more holistic approach towards the assessment and management of seismic risk.

The mining industry in South Africa especially deep level gold mines is still working hard to improve its safety and productivity records. The gold mines are to be in a position to enjoy continued long-term success, and therefore they will utilize the most advanced technology available to reduce rock-related fatalities and injuries.

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Acoustic Anemometry Control Means Elaboration for Coal Mines

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ABSTRACT: New class of airflow rates control devices has been developed in MSMU Electro-engineering dpt. Acoustic method of flow rates measurement is the most perspective and free of inherent to traditional methods defects. The main characteristics of the acoustical anemometer designed are: wide range (0.1 -20.0 m/s); high precision (measuring error 0.02+0.02V m/c); lag-free instrument (time of one measurement 0.01 s); possibility of interfacing with the computer; averaging during any time interval; reliability (absence of frail and moving parts). These features make acoustical anemometer an optimum for airflow velocities control, both in manual, and in the autonomous mode. A phase difference of two acoustical signals transmitted along and against the flow is being measured. Acoustic signals are being radiated and received by piezoelectric transducers mounted in the walls of anemometric channel. A phase difference is directly proportional to average velocity of airflow through the wave duct. Theoretical ard experimental researches have provided a basis of the method, construction parameters optimization, elimination of some inaccuracies (for example, caused by environment parameters changing).Anemometers operating in mines need the regular check, and checking installation has been designed in MSMU. Developed instruments already successfully work in number of mines. In the opinion of authors, acoustics method is the most perspective direction in mine anemometry.

1 ACTUALITY OF DEVELOPMENT

Mines atmosphere conditions have a great importance for mine safety and health. Ventilation is a main means of governing mines atmosphere composition. Ventilation provides both a dilution of toxic gas, and prevention of explosive concentrations of methane, evolved during process of coal mining. The main parameter of ventilation process is a quantity of air. The quantity is calculated as the product of working cross-sectional area and average velocity in the cross-section. Airstream velocity must be measured precisely, because insufficient ventilation (small velocities) may bring about unpermissible level of methane concentration. Opposite, excessively greater velocities may bring about significant condition of miners work worsening (airflow rise coal dust), besides, a greater amount of electric power is dispersed for it. Consequently, velocity of air must be not above and not below determined limits. Airflow velocity changes must be checked systematically by exact reliable instruments. Mine anemometer must have a velocity measurement range 0.1 -20.0 m/s with, reasonable accuracy not more than 10% all over range, high reliability in connection with particular conditions of usage.

At present time usually vane anemometers are used in mines. They have incurable defects of operation principle. On the one hand impeller must be as possible more light and must be inertia-free; on the other hand, it must be strong and firm.

Sometimes, for the measurement within the range of velocities below 1 m/s, thermo-anemometers are used, which have pinpoint accuracy, but also has essential defects. They contain frail heater, nonlinear characteristics and main, small dynamic range.

Therefore, the problem of free from proved above defects anemometer development is actual. And this anemometer must have another new method of airflow velocity measurement.

2 ACOUSTIC PHASE METHOD OF AIR VELOCITY MEASUREMENT DESCRIPTION

Acoustic anemometer was designed in the Moscow State Mining University (MSMU). An acoustic phase method for air flow rate velocities measurement is in the basis of it (Skundin et al., 1990).

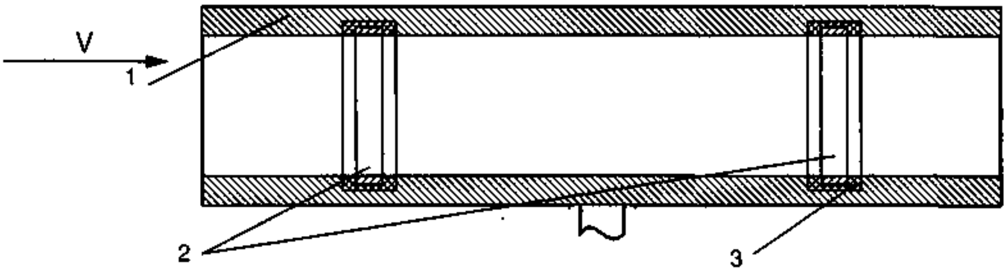

Figure 1 Acoustic anemometer sensor (anemometer channel). 1- Cylindrical waveguide-airduct. 2- Radiating and receiving transducers (piezoelectric rings). 3- Isolators

Acoustic flowmeters for liquid are well known. Attempts to build acoustic instrument for gas velocity measurement also have been made, however such instrument creation is difficult because of low gas density in comparison with liquid. The acoustic-waveguide prevents the dissipation of the acoustic energy, so the signal amplitude is big enough.

Sensor of acoustic anemometer (anemometer channel) is a cylindrical waveguide-airduct with the piezoelectric transducers mounted in the wall (Figure 1).

Acoustic signal propagates along waveguide axis. Aneinometer channel is put into the airflow along its direction. Each transducer is radiator and receiver of acoustical signal by turns. A phase difference between two acoustical signals transmitted along and against the flow is being measured. A phase difference is directly proportional to average airflow velocity in the cross-section of the airduct. Figure 2 illustrates the simplified model of the anemometer operation.

The source of acoustical waves radiates a plain wave, propagating along z-axis. Uniform airflow of velocity V has positive direction (Figure 2a).

The signal comes to receivers, located at the distance / from the source along and against the flow. Input notation;

$$p_+(z,t)=P_0\sin\left(\omega t-\frac{\omega}{c+V}z\right) \qquad (1)$$

$$p_-(z,t)=P_0\sin\left(\omega t\,(2)\frac{\omega}{c-V}z\right) \qquad (2)$$

These are expressions for acoustic pressure waves, propagating in positive and negative direction respectively.

Receiver's acoustic pressure dependency on a time is shown on the figure 2b.

c - sound speed in the air; *a)* - an angular acoustic radiation frequency; $(c+V)$ phase velocity of waves, propagating in positive direction of z-axis; $(c-V)$ phase velocity of waves, propagating in negative direction of z-axis.

Therefore phase difference of these signals is;

$$\Delta\varphi=\omega l\left(\frac{1}{c-V}-\frac{1}{c+V}\right)=\omega l\frac{2V}{c^2-V^2}\approx\omega l\frac{2V}{c^2} \qquad (3)$$

We can see that the phase difference is directly proportional to How velocity, when flow velocity is SITU,I in comparison with sound speed *{V«c).* Sound speed in the air at normal conditions is about 345 m/s, and airflow velocity usually is not higher then 20 m/s, therefore this condition is satisfied.

However the factors such as gas temperature, gas content depend on proportion factor in Formula 3. American researchers of similar design (David et al., 1980) noted absence of repeatability in experiments and special basic research for this phenomena needs explanation.

We also have got to the conclusion that it is necessary to take into account wave propagation particularity for designing such instrument. Basic research results are in publications (Skundin et al., 1998; Skundin et al., 2001)

Characteristics of acoustic wave propagating in the acoustic anemometer channel, i.e. in cylindrical waveguide is more complicated. Acoustic pressure on receiving transducer is a number of harmonic elements - normal modes; each of them has its own amplitude and propagation velocity:

$$p_+=\sum_n A_n\sin\left(\omega t-\frac{\omega}{V_n}z\right) \qquad (4)$$

where A_n - an amplitude of n -th mode, V_n - a phase velocity of n -th mode.

Fundamental investigation has been made to create air-acoustic interaction in the anemometer channel mathematical model. It has been shown that phase difference of signals, which are the sum of modes, is directly proportional to tlow velocity also just as for the plain wave propagation in opened space. So phase characteristic of acoustic anemometer is linear.

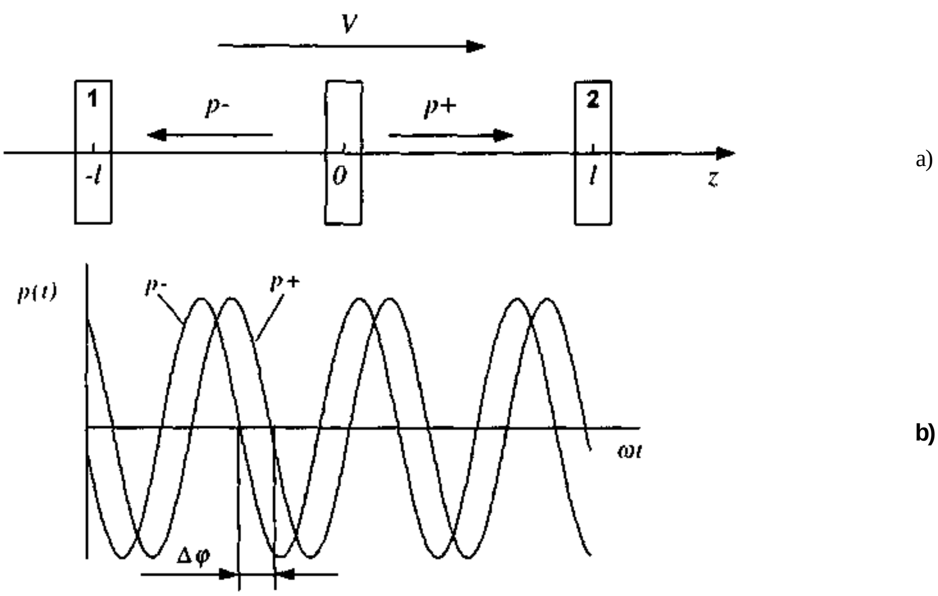

Figure 2. The schematic form of acoustical method of airflow speed measurement

3 CHARACTERISTICS OF ACOUSTIC ANEMOMETER

The acoustic anemometer designed in MSMU has the following characteristics:

- it can define the direction of flow being measured;
- it is not critical to the gas contents of the controlled tlow;
- an aerodynamic resistance - not more than I0 Pa:
- measurement dynamic range - to 1.500;
- It is practically inertia-free (not more than 3 ms);
- airflow velocity measurement range 0.1 - 30 m/s;
- inaccuracy (0.02+0.02V) m/s;
- working temperature 0 - 40 °C;
- combinable with PC:
- supply voltage- 5 V:
- consumed power - 400 mWatt;
- work resource 10,000 hours;
- sizes of anemometer channel - a length of 120 mms, external diameter - 38 mms (internal - 28 mms):
- dimensions of electronic block -200x50x45 mm:
- weight - 400 g.

Structural anemometer scheme is shown in Figure 3. Generator works out pulses with the frequency of 80 MHz. The lime between receipts of same phases of signals propagating along and against the tlow is filled by these pulses. Counter controlled by a controlling system counts these pulses. Therefore, phase difference is proportional to a number of filling pulses.

Controlling system forms a packet of pulses by the frequency 30 kHz and gives it through the commutator to the radiating ring. This system also controls a switching of rings connecting them by turns as a radiator or as a receiver. Signal from the commutator through the amplifier is transmitted to analog entrance of the controlling system, where a comparator is built-in. Data is coining to the computer port through the interface device.

4 INDEPENDENCE ON ENVIRONMENT

Phase difference of acoustic signals, propagating along and against the flow can be calculated according the Formula 3. We can see that the phase difference depends on sound speed. Sound speed is a function of air parameters, such as temperature, pressure, gas content. Excluding the dependence on the out come inductions upon the sound velocity we eliminate the influence of the named above parameters upon the velocity measurement. This algorithm is designed and realized in the anemometer functioning at one of the normal modes (Skundin et al., 2000).

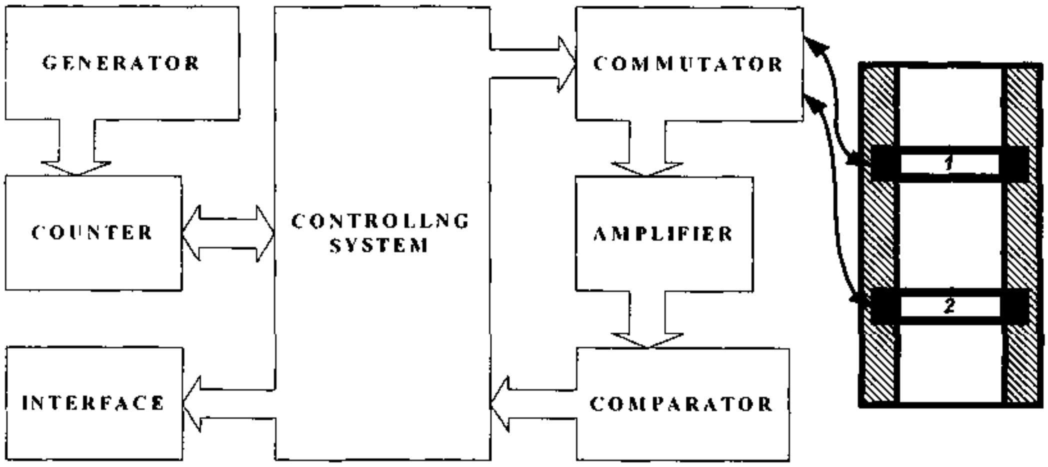

Figure 3. The structure of the acoustical anemometer

5 ACCURACY OF PHASE ACOUSTIC METHOD

Minimum inaccuracy of acoustic anemometer is defined by the time interval between nearby pulses i.e. frequency of filling pulses /„. Minimum inaccuracy is calculated by the formula.

$$\delta V = \frac{c^2}{2 l f_0} \quad (5)$$

We can see that inaccuracy is inversely to length of base (distance between radiating and receiving transducers). However, sizes of portable instrument must be small enough. Besides the longer the tube, the bigger it's aerodynamic resistance. But stationaiy installation based on described above principle of operation can have pinpoint accuracy.

In MSMU the aerometrie installation for the checking of anemometers was designed. It is a wind tunnel with the electroacoustic transducers system which is similar to the one described above. Airstream is created in the pipe. Stability and linearity of characteristics, pinpoint accuracy gives possibility of using an acoustic sensor as the referring one.

6 CONCLUSION

The authors consider that acoustic methods are the most perspective direction for mine anemometry development. The portable acoustic anemometer and aerometrie installation for the checking of anemometers were designed. In the present time our laboratory is developing a stationary air velocity sensor, which could become part of mine safety monitoring system.

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Reduction of Toxic Components Released by Motor Vehicles in Quarries

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ABSTRACT: When the open mines go down deeper into the ground the velocity of air streams becomes lower and the air exchange around the work places gets worse thus polluting the atmosphere with hazardous gases and dust. And furthermore, the ore quarries use dump trucks which discharge large quantities of toxic components into the atmosphere. To reduce the toxicity of the exhaust gases of the dump trucks there have been developed and introduced into production gamma catalytic neutralize«, complying with the type *of* the vehicle, its engine power and operating conditions.The dump trucks BcIAZ operating in MEDET Open Ore Mine have different lifting capacity and use the catalytic neutralizer H-2. It is charged by means of a catalyst based on cobalt and copper oxides laid in thin layers on small porous aluminum oxide spheres. During the road tests it has been established that the reduction of the toxic components in the exhaust gases is close to that of catalysts made one the basis of platinum. The catalytic neutralizer H-2 has excellent noise silencing characteristics as well as low aerodynamic resistance and the lack of precious metals make it inexpensive and readily available.

1 INTRODUCTION

Open pit mines use powerful high production technical facilities permitting the increase of their maximum allowable depth. With the increase of mining depth the velocity of the air streams decreases resulting in air exchange deterioration and pollution of the mine atmosphere with hazardous components. The content of hazardous components in open mines atmosphere produced by diesel internal combustion engines can be reduced to a certain extent by adjustment of the engine combustion system and mode of operation. A major measure to fight the harmful substances contained in the exhaust gases of the diesel engines is the employment of various neutralize« - liquid and platinum based. The liquid neutralizes are not sensitive to the carbon oxide and the necessity of periodical carbon black cleaning from the inside, their unsuitability for low environment temperatures as well as their considerable dimensions makes them rarely applied.

The studies made so far have shown that for the time being the most efficient way for exhaust gases toxicity reduction is their catalytic name-free burning up. The results of the tests of catalytic platinum-based neutralizers have shown both their lower sensitivity to nitrogen oxides prevailing in the exhaust gases from diesel engines and a higher price and that made it necessary to develop and implement a new range of neutralizers with a catalyst based on cobalt and copper oxides.

When they contact the catalyst the toxic components of the exhaust gases released by diesel engines burn up forming carbon dioxide and water, thus limiting the hazards in the mine atmosphere.

2 DESIGN FEATURES OF THE NEUTRALIZER

An example of a catalytic neutralizer of exhaust gases subject of the invention is shown in Figure 1. The catalytic neutralizer comprises a housing I including a heat insulating material 2, placed between cylinders 3 and 4; a reactor 5, the inside space 15 of which is formed by perforated outside and inside cylinders 6 and 7; streamline separator 8 and a back cover 9. The inside space of the reactor 15 is filled with catalyst granules 10. The reactor is filled with catalyst granules through the plugged opening 11 on the back cover 9. The catalytic neutralizer has also a pipe 12 with tapered outlet for feeding the exhaust gases and an inlet enclosure 16 for directing the exhaust gases to the reactor. The perforated inside cylinder 7 has a non-perforated end which serves as a pipe discharging the exhaust gases to the atmosphere. On the inside surface of cylinder 4 at a distance of 1/3 of its length there are two circle rings 13 directing the movement of the exhaust gases to the reactor 5. The front cover 17 has a plugged opening 14. The streamline separator 8 is loosely accommodated in the cylinder 4 and the linear expansion of the individual elements of the reactor 5 due to heat load do not affect the catalytic neutralizer.

Based on the patented design there has been developed and implemented the production of H-2 catalytic neutralizer for dump trucks BCJIA3 of 27 tons capacity and KOM-1 and KOM-2 catalytic neutralize« for fork-lift trucks.

H-2 neutralizer is installed horizontally, KOM-I and KOM-2 are installed vertically. The reactor is heat insulated in order to maintain a higher temperature m the neutralizer for ensuring a more intensive oxidation process and avoiding overheating of the truck units. The heat insulation consists of swollen pearlite sand with heat conduction $X = 0.46$-0.7 W/ m "K.

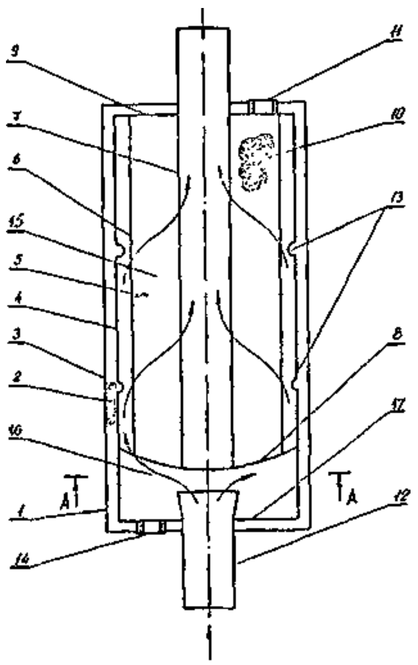


Figure 1. Vertical section of H-2 catalytic neutralizer

3 PRINCIPLE OF ACTION

The catalytic neutralizer acts in the following way: the engine exhaust gases are fed by pipe 12 and are evenly distributed in the streamline separator 8 and guided by the circle rings enter the reactor through the perforated outside cylinder 6. When the exhaust gases contact the catalyst grains the hydrocarbons, carbon oxide and carbon black oxidize, burn up and turn into non-toxic products discharged to the atmosphere through pipe 7. The tapered outlet of pipe 12 and the streamline separator 8 ensure regular and more effective passage of the gases through the reactor.

In case of inclined or vertical installation of the neutralizer the catalyst grains that have incidentally fallen off collect in the enclosure 16 and are removed through plugged opening 14.

The design developed can be used in different internal combustion engines having characteristics corresponding to the neutralizer activity.

3.1 Composition and properties of the catalyst

The proposed design of the catalytic neutralizer complies with the application of a catalyst developed by the Institute of General and Inorganic Chemistry at the Bulgarian Academy of Sciences and patented under No 21437. It has been developed on the basis of cobalt and copper oxides applied on a thin layer of porous carrier having a heat resistant surface. The starting temperature of the catalytic action is 200"C (cold start). The maximum efficiency of purification is achieved at a temperature of 30()"C. The catalyst is resistant to catalytic toxins such as sulphur dioxide (SO_2).

4 TEST METHODS

H-2 catalytic neutralizer was subjected to road performance test in Medet Open Ore Mine. The aim of the tests was to determine the reliability of the neutralizer and its efficiency under operating conditions.

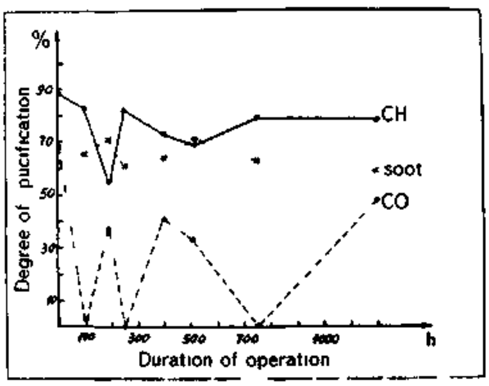
For the test purposes the exhaust systems of two dump trucks 6ejiA3-540 were each equipped with two H-2 neutralizes on the left and right cylinder group, respectively. The measurements included taking of gas samples with the truck moving in first gear, at maximum engine load of 1700 rpm, with a load of about 30 tons in the coach, at road slope of 10%, upstream and downstream the catalytic neutralizer.

The inlet and outlet temperature of the exhaust gases as well as soot content were measured. The gas samples were taken to measure the content of carbon oxide (CO), hydrocarbons and nitrogen oxides. The temperature was measured with the aid of thermocouple of copper-constantan and millivoltmeter. The soot content was measured with the Polish sootmeter D-400. Ten measurements were carried out and the soot content was determined as an average arithmetical value taking into consideration only those measurements with deviations up to +/- 10%. The CO content was measured by a gas analyzer Meihak , Germany while the hydrocarbons concentration was determined with Chrom-4 gas Chromatograph. The content of nitrogen oxides content was measured with "Toxiwarn", Deegger, Germany

5 TEST RESULTS

Figuie 3 shows the dependence ot the purification elfıcıency ot the catalytic neutraliser on the duıatıon of its activity As it can be seen throughout the complete test ıun H-2 neutralize) shows high activity
The degree hydiocarbons îcmoval is about 80% In lespect ot CO it shows high activity as well, but the content ot this component in the exhaust gases at noimal diesel engine control is relatively low, Theiefore when the inlet CO content gets lower than 0 2% the neutiahzer activity becomes zero

Since the chaıacteıistıcs ot the catalytic neutralizes offered in the international market do not indicate the reduction of nitrogen oxides Figuie 3 docs not show such data about H-2 neutrahzer But, however the tests earned out demonstiated that as a result ol the leduction aiea formed in the H-2 ıeactor the content ot nitiogen oxides ıs paıtıally reduced too by about 20-25%



Figuie 2 Dependence of the punlication efficiency ot the catalytic

Table I gives comparative data in teims ol punficatıon el fluency ot different types of catalytic neutralizes otteied on the woild mai ket and the Bulganan H-2 neutiahzer

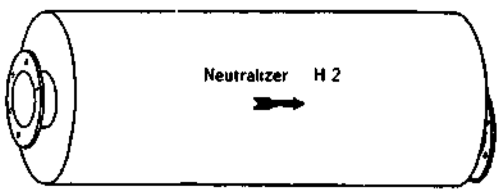
All neutralizers offered aie based on pıecıous metals such as platinum or palladium the catalyst ot which is passive in lespect ot nitiogen oxides

As can be seen in Table 1 H-2 neutralizer has almost the same activity as the otheis Only H-2 and Engelhard are the most efficient in ıespect ot hydıocaibons At the expense ol the lelativeiy low activity in ıespect ot CO it has the highest degree of soot removal
The concentration ot CO in the diesel engines is small i e tıom 0 01 to 0 5 vol % and consequently the degıee ot puıificatıon is quite good taking into consideration the small share of CO in the total

Table I Efficiency of Diffeient Catalytic Neutralizers

Component	Degi ee of ı emoval vol <7c			
	Bulgana H 2	USA Engel haul	Gei many Helens 20	Russia H Kfl 241
Hydiocaibons	XO	80	6S	70
C il bon oxide	45	X5	90	75
Soot	60	20	45	
Nitiogen oxides	20			

exhaust gases toxicity In the exhaust gases the hydrocaibons soot and nitiogen oxides are considered to be the most toxic They aie pıesent in the highest concentrations, too Thcretoie, the efficiency of a ceıtaın catalytic neutıalızeı tor diesel engines is assessed by the extent ot removal ot those components



Figuie 3 Geneial view ot H-2 Neutralize!

Technical characteıistıcs ot H-2 catalytic neutralızeı

1 Dimensions mm
   Length 822
   Housing diameter
   300
2 Weight ot charged neutralizer kg 35
3 Weight ot catalyst, kg 11
4 Weight ol heat insulation mateıial kg I
5 Aerodynamic resistance at noimal engine operation mode mmHg 25
6 Durability, km
   - after initial charging 20 000
   - after second charging 40 000
7 Degree ot exhaust gases purification, vol %
   - Hydiocarbons 80
   - Cai bon oxide 45
   - Soot 60
   - Nitrogen oxides 20
8 Engine powei kW 175

6 CONCLUSION

The results of the H-2 tests show that it complies with the current requirements for diesel engine exhaust gases purification.

It is easy to install, does not affect the operation of dump trucks, its activity is secure and requires no special maintenance.

The fact that this catalyst is not based on precious metals oxides makes it inexpensive and readily available. The rechargeability of H-2 neutralizer is an additional asset to its economic protltability and implementation.

Moreover, H-2, KOM-1 and KOM-2 have excellent noise supression capabilities and thus the need of installing noise damping pots on the trucks is non-existent.

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Designing The Ventilation System For Galandroud Coal Mine

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ABSTRACT: Galandroud colliery, which was recognized as a highly gasous coal mine in north of Iran, is one of the important mines in terms of its coal properties. In view of the fact that methane was the major existing gas in the mine and its natural flow is upwards, the ascending ventilation was believed to be the most efficient. In addition to this, due to presence of a number *of* adits in upper levels of mine, the *forcing* ventilation system acting from the lower level of mine was chosen. Firstly, the total required fresh air quantity was empirically calculated to meet the ventilation requirements. Secondly, the total pressure lost was found using analytical methods. Furthermore, computer model network simulations were conducted so as to evaluate the effects of different network configurations and ventilation adjustments. Finally, an axial blower type of fan which meets the above needs as well as the future ventilation requirements was also selected according to an economic challenge. All results obtained from different methods were compatible.

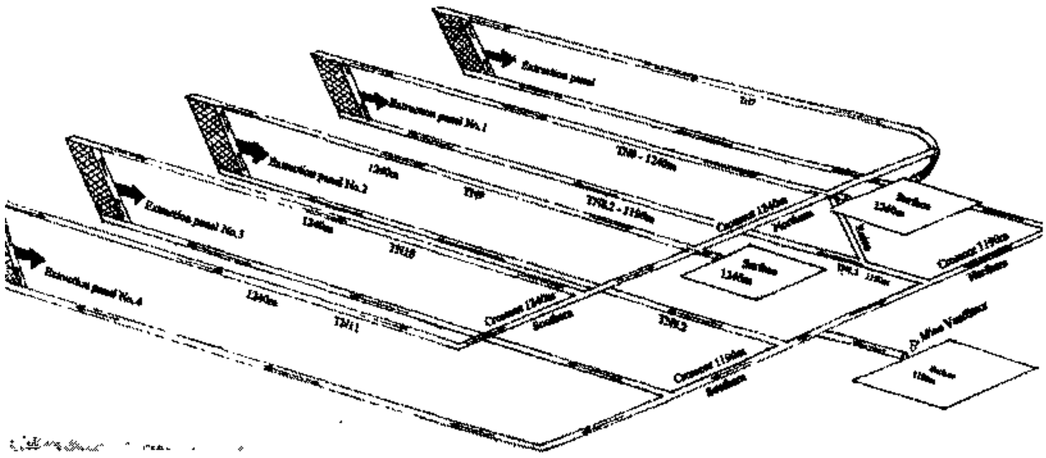
I INTRODUCTION

The Galandroud coal mine, affiliated with National Iranian Steel Company (NISCO), has been opened and developed quite recently. Having been completely explored, the mine was divided into 2 main levels namely +l 190 and +1240. Due to the topography of the area, horizontal adits having E-W orientation were considered in order to access the deposit. Crosscuts with orientation of N-W were derived in order to make the extraction panels. Owing to the thickness and dip of coal layers, five conventionally diagonal retreat longwall faces were regarded as the main method of mining. The total annual production from the mine was approximately 150,000 tons. Since the methane emission rate per ion of coal mined was IOnr/t, Galandroud coal mine is categorized as a highly gaseous coal mine (GIS GeoincTustria, 1998).

Design methods of mine ventilation system can be categorized into three groups: (1) analytical methods, 12) empirical methods and (3) observational methods (Osgoui, 2000).

Analytical methods deal with fundamental concepts of fluid mechanics in conjunction with applied mathematics. In cases where the ventilation network is simple (initially planed ventilation network), analyses are carried out easily; on the other hand, in the case of complex networks (investigating or upgrading the in-use ventilation network), handy methods would be inapplicable: accordingly, computer-aided and numerical methods are taken into consideration to be more appropriate. Empirical methods include the governmental, health and safety regulations as well as experimental rules. Nowadays, empirical methods are best known as a prerequisite for analytical methods. Observational methods rely on consistently monitoring the ventilation system in the course of time. Incessantly measuring and inspecting the amount *of* harmful or explosive gases, air velocity and quantity and air pressure are the chief essence of this method. Therefore, a primary ventilation system can not be developed unless observational methods are taken into account. Using the above mentioned methods appropriately, a perfect ventilation system for both initial and further stages of mining will be achieved.

In the Galandroud coal mine, planning the ventilation system was carried out based on analytical and empirical methods; however, computer model network simulations were conducted in order to evaluate the effects of different network configurations and ventilation adjustments. A overview of the ventilation network layout is presented in Figure l.



Figuie I Mine ventilation network layout

2 REQUIRED QUANTITY OF AIR

The amount of air needed should be determined in such a way that it will satisfy different requirements such as breathing, dilution of harmful gases and dust produced in the mine (Hartman et al. 1997 & Madani, 1987).In the Galandroud coal mine, air quantity for working places was estimated based upon the rate of gas emission and the number of workers. In view of the differences between the rate of gas emission per coal mined at the beginning and end of every work shift stem from workers' fatigue, the rate of product at the beginning of the shift is therefore counted twice as much as that at the end of the shift. According to a survey conducted at longwall faces, the rate of gas emission at the beginning of the work shift was found to be 0.0572m"/s. Consequently, the amount of fresh air for any longwall face to keep the gas concentration below 0.75% is calculated as follows! Osgoui. 2000):

$$Q = \frac{100}{0.75} p\lambda$$

Where Q is the required air quantity (m/s), p coal production of any stope (l/h) and x is gas volume acquired from one ton mined coal *(mit)* and therefore:

$$Q = \frac{100}{0.75} \times 0.0572 = 7.63 m^3/S = 457.77 m^3/\min$$

Considering 30% air leakage from different parts of mine, required fresh air necessary for each stope is computed:

$$Q = \frac{7.63}{1-0.3} = 10.9 m^3/S$$

For unpredicted problems, which cause the inadequacy of the ventilation requirements as well as in order to meet the federal regulations, estimated air quantity has to contain a safety factor of 20%. Accordingly, given 5 longwall faces, the total air quantity for all stopes (or the total air quantity for mine) will be 65nrVs. The manner of total air distribution is demonstrated in Figure2.

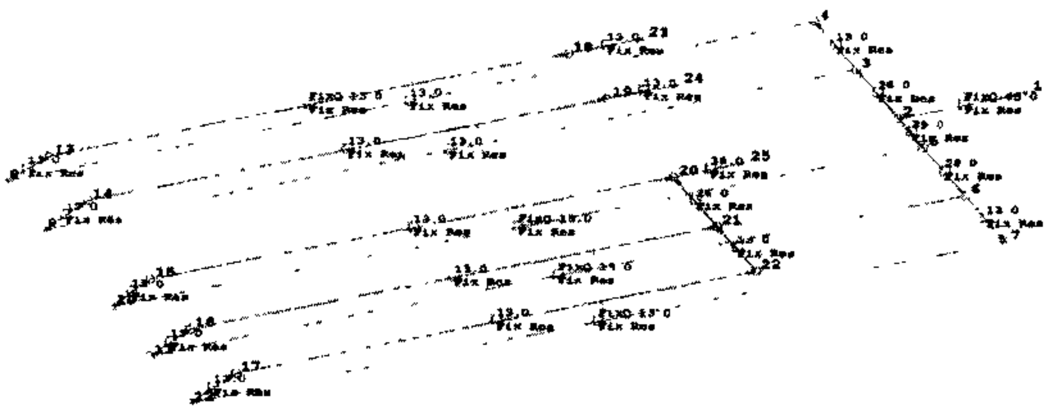
3 PRESSURE LOSS ESTIMATION

In order to calculate the pressure loss in each branch of mine airways it is necessary to compute the friction factor and the resistance. Based on physical characteristics of airways as well as empirical equations, the friction factor values utilized were taken into account as following:

For main adits and major tunnels (A= **11.2 m^2)** K=().01484kg/m''

For minor tunnels and drifts (A= 7.2 in") fc^O.OIHSSKg/m'

For working places (A= 4.66 m^2) K=0.04638 Kg/m'



Figuie 2 An quantity distribution (m'/s)

It was interesting at this point that while back analyzing the data obtained from monitoring the ventilation system; the K values acquired by both empirical expressions and inverse equations were in reasonable agreement.

Having been calculated the friction factor, resistance of airways were to be computed by Atkinson equation(Hartman et al. 1997 & Ramani 1992a, b):

$$R = K \frac{LP}{A^3}$$

Where R is resistance of airway (NS^3/™*), K is the friction factor of airway (Kg/m), L is length of airway (m), P is airway perimeter (m), and A is cross-sectional area of airway (nr).

Note that pressure loss in each branch of the ventilation network is equal to the summation of corresponding factional pressure loss and shock head loss: accordingly, in order to determine the total pressure lost, the value of shock loss has to be taken into consideration. In the Galandroud coal mine, the effect of shock loss is surveyed by means of equivalent-length method. This method, indicated by below equation, has been strongly suggested by Hartman et al.(1997).

$$R = K \frac{(L + L_e)}{A^3} P$$

Where L_L is equivalent length of airway which is being obtained from literatures. Therefore, statistical pressure loss of each branch of ventilation networks is determined by:

$$\Delta P_s = RQ^2$$

Where Q is the air quantity passing through airway (m /s). R is airway resistance, and AP, is statistic-pressure loss (Pa or mm-water).

3.1 Adjusting the ventilation network

When airways are arranged in parallel and a prescribed quantity of air is made to flow through each branch, controlled splitting is being utilized. Controlled splitting is used to obtain the desired quantity of air through workings and airways forming splits, rather than allowing natural splitting to prevail and the air to apportion itself. In order to solve mine-ventilation networks employing controlled splitting, the number of required meshes for applying Kirchhoff s Second Law must be calculated as follows:

$$N_m = N_b - N_j + 1$$

Where N_m is the minimum number ot meshes, N_b is the number of branches, and N, is number of junction. Therefore, the minimum number of meshes were determined as follows:

$N_{m} = 12 - 9 + 1 = 4$ independent loops

Having applied the first and second Kirchhoffs law, desired pressure losses provided by modifiers (negative regulators) in defined branches were computed. Adjusted ventilation network is illustrated in Figure3 and the total mine statistic pressure loss was determined to be 320 Pa (32.58 mm.water).

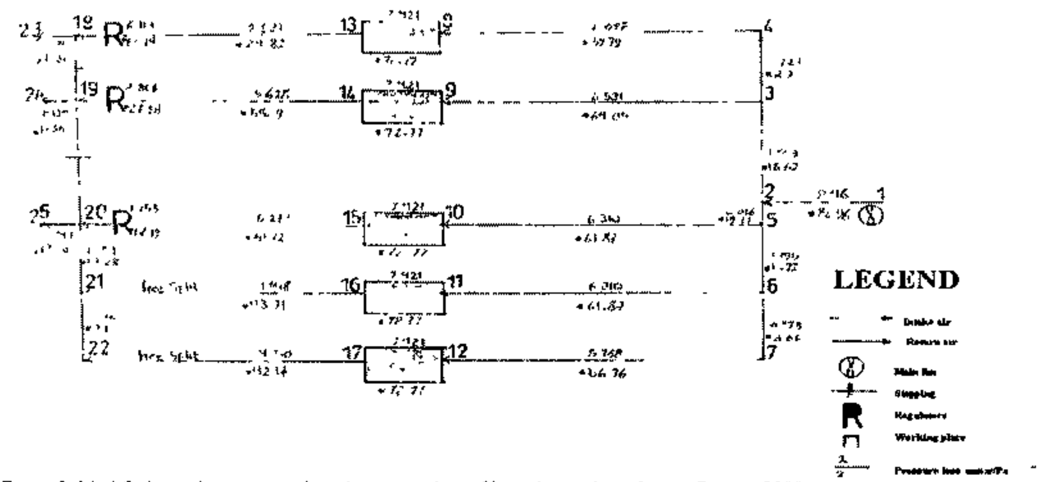


Figure 3 Modified ventilation network with pressure loss of branches and regulators (Osgoui, 2000).

3.2 Estimation of dynamic and total pressure loss of inme

In order to determine the total pressure loss, it is essential to calculate the dynamic pressure loss in all exit airways. Because of forcing system of air distribution in such a way that total input air enters the system by tunnel 6 and exits the system through tunnels 7.8,and 9, it is therelore imperative to determine the dynamic pressure loss in each exit tunnel. The maximum dynamic pressure loss obtained is regarded as the dynamic pressure loss of the mine. In view of the fact that maximum intake air quantity passes through exit tunnel 9, it is therefore expected that maximum dynamic head loss will occur in that tunnel. Hence, dynamic pressure loss will be:

$$\Delta P_{1} = \gamma \frac{V^{2}}{2g}$$

Where γ is unit weight of air, Vis air velocity ,and g is gravitational acceleration. Hence, for exit tunnel 9:

$$\Delta P_{1} = \gamma \frac{V^{2}}{2g} = 1.2 \times \frac{3.48^{2}}{2 \times 9.806} = 0.7409 mm.water$$

As a result, the total head loss of mine will be obtained from summation of both dynamic and static pressure loss, hence:

$$\Delta P_{t} = 32.58 + .7409 = 33.4 \, nim.water \, (328 pa)$$

4 SELECTION OF THE MAIN FAN

In general, the main fan to be selected must be capable of delivering 65mVs at a minimum pressure of 328Pa. In order to meet the ventilation requirements, two types of main fans were therefore chosen. According to their adeptness and capital cost, it was advisable to purchase a Iranian-made fan instead of importing one. Consequently, an axial blower fan with blade setting ability was exclusively preferred to be more efficient. As a result of relatively lower pressure loss at the first stage of the ventilation system, at the blade set to 30" and little efficiency of fan, ventilation needs of system is readily satisfied. Technical characteristics of the chosen main fan are demonstrated in the Table I.

C- analyzing the ventilation network in the case of employing fan made by USA.

Table I - Technical charactei istics of axial main fan (2CQ-200)

F.m du	Vdcx ity	Power	Efficiency		Provided pressure			Provided air quantity	
			Max	Mm	Max	Mm	Ave	Max	M in
2000	900	5-250	80	62	2 1	0 1'J	70	120	35
mm	ipm	kW	</<		kPa		mVs		

Over the life of the mine, the demand for head and quantity will vary; therefore, the duty on the fan may dramatically increase due to new operating sections or decrease due to the addition of new shafts and tans. The selected fan will be able to meet all future ventilation requirements. Based on the operating point of mine, the required power of the fan can be calculated as follows(Hartman et al. 1997 & Ramanı 1992a, b):

$$N = \frac{Q_t \Delta P_t}{\eta_t \eta_2 1000}$$

Where Q, is total air quantity (m /s). AP, is total head loss (Pa), ŋi fan efficiency, n, electromotor efficiency, and N required fan power. Hence fan power will be:

$$N = \frac{65 \times 328}{0.66 \times 0.70 \times 1000} = 43kW$$

In view of the required fan, it seemed another alternative, Jeffrey fan 8HU-72, could have been a suitable choice, satisfying the initial ventilation requirements. The fan characteristics and operating point of mine when using Jeffrey fan 8HU-72 illustrated in the Figure4.

5 VENTILATION NETWORK SIMULATIONS

Generally, computer model network simulations are realistically essential in order to appraise the effect of different network configurations and ventilation adjustments (McPherson, 1993 & Bandopadhyay, 1992). Ventilation network simulation were conducted by three simulator softwares namely; MINVENT, VNETPC, and VENTSIM. Ventilation network was then surveyed based on the following cases:

A- analyzing the ventilation network without use of a fan.

B- analyzing the ventilation network in the case of utilizing fan made by Iran.

In all mentioned categories, the primary layout of the ventilation network was chosen in the way that it would be in agreement with other mining operations in addition to ventilation adjustments. Based on results obtained Irom simulating the ventilation network by three simulators, it was concluded that they are in reasonable agreement (Osgoui, 2000).

In the case where the software has to select the main fan, all three simulators suggest delivering 65mVs air quantity at a pressure of approximately 300 Pa through tunnel 6. Moreover, for delivering the fixed air quantity through working places, both VNETPC and VENTSIM advise that booster fans must be installed in tunnel 9, 10, and 11 and a regulator has to be put in tunnel 7. According to MINVENT's results, however, in tunnel 8, 9. 10, and 11 the booster fans must be established. It can be easily concluded that while VENTSIM and VNETPC used both negative and positive adjustment (combination regulation), MINVENT only utilizes positive regulation.

In the case where the Iran fan is used, all three simulators have identical results. Installation of main fan at tunnel 6 with air quantity of 65mVs at the range of pressure between 260Pa and 275 Pa are elucidated by them. However, to satisfy the operating point of the mine, it is necessary to use the booster fans in tunnel 8, 9, 10, and 11 and utilize regulator in tunnel7.

On the other hand, as far as the jeffry's fan is employed, all three simulators have also indistinguishable conclusions. The significant difference between the results of VENTSIM and MINVENT is that in

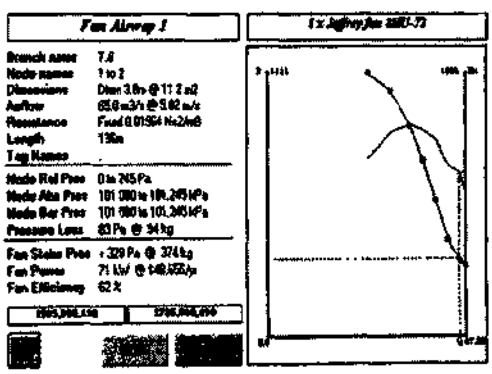


Figure 4 Operating poiiu of mine by simulating the ventilation network in the ease of using Jeffrey fan' fan static piessure =328MPa. an quantity=65m /s. fan efficiency =62% (Osgoui. 2000)

the latter one, the main fan load is reduced and this deficiency will be compensated by booster fans.

5 CONCLUSIONS

The design of ventilation system in Galandroud coal mine was conducted base upon main network of mine in the way that no additional airway was driven. However, to open and develop the mine, the ventilation effects were notably taken into account. Moreover, in regard to mine network configuration and adjustment, multi split system was recommended to be more appropriate. The main advantage of this system is that a new potential panel can be easily either linked or separated from the other ventilation circuits. All results obtained from ventilation network simulation are in remarkable compatible with those acquired from handy calculation. With the help of ventilation simulation, as the mine is extended, so the ventilation parameters such as required air quantity and pressure loss of each and then whole airways can be determined resulting in new operating point of mine would be clarified.

ACKNOWLEDGEMENTS

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Health, Safety, Environmental & Community Management: A Case Study For Rio-Tinto IMEx Turkey

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ABSTRACT: Health, Safety, Environmental and Community (HSE&C) issues are important parts of all stages of mining including exploration. To achieve a successful HSE&C management a good Environmental Management Plan and well organized training programs, audits and reporting is necessary. In this article a new HSE&C management program prepared and successfully applied for Rio-Tinto Industrial Mineral Exploration Group's drill sites in Turkey was explained.

1 INTRODUCTION

Health, Safety, Environmental & Community (HSE&C) management is one of the most important issues related to mining. Each stage of mining differs from another in this concept. Exploration being the one having least environmental impact but still need to be considered. There are strict regulations on mine safety and environmental pollution caused by mining industry. In United States these regulations are somehow changing from state to state. For example California is the one having most strict environmental regulations. Canada and Australia are very sensitive on environmental issues related to mining. Besides developed countries some developing countries also have environmental regulations related to mining.

Before to start any mining project it is important to prepare an environmental management plan in order to identify problems long before they become difficult and costly to repair. In mining this is done by the mining company itself or by an independent consultant. In this article the health, safety, environmental & community management carried by an independent consultant for a Drilling Company, working for Rio-Tinto Industrial Mineral Exploration Group in Turkey, which will be called IMEx hereafter, is explained.

2 LEGISLATION

In Turkey all environmental pollution related activities are regulated by the Turkish Ministry of Environment. There are several regulations related to mining such as:

- Water Pollution Control Regulation
- Air Pollution Control Regulation
- Solid Waste Control Regulation
- Hazardous Waste Control Regulation
- Environmental Impact Assessment

Ministry of Health and Ministry of Labor and Social Security has also some regulations related to mine health and safety.

All of above regulations consist specific chapters related to mining activities to be considered while preparing the environmental management plan.

3 ENVIRONMENTAL MANAGEMENT PLAN (EMP)

EMP was covering drill sites, office and core-shed. In this article only the part related to drill sites was described. EMP starts with the "Environmental Policy" stating the company's willingness to prevent environment and to be sensitive to natural resources. "Planning" stage starts with the description of all environmental aspects related to drilling. These aspects can be listed as follows:

- Waste water
- Solid waste management
- Use of raw materials and natural resources
- Contamination of soil and groundwater
- Local flora & fauna
- Natural disaster
- Construction of road
- Spills
- Cement platform
- Transportation
- Healing

- Electricity
- etc.

Then objectives and targets are described related to each environmental aspect. This part includes predictions and explanations on possible pollution, and contamination problems due above items. In "Environmental Management" part preventive measures could be taken against all of the above aspects were explained. For example, according to this plan used oil was being collected in drill sites and after reaching a considerable amount given to an oil company or used lubricating oil treatment facility. In an EMP policy, targets and objectives and actions should be consistent. An important part of EMP is the emergency action plan which is described below.

3.1 Emergency Action Plan (EAP)

A general emergency action plan, which is a requirement of EMP, describing all possible emergency cases in a drill site was prepared and located into a place where all employees can see it easily. Then separate emergency plans were also prepared for following case:

- Fire
- Earthquake
- Flood
- Lightening
- Oil or chemical spill
- Any injuries requiring first aid

All of above emergency plans were prepared both in English and Turkish, covered with plastic and located properly within drill tent. It was supplied that all employees should be aware of the content of each plan by reading them during the meetings. It was also strongly emphasized that the contents of these plans should be kept in mind.

4 HEALTH, SAFETY, ENVIRONMENTAL & COMMUNITY MANAGEMENT PLAN

The safety rules applied to all drill sites may be divided into two groups: pre-drilling and during drilling.

4.1 Pie-Drilling

In pre-drilling stage, the safety and environmental issues are taken into consideration and the locations of sites were selected accordingly if possible. Before to start, the pictures of selected site were taken to compare with those after drilling. Then a drill site plan was prepared and the site was established according to this plan, which is located at the entrance of the site, after establishment, as can be seen easily. In this plan, all drill equipments, drill tents, drill rig, safety equipments, fuel and oil containers, trash, lights were all shown. In our drill sites we have two tents one is the drill tent covering drill rig and main drill equipments and another one for the storage of core boxes and possible use for engineers. Fuel and oil were stored in metal containers onto metal pans to prevent any seepage and all were located at least 50 m away from drill rig. Main drill site and these fuels were fenced around. At the main entrance warning signs were placed. Drill site and drill tent have emergency exits also. Ground of drill site was covered with metal platforms to prevent any slides of employees. All pipes were located properly having stoppers at both sites. Plastic plugs should be put for both ends of drill pipes. Sites should be lightened enough to work at nights safely.

4.2 During Drilling

According to safety standards all employees should obey following rules during drilling:

- Use of personnel protective cloths (hard had, steel toe boots, safety goggles, gloves, dust mask-if necessary, etc.)
- While working at heights (where the potential to fall is greater than 4 m -13 feel) safety belt with full body hardness and shock absorbing lanyard should be used,
- Electrical safety was obtained with written isolation procedures for plant and equipment.
- It was strictly forbidden to enter confined space which is an enclosed or partially enclosed space such as storage tanks, hoppers, boilers, flues, ventilation and exhaust ducts, sumps, manholes, pipelines, trenches, excavations etc.
- Smoking is not allowed in 3 m to the rig.
- Tools should always be placed below shoulder level after use.
- Any trash should not be present in drill site.

4.3 Safety Equipments

In drill sites following safety equipments should always be present and located properly:

- Fire extinguishers (enough number and proper locations),
- Fire blanket,
- First aid kit,
- Warning signs (at the entrance, near the fuels, exits, etc.),
- MSDS forms,
- Emergency plans,
- Emergency phone numbers,
- Drill site plan.
- Eye wash station,
- Flashlights and spare batteries,

4.4 Trainings

There are three main training programs carried out under HSE&C procedures. These trainings were briefly explained below.

Fire training:

Fire training was performed regularly with the participation of all employees. Trainers were usually the representatives of private fire fighting companies. During these trainings all employees became familiar with use of a fire extinguisher, learned how to act during a fire and how to use a fire blanket etc.

- "First Aid/CPR training:

First aid training was performed regularly. Trainers were certified first aid trainer doctors. Trainings were held in drilling company's office. Training materials were available for every employee to practice the theoretical part of training. During these trainings employees learned how to act in an emergency case, what are the responsibilities of first aider, what is the aim of first aid, when and how to apply CPR, etc.

Safety training:

Safety training were performed regularly and especially if there was a new employee. Trainer was usually the HSE&C Manager. During these trainings, health, safety and environmental standards and rules were explained. Importance of obeying these rules was emphasized and previous experiences were discussed.

4.5 Checklists

There are some standard forms to be filled according to our HSEC program. These three forms were briefly explained below;

- Shift reports:

Filled by driller and controlled by the foreman. Basic information related to drilling was given in addition to some safety information. Questions related to protective clothing, tailgate meetings, safety equipments, etc., were included in standard forms used for shift reports.

Daily maintenance checklists:

To prevent any accident due to the lack of maintenance or any regular control of engine parts, daily maintenance checklists were filled. These were filled and signed by foreman. Questions related to rig engine oil level, pump's belt, etc. were included in these standard forms.

- Weekly maintenance checklists:

Again to prevent any possibility of an accident weekly maintenance is required for some parts and these forms guarantee that the required maintenance was performed on time. Similar to daily maintenance checklist some questions related to engine oil replacement, diesel filter replacement etc. were included in these forms.

4.6 Audits

There are four different audits in our HSE&C program as follows. Aim of these audits was to control the application of safety standards properly all the times.

- Drill site audits:

Audits are important parts of our program and performed by the HSE&C manager at least once a week. Frequency of audits can change depending on the situation of site. For example, if serious mistakes and lack of important safety equipment were recognized new audits could take place one day after the former one. Although most of the audits were scheduled, some of them were unexpected, such as midnight audits. Standard audit forms were used during the audits. In these forms questions about; use of personnel protective cloths, safety equipments, pressure levels of fire extinguishers, methane measurements, etc, were included.

- Vehicle audits:

Vehicle audits were also performed at least once a week. Standard forms used for these audits contain questions related to regular maintenance of vehicle, brakes, safety belts, fire extinguisher, first aid kit, chain, shovel, towing rope, etc. According to our standards one of the company cars should always be park on drill site keys are on, unlocked and ready to be used.

- Core-Shed audits:

These audits were not performed regularly since the company was not using the core-shed all the times. Standard forms used for core-shed audit includes questions related to the presence of fire extinguishers, first aid kit, fire blanket, eye wash stations, etc.

- Office audits:

Office audits were performed rarely since all employees were on site usually. Questions included in standard office audit forms are related to; exit signs, emergency action plan, fire extinguishers, first aid kit, etc.

4.7 Meetings

Meetings are another important part of our HSE&C program. Major aim of these meetings is to provide the active participation of all employees and make them to understand the importance of their willingness in the application of these procedures. In addition to these the procedures were repeated each time and some examples of accidents were given from other sites not applying such strict safety rules. Examples were also given from other countries. Sometimes comparisons and evaluation of our standards were made to encourage them. Followings are the meetings included in our HSE&C program.

Pre-hole safety meeting:

Before each hole a brief safety meeting was performed with the participation of all employees. During the meeting health, safety, environmental and community program being applied was explained brietly and discussed with them.

- Tailgate meetings:

These are short meetings up to 5 minutes and performed before each shift. Usually the driller talks about safely issues or they read a chapter from first aid handbook. It was emphasized them to read the book and look at the pictures in this book frequently to refresh their memories.

- Weekly safety meetings:

These meetings were performed by HSE&C Manager and the standard form including a brief summary of the meeting was signed by all participants. These meetings were performed between two shifts hence all employees could participate in it. Aim of these meetings is to discuss the standards and procedures applied in drill sites.

4M Reports

In our HSE&C program the paperwork is quite similar to all Environmental Management Plan studies based on ISO 14000 standards. Four different reports as described below were prepared and submitted to IMEx regularly.

- Weekly reports:

Weekly reports mainly constitute the daily and weekly maintenance checklists, shift reports, drill site audits, vehicle audits and weekly safety meeting forms. Observations during the audits were discussed in these reports. Topics covered in tailgate safety meetings were also included in weekly reports. Weekly report usually includes some photographs from drill site.

Monthly reports:

Monthly reports are generally the summary of weekly reports. Main difference of these reports is the monthly statistics related to lost time injuries, medical treatment case and first aid cases.

Hole reports:

After the completion of a drill hole, hole report was prepared including everything related to the hole. In hole report a picture of drill site after drilling operation was always present to show that the site was remained as clean as its original situation without any change on it. This photographs were compared with those taken before drilling.

Hole follow up report:

This report was prepared at least 3 months after the completion of a drill hole to show that the top soil was recovered completely. If the vegetation was not enough further study will be performed to obtain the same vegetation on drill site and its environment.

4.9 Community Relations

Community relations are also another important issue of our HSE&C program. In all environmental issues we should always consider the public involvement. One of the aims of our program is to have a good community relation and in almost all drill sites all employees had very good relations with local people. In some regions local people were very interested on drilling activities and want to see the site. In such circumstances we inform them about our safety procedures, warn them about any risk, made them wear hard had and steel toe boot and finally sign the visitor log before to enter the site.

- Visitor's log:

Visitor log contains the name of visitor, purpose of visit, date and hours of visit and a statement mentioning that the visitor was understand and accepted all safety rules and possible risks.

4.10 Event Loa

An event log was prepared when there was an important event such as an accident or a visit by governor. Event log describes the even stating the dale, place, involved parties, results, etc. and send to the IMEx within 24 hours.

4.11 Hole Follow Up

Although all drill siles were remained as same as their original situations, they were visited at least 3 months later to be sure thai the soil was recovered itself and vegetation is same as the surrounding. During these visits photographs of site were taken and included in a hole follow up report. It was observed that in all of our drill sites self-recovery was enough and no further vegetation was required. The recovery time, of course, depends on the region and climate.

5 CONCLUSION

Key issues to achieve a successful health, safety, environmental and community management are planning, training, auditing and reporting stages and each of (hem should be well organized. We are continuously improving our HSE&C program with the feedbacks from the current applications. We are proudly continuing our program after 150000 hours without any lost time injuries.

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Using of Expert Systems in Ventilation Systems Controlling

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ABSTRACT: In the paper results are presents of investigations on choice and justification of a type of computer system for control of mine ventilation systems in normal and abnormal conditions of exploitation.

I INTRODUCTION

Traditional formal mathematical models of mine ventilation systems not fully represent real object of control. This disadvantage compensates when using of principles of construction of expert systems (ES), which use experience and knowledge of specialists in specific field of activity. Presence at a market of software great volume of expert systems makes it difficult in choose the most suitable one. When comparing software with so various functions as expert systems, it is difficult to do any objective choice. That is why acceptable are, probably, three main properties of expert systems:

potential of a system;
- easiness of development of databases;
- effectiveness and ease of operating of end user with designed application programs.

Working out of application expert systems, in our opinion, must be carried out with taking account of a fact that FS operating will be. at in particular conditions of a particular enterprise, where providing the main regimes of any system operating is possible.

2 DETAILS OF DEVELOPMENT OF EXPERT SYSTEMS OF MINE VENTILATION SYSTEM

All possible variants of mine ventilation system (MVS) behavior both in normal and abnormal situations are forecasted in accordance with United Safely Regulations. That is why there is no necessity for complex expert shell. During this ES development it is necessary to take account of behavior of the main system of safety ensuring - mine ventilation system. Designing computer system must use programs for calculation and control of air distribution in a system of mine workings both in abnormal situations (rock bumps, outbursts, cavings, fires and so on) and in normal conditions of mine ventilation system operating.

First of all, let's concentrate on properties of necessary information when creating ES on safety for mining enterprises, and define what they represent and with what purpose they are used. In this connection, bases of input data must include information about mine workings, which is presented as records that include the following fields: number in order, initial and end nodes, name of a mine working, value of perimeter of mine working, its cross section, marks of height of its start and finish, aerodynamic resistance, initial and final temperature on mine workings and so on. So database must include considerable volume of information. For example, mines of Kazakhstan operating today have about 800-900 mine workings, and entering information about them by foregoing positions will take considerable time and require intensive work of specialists of ventilation system management. However, when entering of information some data may be wrong, and this causes cycling of programs for calculations, receiving erroneous results, which do not represent real aerodynamic processes of mine ventilation system. Besides, on the basis of these calculations emergency control plan will be formed, and using it may have catastrophic consequences. For decreasing probability of mistakes during initial information entering, ES systems of control are provided. Databases have large mass of numbers. For verification of reliability of initial data programs ICXINFOR and WENTCHAR were worked out.

ICXINFOR program automatically carries out calculation of branches and nodes of ventilation system and testing presence of missed branches. Input file of this program includes number of a branch, initial node, end node. For printing table outputs, followings are included:

- number of mine workings in ventilation system - LN;
- number of nodes in ventilation system - YZ;
- maximum number ol a branch in ventilation system - max V;
- maximum number of a node in ventilation system - max YZ;
- number of nodes of surface - YZ pov;
- number of fans - NVEN;
- accuracy of calculations on air consumption - DQ.

In the process of operating, the program tests presence of missed branches. If there are such branches, the program outputs message: "Branches number ... are absent". If all branches present, the program outputs message: "Missed branches are absent".

For operating of program of forecasting calculations of air distribution in mine ventilation system, it is necessary to give characteristics of fans. For promoting of calculation of fans' coefficients, program WENTCHAR was worked out. For giving of curve of aerodynamic characteristic of any fan, it is necessary to know fan's type and angle of inclination of blades of the impeller of the fan.

Functions of ES program shell on maintenance of a user must be enough developed and ensure dialogue with the user in his language. They must have developed multi-level net of menu and lists of help on different problems of control of mine ventilation systems, give at a screen of display or printing explanation of taking one or other computer decisions in language, and must be comfortable for users. Let's study functions of ES program shell by above-mentioned positions. Besides, shell of expert system must take into account, that initiator of dialogue may be both itself and the user. In the first case ES passes to dialogue regime, if it detects the presence of discrepancies, which require revision of the user at any stage of ES operating. The second case is commanding for ES and this fully depends on the user wishes. Here ES shell must provide, that the user may give a command to ES transition to dialogue regime at any stage of formulated problem solving. So, when ensuring of dialogue regime ES program shell must functionally take account of two main positions:

1) two sources of initiating of dialogue regime: ES and user;

2) deformalization of current information up to language and terminology, comfortable for those users, who has no special knowledge in a field of programming and computer systems using.

Menu system, as a whole, may be presented as a tree, in root of which initial menu is located, representing, for example, kinds of emergencies at underground mining enterprises: fire, explosion, caving, rock bumps, water inflow, quicksand and so on. Every kind of emergency is a beginning of tree's branch with corresponding menu of different levels. For example, for emergency "fire" following menu may be a list of places of fire (ore yard, main workings, chambers and so on), then what is on fire (support, spontaneous ignition, cable, equipment and so on) and then condition of protective gears in lire conditions (ventilation system, fire-extinguishers and so on). The end top of a way on menu tree is shell transition to forming of initial information and decision of given problem.

Lists of variants of help for user ES uses at a stage of initial information forming after passing menu tree up to end top. For promoting of a search of necessary variants of help these lists may be structurally united also in a form of a tree, which is analogous to the menu tree, where this or that list of variants of help corresponds to specific menu.

The shell must functionally ensure output on display or printer current information about ES operation. User may do inquiry at any stage of problem solving.

Expert system "Safety" includes the following service programs:

- testing of correct input of initial data;
- testing of the first law of networks;
- data on mine air dams;
- program for calculation the coefficients of fans curves;
- program for view of calculated data.

3 CONCLUSIONS

On the basis of carried out investigations demonstration version of expert system "Safety" was worked out. This program product allows solving problems on calculation of mine ventilation systems both in normal and abnormal conditions. It was tested in practice and it is of interest for workers of ventilation service, mine rescue service and State mining inspection.

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Prediction of Methane Emission Out of Seams on The Basis of Geodynamic Deposit Zonation

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ABSTRACT. By the results of geodynamic zonation of coal deposits in the south-eastern part of Kuzbass the districts of predominant compression (domes), of extension (subsidence troughs) and of areas separating them are rather confidently delineated. Within them the rock mass is under relatively stabilized state.

The discovered regularity for the zones of dome and subsidence trough has the obvious explanation. As a result of geodynamic peculiarities of relief formation: the horizontal compression is accompanied with "hummocking" of lithosphère blocks with the alternation of the districts of predominant compression or extension.

Localization and characteristics of geodynamic processes on the territory and in the mines of Kuzbass including geodynamic deposit zonation are being done on the basis of geoinformation system (GIS) "Geodynamic phenomena in Kuzbass'. Information system of geodynamic phenomena in Kuzbass presents itself an automated system of data base and electronic cards of natural and technogenic phenomena.

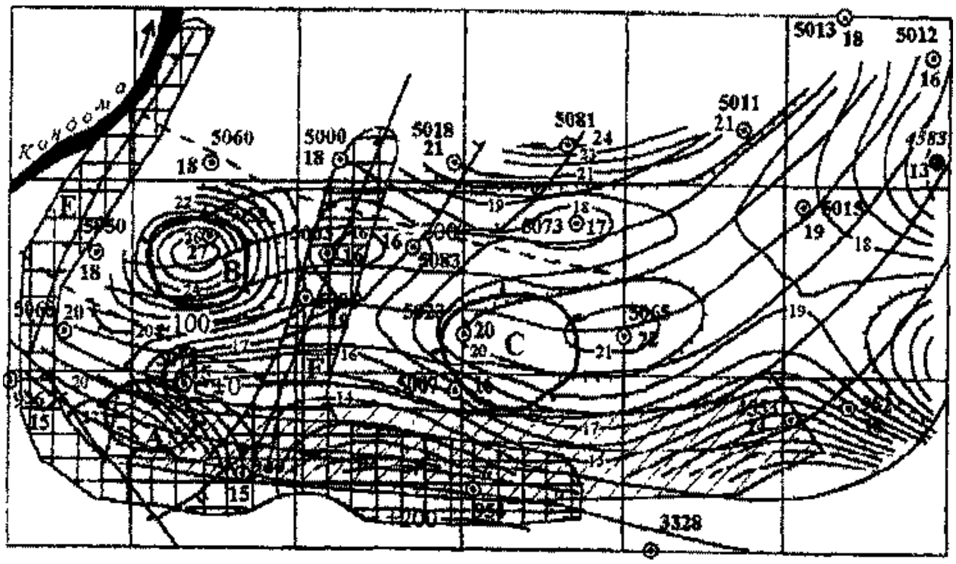
I INTRODUCTION

Methane content of coal deposits in Kuzbass grows in conformity to the increase of degree of metamorphism of coals and reaches the maximum in hard coal 10A. This relationship is based on the analysis of the search results for coal deposits and was the basis for the estimation of predictive resources of methane in the Basin and for the choice of more perspective districts within it (Erunakovsky, Tersinsky, Tom-Usinsky, Mrassky). There are some reasons to consider that the output of the operating holes in the designed coal-gas works and the degree of danger of gas-dynamic phenomena in the mines are determined with modern geodynamics-characteristics of spatial strained - deformed state of carbonaceous rock in the rock mass.

2 GEODYNAMIC STRUCTURES OF MINE TAKES OF COLLIERIES AND GAS CONTENT OF A SEAM

By the results of geodynamic zonation of coal deposits in the south - eastern part of Kuzbass the di.stncts of predominant compression (domes), of extemon (subsidence troughs) and of the areas separating them are confidently enough contoured. Within these areas the rock is under the relatively stabilized state. The detalied elaboration of geodynamic reconstructions within mine takes of collieries Alardinskaya, Tomskaya, the Shevyakov showed that maximum gas content and the manifestation *of* dangerous gasdynamie phenomena as a result are typical to neotectonic domes having been formed under the conditions of horizontal compression of rock mass. The subsidence troughs are characterized with minimum gas content. The example of the geodynamic zonation of "Alardinskaya" mine's field is presented in figure 1.

The exposed relationship for the zones of dome and subsidence troughs has the obvious explanation as a result of geodynamic peculiarities of relief formation: the horizontal dome is accompanied with "hummocking" of the lithosphère blocks with alternation of the districts of predominant compression or extension. The extension along the vertical axis corresponds to the horizontal compression and vice versa. Therefore within the domes there is a possible formation of subhorizontal cavities of exfoliation of coal -bearing deposits in which the concentration of free hydrocarbon gases can be formed. As a result of the predominant compression of coal-bearing rock along the vertical axis subsidence troughs being exposed to natural degasillcation serve as an additional source "alimentation" with free methane for adjacent domes. One can suppose that natural "pump" which provides the methane migration along the lateralia



Conventional signs:

21⊙ 5011	well, numher. gas content, m /t dry ash/free		subsidence troughs
	worked oui part ot the seam		vertical sepaıatıons of dischaiging gas-liydi «dynamic type
±0	contour line of seam's flooi		sub-vertical faults (mainly discharging ones)
12	ısogas. m /t. dıy ash/fiee		mainly sub-vertical displacements and fractures with non-established kinematics and dynamics
A B C	contouis of domes		

Figure I Results of geodynamic zonation of the field of mine "Alaıdıııskaya" (seam 3 - 3').

out of the subsidence troughs into the domes is the creep movements of earth's crust, the phenomena of earth's tides, other seismic and geodynamic phenomena.

The model of the mechanism of dome formation is presented in figure 2, and the calculations made by means of final elements for two-dimensional computer models corroborated the possibility of dome formation and open cavities under them. The upper boundary of gas-bearing domes is from 250 to 300 meters, the depth of the deposits can have several hundred meters of coal-bearing formations. The domes and subsidence troughs are low amplitude neotectonic structures.

The amplitude of the domes (gas reservoirs) in the districts of mine fields of collieries Alardinskaya, Tomskaya, the Shevyakov is from 1,0 to 3.0 m. The amplitudes of the shift of neotectonic upthrow faults into the domes didn't exceed 1,0 - 2,0 cm.

The dynamic activity of these structures was studied with the method of the registration of natural impulse electromagnetic radiation of the rock by means of VNIMI hardware of type "Impulse". The investigations were carried out with the method of dipole electromagnetic profiling on the surface and under the underground conditions in the workings contouring the longwalls.

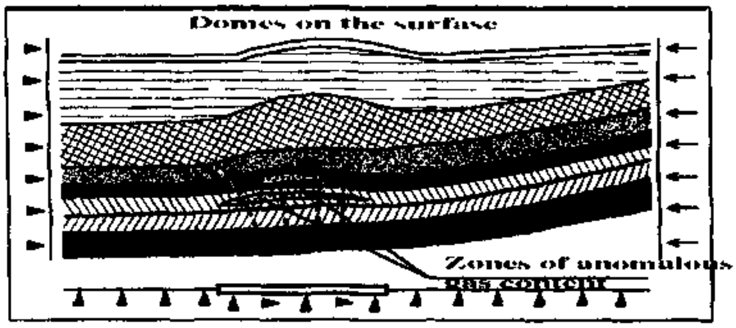
As a result of studies more then 5000 measurements were selected:

- zones of compression (domes) having permeability close to zero;
- inclined upthrow faults, thrusts and underthrusts (mainly filming).

The increase of gas content per 40 - 70 percents is typical for these structures.

The structures with low methane content are:

- zones of extension (subsidence troughs, mines "Alardinskaya", the Shevyakov);
- vertical separations of discharging type (mine "Alardinskaya");
- vertical separations of infiltrated type (mine "Tomskaya");
- subvertical faults (mines Alardinskaya, Tomskaya, the Shevyakov).



Signs

fixation ot veilical displacemenl

fixation of horizontal displaLemenl

iiend ol hoiizonlal piessuies.

ugid blocks ot substratum basemenl

zones ot anomalous gas content

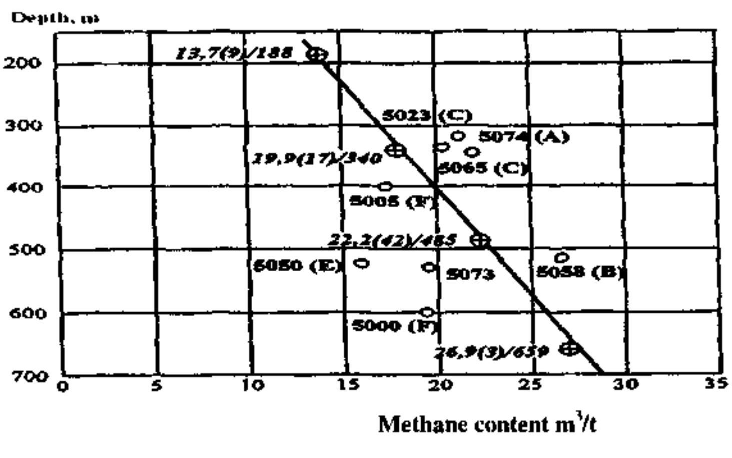
displaceis or upthiow oveitaults inside the dome sliuctures

Figure 2 Model of mechanism ol loimation ot dome (compicssion /ones) and subsidence Houghs (extension zones)

Foi these sliuctures the deciease ot gas content 50 - 70 percents is typical It is horn expected gas content accoidtng to the tıadıtıonal prospecting estimations (accotding to the charts "methane" - "the depth ot sampling selection")

In Figure 3 the chart ot change ot the depth of the gas content ot seam 3 - 3[1] in mine "Alaıdınskaya" in the dıslıct wheie mining operations aie being conducted is presented In table I there are the data ot the sampling ot gas content ot seam 3 - 3 in the distnct ot the active structures geodynamically pointed out

It is seen in Figures 1, 3 and Table 1 that between the indicators ol gas content and the position ot



Signs

© *22,2{42)/48S*	avuage methane content (quantity ol used samples) aveiage depth of selection
O 5000(F)	position ot the sample on the chart number ot the well and belonging to dome (A B C) or to subsidence tmugh (E F)

Fıguıe 1 Combined chait ot change ol ihe depth ot the gas content of coal seams and the position ol sampling points on gas ot seam 1 - T' (field ot mine Alaıdınskaya) on domes (holes S074 5058 5065 5021) on ınteımediale stiuctuies (hole 5071) and on subsidence troughs (holes 5050 5005 5000)

Table I Content of methane in seam 3-3[1] m separate geodynamic structures on the field of mine "Alarclmskaya"

jsr» item	Number ot well	Depth of sampling selection, m	Content of methane. mVt
		Domes (zones of compression)	
I	5074	300	20.1
2	5058	518	27.4
3	5065	391	21.5
4	5023	368	20,4
		Intermediate structures	
s	5073	529	17.3
		Subsidence troughs (zones of extension)	
6	5050	508	16,0
7	5005	412	18.0
8	5000	581	17.7

boundaries of the structures being pointed out there is rather a convincing correlation:

- anomalously indicators of gas content corresponds to the zones of compression (domes);
- anomalously value of gas content corresponds to the zones of extension (subsidence troughs).

Dome "A" in fig 1 uncovered with workings is characterized with such intensive absolute and relative gas content (up to 40 m'/t) - that a three - year complex of operations was required to degasificate the district of seam 3 - 3'[1] which is prepared for working.

Dome "B" is uncoved with hole 5058. According to the calculated data of the chart in fig. 3 the value of gas content was to be expected within 21 m^3/t. The actual data of sampling were 27,4 m /t.

Dome "C" is uncovered with holes 5023, 5065, sampling selection on gas content gives value 20-22 m'/t, according to the chart in fig.3 the value was to be expected 16 m'/t.

In the zones of extension (subsidence troughs), there is low gas content, so in the extension zone "E" instead of expected gas content 23 m A, 16 m''Vt was received with actual sampling (hole 5050).

Within the extension zone "F" due to the samplings of holes 5000 and 5005 gas content of the seam was 16 and 18 m7t instead of 17.7 and 24 m'/t which were expected due to the chart (fig.3).

Thus the results of geodynamic zonation being done by means of methodics of VNIMI allows to determine linear and area structures promoting to preserve and to accumulate the methane in the coal rock mass. i.e. to give the possibility to predict the location of gas reservoirs.

To the structures with increased methane content in coal seams arc referred:

- the districts with low level of stress, characterizing the subsidence troughs, in these districts the intensity of radiation "N" was from 0 to 200 imp/min;
- the districts of unstable rock mass state (according to the series of repeated observations). corresponding to the zones of transition from the domes to subsidence troughs "N" = 200-600 imp/min;
- the districts of anomalously high intensity of impulse electromagnetic radiation ("N" >600 imp/min) corresponding to high level of stress in the rock mass-districts of the domes. In the district of mine "Tomskaya" "N" reached 1000 imp/min;
- the districts of spasmodic changes of radiation level characterizing the boundaries of structures and active fractures.

The checking of the results of geodynamic prediction in reference to the tasks of the estimation of natural seam gas content is carried out over the sampling data at the exploratory drilling and at the gas content data of development workings and worked out faces in mines Alardinskaya, Tomskaya, the Shevyakov.

3 COMPLEX OF MEASURES FOR GEODYNAMIC ZONATION AND PREDICTION OF NATURAL GAS CONTENT

Taking into account all these we consider that before to start the construction of the mine (a block, a flank) geodynamic zonation of deposits must precede the map development of gas content, the planning of degasificated measures

From the example of mine takes of collieries Alardinskaya, Tomskaya, the Shevyakov it is seen that the most dangerous complications of coal mining conditions, being caused with anomalously high seam gas content are connected with neoteclonic fractures, with the zones of linear concentration of dynamic stress in rock mass, and also with low amplitude neotectonic fold structures; subsidence troughs and domes. In principle the mentioned neotectonic categories can't be registered with a borehole exploration and are established only by means of geodynamic zonation with the use of special methods of investigation. Such investigations are: special methods of processing of structural hypsometric surfaces of modern relief, the interpretation of aerial space photographic materials, conjugated analysis of received mapping data with the use of geological and geophysic materials, the calculation of derivative features for final geodynamic map development, carrying out the complex of field geological, geophysic, gas-hydrodynamic observations for the verification of the positions of active fractures and for the delineation of potentially dangerous districts. The mentioned complex of operations under the most responsible situations can be added with volumetric mathematic modelling of strained-deformed state of rock.

The experience of work in geodynamic zonation testifies the necessity of the fulfilment of such zonation with the method of consistent detailization of the scales: a coal basin, a coal district, a coal deposit, a mine take of the colliery. Some individual methods of geodynamic zonation are effective and at more detailed forecast can be used for panels and their separate parts. Acceptable results of the detailed forecast can be received at the additional certification of the predictive constructions by means of VNIMI seismoacoustic hardwares on the surface and in the development workings contouring the panel.

At the map development of gas content and planning of degasificated measures and also at the organizing of methane mining in the coal deposits it's necessary to count up the results of the geodynamic zonation. The first and the main measure in this direction is to organize the work in geodynamic mapping of Kuzbass territory, which is still in the very initial stage. Less then 10 - 12 % of coal deposits are covered with the survey, the scale is 1 : 25000, with the technological scales, accepted for the underground operations (1 : 5000) is much less.

4 INFORMATION SYSTEM OF GEODYNAMIC PHENOMENA IN KUZBASS

One of the first measures in the information systematization of geodynamic zonation in Kuzbass is the creation of information systems of geodynamic phenomena.

Information system of geodynamic phenomena in Kuzbass is a final part of the monitoring system of geodynamic phenomena on the area and of geomechanic (including geodynamic) manifestations in the mines of Kuznetsk Basin. The monitoring system being worked out is a complex system of the regulated observations, the estimation and the prediction of the mining, physico-technical phenomena .including geomechanic and geodynamic ones.

The information system of geodynamic phenomena in Kuznetsk Basin presents an automated information system of electronic cards and data base of various geodynamic phenomena, which take place on the area and in the mines of Kuznetsk Basin.

Electronic cards being used in the system are based on the digital electronic geographical map of Kuznetsk Basin. The digital electronic geographical card is developed from the geographical map the scale of which is 1:500 000 and contains geographical system of coordinates (from 52 to 57 degrees of North longitude and from 84 to 90 degrees of East longitude), the biggest town and populated areas of Kemerovskaya region;

- water basin of river Tom with tributaries;
- coal industrial districts and coal reserves of Kuznetsk Basin;
- functioning and closed coal mines and opencasts of Kuzbass;

Information system of geodynamic phenomena in Kuznetsk Basin consists of three blocks, which are realized with data base of given phenomena and electronic cards:

Block I. «Geodynamic zonation of Kuznetsk Basin area».

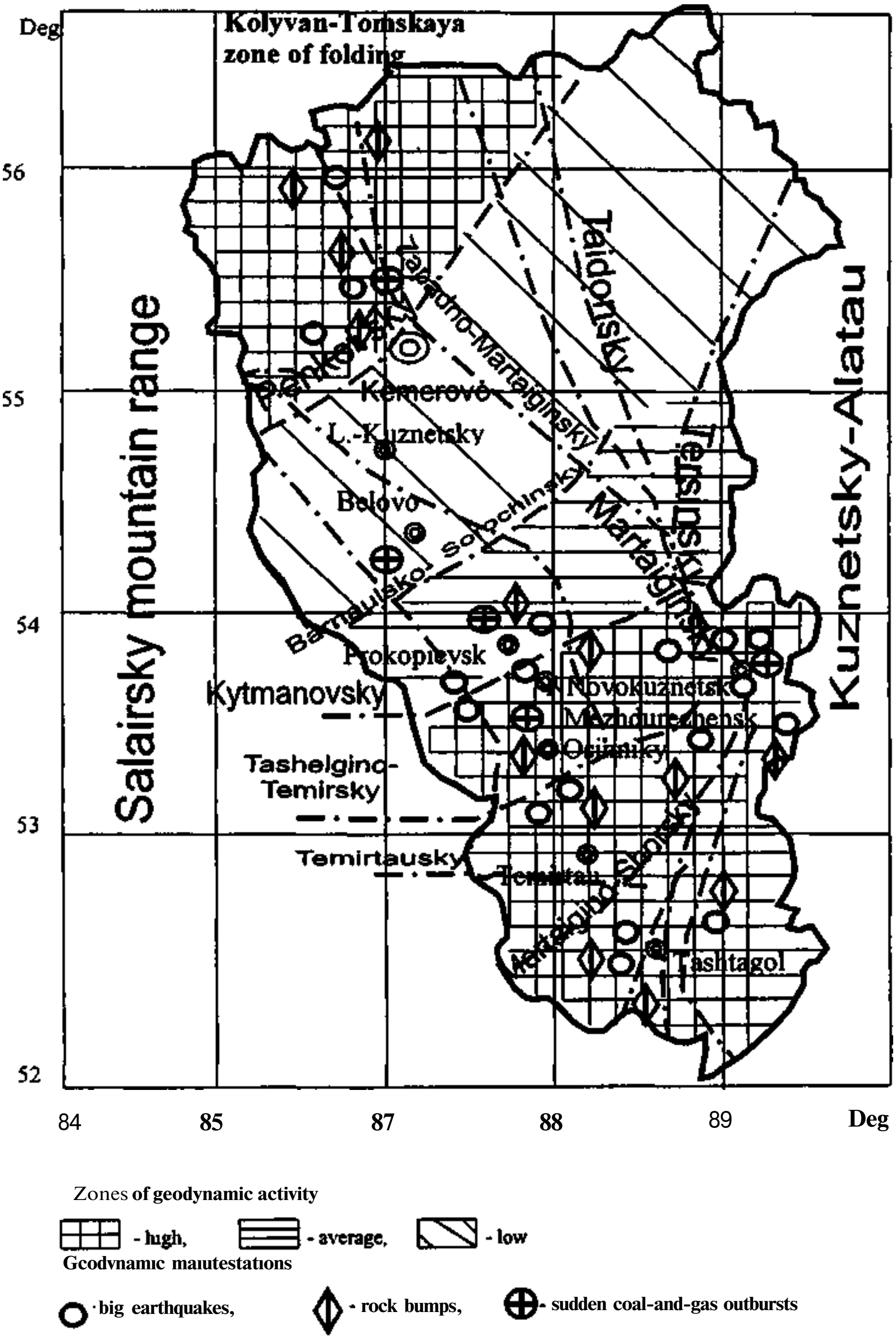
The electronic cards of this block show the locality of tectonic fractures, the boundaries of the blocks and the zones of various geodynamic activity in Kuzbass. In Figure 4 is given the model of the map of geodynamically active zones in Kuzbass.

Block 2. «Great geodynamic natural and technogenic phenomena on the area of Kuznetsk Basin» includes the data hase of phenomena:

- «historical» earthquakes from 1734 to 1943 years;
- modern earthquakes from 1943 to 2000 and 2001 years;
- industrial explosions, mining tectonic rock bumps and sudden coal-and-gas outbursts.

Kuzbass is a modern seismoactive region, where about 20 big earthquakes with amplitude of 3.6 - 6.5 took place during the last 200 years, and annualy hundreds of earthquakes with less amplitude are fixed; the most part of them presents itself technogenic seismic events connected with active exploitation of Kuzbass bowels. The considerable increase of quantity of technogenic phenomena is registered during the last decade.

Great geodynamic phenomena are determined with event time, a co-coordinate and phenomena power. On the basis of data base the electronic cards of geodynamic phenomena are built and the estimation of phenomena is made. The construction of the electronic cards is being made with the possibilities ot classification ot events both according to time and to power ot phenomena



higuie 4 Map of geodyiumic donation o! Kuzbass

Taking into account the big historical earthquakes, the biggest part of Kuzbass area up to 2000 year related to the hexaball zone of tremor according to the map of the seismic zonation. Since 1898 the big earthquakes (as seen in Figure 4.) are registered in the locality of Novokuznetsk. Here the biggest natural Kuznetsk earthquakes in Kuzbass look place (1898,1903,1966 years) and also recent Prokopievsk and Mezhdurechensk earthquakes (1995 and 1998 years) a monosemantic reply for the nature of which isn't received yet. By the present time on Kuzbass area 853 natural earthquakes have been registered, energetic class "K" from 7 to 15.

Block 3. "Rock bumps in the mines of Kuznetsk Basin. "

The block contains the data base of rock bumps, which have taken place in the mines of Kuzbass since the moments of their origin. On every rock bump is given the information about the association, the mine, the time and the place where the bump has taken place, about the depth, the seam's data, the characteristic of adjoining rock, the position of mining operations at the moment of the bump, about the phenomena preceeding the bump, about mining operations at the moment of the bump, about the characteristic of the bump manifestation, about the fulfilment of the measures warning rock bumps. The information about the rock bump from the data base can be presented graphically in terms of schemes of mining operations and sketches of actual sight of bump manifestations in workings.

In the block, there is data base of mines and coal seams liability to rock bumps, it is made on the ground of «The catalogue of coal seams liability to rock bumps of Kuznetsk Basin» CSt. Peterburg. 1996). The data base contains the information about the mine's name, the seam's name and its index, the mining operation depth having been reached, the minimum depth from which the seam is related to threatened and dangerous one being liable to rock-bumps. In all the mines of Kuzbass there are 266 seams and seam districts liable to rock bumps.

The block includes data base of geodynamic states of unworked coal in the mines of Kuznetsk Basin. It is made by the mine gcophy.sic measurements of rock mass state under various mining - geological conditions and by the mining situation of panels of Kuzbass mines which have been carried out by Kemerovo Representation of VNIMI. Geophysic measurements characterize the level of rock - bump hazard and allow to determine the zones of the coal seam liable to rock-bumps. The disturbance level of rock mass is determined with a sounding of rock mass in the locality of development faces.

The analysis of the technological parameters of the operations on the methane extraction out *of* the coal seams showed the priority of geodynamic processes at the formation of methane reservoirs on the coal seams. This circumstance motivates the applicability of the morphometry and other methods of computing of remote sounding for their discovery.

Detailed elaboration of geodynamic reconstructions within mine allows to determine the maximum gas content and as a result possible manifestations of dangerous geodynamic phenomenon. On the basis of geodynamic zonation in the mines it is possible to predict the methane anomalies in the coal area.

5 CONCLUSION

The work experience in geodynamic zonation shows the necessity of their fulfilment with the method of successive detailed elaboration of the scales: a coal basin, a coal district, a coal deposit, a mine take. Some individual methods of geodynamic zonation are considered to be effective at more detailed prediction in the districts of the panels and their separate parts. Acceptable results of the detailed prediction can be received at additional predicted buildings by means of seismoacoustic appliances on the surface and in the development workings of the panel.

At the building of gas content maps and at the planningof degasification measures and also at the organization of methane extraction out of coal deposits the registration *of* geodynamic zonation results is necessary. The first and the main measure in this direction is the organization of the work in geodynamic mapping of the area.

The developed geoinformation system (GIS) "Geodynamic phenomena in Kuzbass' consists of blocks of electronic cards of geodynamic phenomena and geodynamic zonation of Kuznetsk Basin area, of catalogues of rock bumps, of mines and coal seams liable to rock-bumps, of in-situ data and pictures of geodynamic rock mass state in the mines.

1 ê/' International Mining Congress and Exhibition of Turkey-IMCET 2003, Cô 2003, ISBN 975-395-605-3

Health and Safety at Work in The Romanian Mining Industry Present and Future

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ABSTRACT: The mining activity implies a certain degree of familiarity with the underground environment, and the risk factors determined shall play their part on condition they become known before reaching a certain degree and stage of development. Accordingly the basic concept for the new legislation in the field of labour protection in Romania starts from the idea the risk prevention and health and safety assurance should be a constant concern of all the person involved in the working process. The paper present a short review of some aspects related to the current situation of occupational diseases and accidents in the Romanian mining industry. Based on this analysis, there are also shown the problems the National Institute for Mine Safety and Protection Proof Explosion - INSEMEX Petroşani is involved in. as well the results in the activity developed to assure good safety and health condition.s in the Romanian mines, viewing the harmonisation of the Romanian legislation with the legislation in the European Union.

I HEALTH AND SAFETY INDICES OF THE MINING SECTOR BETWEEN 2000-2002

For the above-mentioned period, both the mining and geological sectors faced a lot of occupational accidents, diseases hazardous incidents and events with human victims and important damages.

The values of labour accidents recorded in the mining activity, divided for each sector of the extractive industry and compared to those values of the whole industry in Romania show a strong increase; the main reason of this situation is an inadequate health and safety policy led by the managing teams.

The analyses carried out in each company or society in the mining sector show a low safety level at the working places with a diminished efficiency of the decisions taken for the development of an adequate working climate.

The diagrams 1 and 2 show the situation with the accidents in the mining sector for the above-mentioned period and their weight in the total number of accidents recorded in the whole economy.

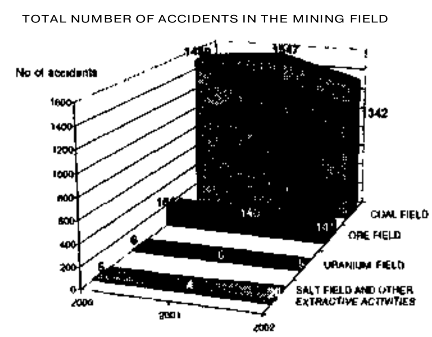

Figure 1

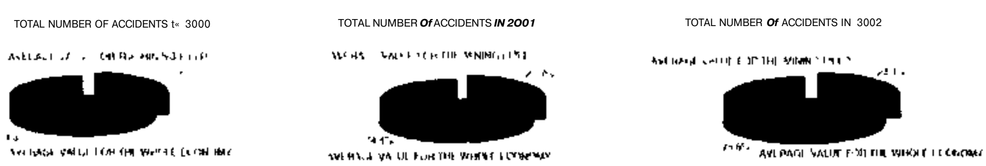

Figure 2

The weight ot lethal accidents ıated to the total number of accidents produced in the same penod tor the main extractive sectors in Romania is shown in the diagram no 3

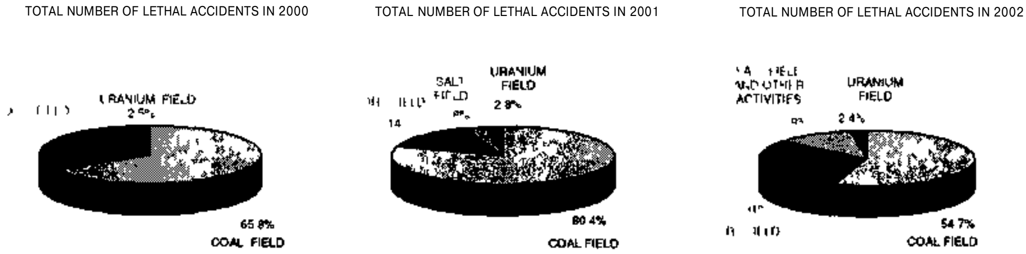

Figure 3

As tor the indices ot operation that define the activity ot labour protection (index ot frequency, index ot seuousness and index ot average period), the diagrams 4,5 and 6 make a comparison between the situation in the mining sector and the one in the whole economy

The weight ot occupational diseases in the mining sector out ot the total value recorded on the economy with the main types ot occupational diseases are all shown in fig 7 and the material damages due to accidents and occupational diseases aie shown in Figure 8

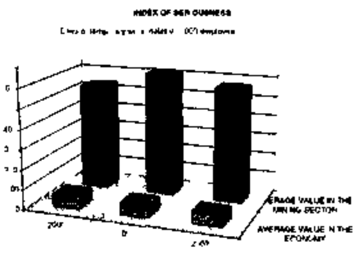

Figure 4

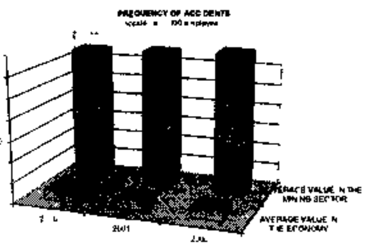

Figure 5

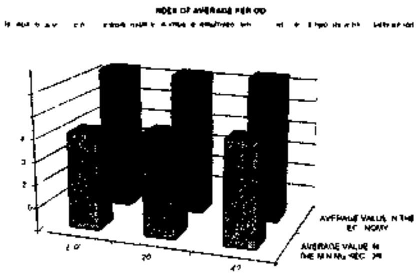

Figure 6

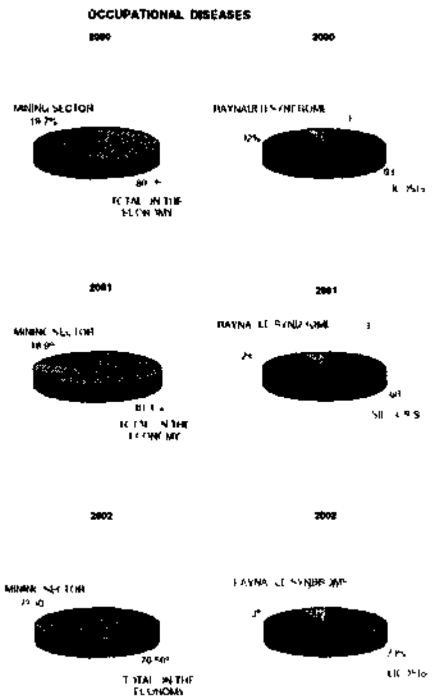

Figure 7

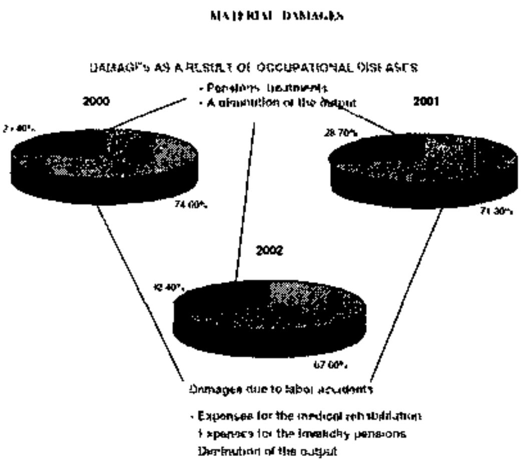

Figure 8

The labour accidents and occupational diseases had originally the following causes:

a) Use inadequate technical equipment:

- the use of some working gear that hasn't been previously certified or verified accordingly, with a low degree of safety during the operation periods;
- the implementation of old technologies with a high risk degree on of some non-authorised and non-certified procedures;
- a lack or an inadequate operation of measuring and survey devices in those places who need regular check-up all through the working process ;

b) A faulty organisation of the working flow and of the staff training at the working places:

- the rules for health and safety at work are not implemented in the production process and on each working stage;
- the measures for a preventive safeness are implemented sporadically;
- the occurrence of a less strict working climate as the authority of the managing staff has diminished
- a faulty organisation of the working stages
- the lack of a less rigorous system for the selection, training and use of the personnel depending on their professional skills;
- a lack of balance between punishments and rewarding measures for all those persons involved in the safety and health at work;
- a lack of an adequate professional training;
- a lack of detailed training system, with a practice implementation so as the working personnel should get the skill, experience and awareness necessary to work in a safe manner;
- gaps in the supply with materials, working gears and protective equipments.

c) The responsibilities are not pearly defined for each person and the part played by each participant in the working process and by each expert organisation structure are not clearly specified with respect to the safety of working places.

2 INCREMENT OF HEALTH AND SAFETY IN THE MINING SECTOR

The registration and the strategic program with respect to some special initiatives necessary in the mining filed are a stringent necessity considering the present situation in this sector that can be characterised by specific risks of occupational accidents and diseases, difficult working conditions and problems in the training of the personnel; also, there are problems in a safe limitation of the working areas, in the system used for survey and warning in the working environment.

The mam aims for the both the near and the distant future are the following ones :

- the development of a complex training system, divided into professional categories;
- the implementation of survey, support and protection systems suitable for mine beds and based on well - documented studies from the point of view of labour safety;
- improvement of the means and methods used to determine the environment safety parameters and the purchase of modem apparatus suitable for this purpose;
- improvement of the present system with respect to the organisation of the preventive medical assistance.

One of the main requirements in a market economy is a high professional training of the labour force, i.e. it is necessary :

- a suitable recruitment, selection and employment of the labour force;
- the development of a suitable organisation and motivational framework with a clear underline of the rights and of the commitments of each worker in the working process;
- a suitable training and specialisation of the labour force;
- a suitable health and safety protection and a social protection of all the employees.

All these clauses are included in a health and safety program orientated especially towards a suitable supply of health and safety at work and towards a good social protection of the labour force. This program is necessary both due to economic reasons and due to the implementation of the Law for the assurance against labour accidents and occupational diseases currently in force.

The main aim of this program is to eliminate or to diminish as much as possible the unhealthy and unsafe working conditions and the hazardous operations that might produce accidents and occupational diseases.

The program includes two basic elements that refer to:

A) The prevention of all types of accidents and of occupational diseases;
B) The rehabilitation of the injured persons and of those persons recorded with occupational diseases.

The prevention bases on the development and implementation of a complete and permanent system for the promotion of information and training of the working personnel and of the managing team.

The assessment of occupational risks shall have a permanent character within the working units, with the consideration that both the conditions at the working place and the working conditions may change in time. Also, an assessment of risks aims to make the workers think, understand and recognise dangers and take all the necessary correction measures, fully aware of everything.

The program for the rehabilitation of the injured people or of the persons affected by occupational diseases aims to settle all the problems and concerns of those who suffered an accident or suffer an occupational diseases for the purpose of their social and economic re-integration.

The medical, professional and social rehabilitation is made based on framework agreements signed with medical units, for special rehabilitation, agencies for the training of the labour force.

3 PART PLAYED BY INSEMEX PETROŞANI AND THE PROBLEMS APPROACHED IN THE FIELD OF HEALTH AND SAFETY AT WORK

The industrial practice has shown that all the activities and especially mining are accompanied by risks that may give birth to several technical problems and further on, to labor accidents and / or occupational diseases. As the damage and human deaths cannot be accepted, no society allows to let free the development of such unfortunate technical problems.

To keep risks under the acceptable limit it is necessary to devise a set of measurements with the purpose of prevention and imposed by the rules settled in a society; to this end the scientilk research can play an important part.

Accordingly, the National Institute for Mine Safety and Explosion - proof Protection - INSEMEX Petıoşani - has been nominated as a state authority in the following fields of activity;

a) verification, approval and certification of the documentation equipments, electric-installations, materials and of home-made or imported apparatus that are to be used in potentially explosive atmospheres and the certification of economic agents involved in the manufacturing and marketing of these products;
b) certification of the personnel involved in designing, mounting, maintenance and repairing operations carried out in industrial areas classified as hazardous from the point of view of the probability of occurrence of explosive atmospheres;
c) verification and approval of explosive materials and of means of irritation as well of blasting technologies;
d) training and certification of the personnel that operates in explosive and / or toxic environments;
e) classification / re-classification of mines from the point of view of their emissions of toxic and / or explosive gases.

As it is the only research institute in Romania that makes approaches of health and safety aspects in mining, there has been recorded an ever diversified implication all through the years. Now, based on the results, it can better support the development of the above - mentioned Program for health and safety.

Among the latest study aspects that are finalised now or are being developed, the following ones may be cited;

V development of a Health and Safety Manual for lignite mining in Romania that meets the requirements stipulated in the Framework Directive no. 92/104/EEC. In the near future, a same type of manual shall be developed for the hard coal mines;

> after an analysis of the labour risks in the mining activity, it has been considered as necessary to determine the most decisive factors that help an improvement of the working conditions and support the implementation of the programs for health and safety; accordingly, there has been produced a method that can be used assess risks at the working places. It based on a safety policy that aims its management for the personnel employed by the mining units;

> the technical part in ATEX Directive has been included in the Law for Labour Protection no 90/1996, including the part related to the harmonised with the European standards (CEN/CENELEC). Of this series of standards, the main standards for the protection to explosion have been transposed into the Romanian standards at the initiative of the Romanian Technical Committee 137 and of INSEMEX.

the department of certification, recognised by the Ministry of Labour and Social Protection is the most well equipped in Romania ; it has special installation for tests to explosions and it is in process accreditation by Physikalish-Technische Bundesanstalt in Braunschweig;

the development of a large program for the survey of health and safety parameters specific to hard coal mines located in the Jiu Valley. The final results shall support the development of framework mining programs;

implementation of new technologies used to prevent and fight against endogenous fires in coal mines; these technologies help to diminish the output losses and make all the coal deposits safer;

a co-operation with the specialists in the Ministry of Industry and Commerce and with counsellors of the World Bank for the development of a Manual for Mines Closure; this document ca offer a legal framework for the re-organisation of the extractive industry in Romania, process that started in 1998.