European Commission

Research Fund for Coal and Steel

Optimisation of surveillance, technical equipment and procedures to prevent workers from danger attributed to fire, hazardous or toxic gases, firedamp or climatic conditions

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1 FINAL SUMMARY

The project research was divided into the following tasks:

- Highly sensitive system for measuring and analysing gas (DSK, WP1)
- Preventive maintenance of sensible electrical installations used in mines susceptible to explosive atmospheres or firedamp (2 partners: CdF, WP2, and INERIS, WP8)
- Improved gas capture and climate control within high performance workings (DMT, WP3)
- Climate studies of arduous working and emergency activities (MRSL, WP4)
- Reduction of costs associated to gas control in caving faces (2 partners: AITEMIN, WP5, and SAHVL / HUNOSA, WP7)
- Improved gas capture, control and management within high performance workings (UNOTT, WP6)

The bar charts of each work programme are illustrated by point 4.

1.1 WP 1 - Highly sensitive system for measuring and analysing gas (DSK)

In the course of the project work the under-ground atmosphere has been characterized by the use of an FTIR spectrometer and gas chromatograph investigations and simulated by lab measurements. For initial investigations and for the field tests carried out at the end of the project time, an under-ground infrastructure has been built up consisting of a pressure-resistant housing with communication possibilities to a PC above-ground. Apart from this, an electronic hardware platform specifically adapted for reading out sensors under the specific condition of their response in the under-ground atmosphere (high required measurement range) has been designed and realized. A decision algorithm allowing the discrimination between alarm and non-alarm situations has been developed and implemented using specific software tools. The algorithm has been successfully tested under lab conditions. The results obtained in lab measurements and first results from field tests showed conformity so that a portability of the developed decision algorithm to an under-ground system is feasible.
The research work for developing a highly sensitive system for measuring and analysing gas will continue by at first testing the prototypes in underground trial run.

The overall approach of using semiconductor gas sensors for early fire detection under-ground is very promising especially due to the inherent sensors sensitivity and the selectivity obtained from the special read-out approach of temperature cycling. The German test laboratory for mine ventilation, EXAM, made an assessment how the existing test standards which are currently applied to CO gas detection apparatus for use in hardcoal mines can be applied to such apparatus with new techniques of signal processing (evaluation of the signal of a virtual sensor array). Proposals for the extension and modification of the current European standards concerning the performance requirements and test methods for apparatus for the detection and measurement of toxic gases (EN 45544 series) were formulated. Additional laboratory examinations were performed on a prototype apparatus on a trial basis.

The advantage of the use of special semiconductor sensors - in comparison of the current available CO-measuring systems in coal mines - makes it already possible to measure concentrations of gases within the ppb range (instead of ppm). This means that coal mine fires can be detected during the initial stages and suitable counteractive measures can be early introduced. Risk to employees by toxic carbon monoxide concentrations or explosive gas concentrations in the mine can therefore be minimised.

1.2 WP 2 and WP 8 - Preventive Maintenance of Sensible Electrical Installations Used in Mines Susceptible to Explosive Atmospheres or Firedamp (CdF and INERIS)

Some installations are indispensable for safety and remain without human supervision over several shifts. This is the case for high voltage supply stations, exhaust water pumping stations, ventilation, automatized processes. The detection of the degradation of some components in these installations is difficult prior to failure, an incident, and in the case of a fire.

Charbonnages de France (CdF, previous HBL, WP 2) worked in conjunction with INERIS (WP 8) in this programme. The first step was the collection of relevant installations considered necessary for the safety of underground workings from the end of the production area until the closure of mines and when then become abandoned.
A selection of suitable installations, followed by an analysis regarding the prediction of this equipment potential failure, was made.

The following tasks were achieved:

- Carrying out technical studies on selected equipments and their modifications to meet CdF specifications.
- Making necessary modifications in order to get a M1 or M2 certificate and to use selected equipments likely to be subjected to an atmosphere contaminated with methane.
- ATEX Certification M2 for the selected underground video camera.

The equipment had been modified and was being tested. The certification has been completed for the following equipment:

- The infrared camera (modified by the manufacturer to meet INERIS requirements)
- The video camera for deep pit inspection (modified, especially added an external lightning)
- The monitoring system for surveillance of mine atmosphere.

The infrared camera and the video camera will furthermore use for examining the shafts and boreholes in France. Only the monitoring system MX62 is not in use since the closure of the CdF-mines in Lorraine. Prospectively the developed devices could also used in other mines of all European countries if the tasks and the conditions are comparable.

1.3 WP 3 - Improved Gas Capture and Climate Control within High Performance Workings (DMT)

Primarily, the following areas were considered:

- Investigation of the influence of fugitive air on the face climate
- Measurements of and calculations pertaining to the face coolers

Task 3.1 - Investigation of the heat and gas flow in relation to face position
Within the framework of the research project, two workings were investigated as regarded their climates. The main aim of the measurements was to understand the influence of fugitive air on climate conditions, especially in the last third of the face. Between 20 and 30 % of the air directed to the face passes through the goaf in the area close to the face and picks up significant heat there. The inflow in the open face area can be influenced by the face positioning.

The workings were run during the measurements in the normal operating manner. In one of the workings the centre of the face was advanced approx. 10 m in front of the main drive. The measurements determined that the other influencing factors, such as heat emission from the equipment, heat emission

from the rock and the run-of-mine coal, and the influence on the fugitive air of the slope of the roof in the goaf mask the influence of the air inflow, dependent on the face position.

Task 3.2 - Investigation of heat flow reduction by means of climate window configuration
Climate measurements in a working with very high rock temperatures showed that climate windows could significantly contribute to reducing the climatic load in the end area of the face. Subsequent calculations for the working examined determined that without a climate window in the end area of the face, an effective temperature increase of 1.4 K would take place.

Task 3.3 - Formulation of a method of calculation to predict the heat reduction as a result of a certain face position and/or a specific climate window configuration.
The climate influence of the face position is covered by multitude of other climate effects which can't be regularly planned. Therefore a calculation method to for the climate prediction as a result of certain face position will not lead to any success.

Task 3.4 - Improvement of the infrastructure and the capacity of heat exchangers specifically in thin seams
With knowledge of the thermodynamic context, an adequate programme of testing was conducted to determine the dry and moist heat transfer coefficients for face coolers SK 3 and SK 4. These coefficients are among the basic data required for calculating the cooling capacity, dependent on the air and water inlet conditions. With the aid of the coefficients thus determined and the DMT heat exchanger calculation programme, a very good correlation was achieved between the cooling capacity measured and that which was calculated.

Task 3.5 - Investigation of the potential reduction of the coolant volume stream at the face
The calculation programme for face coolers gives the preconditions for reliable planning and calculations for optimised cold water distribution for the face cooling. Various models of parallel and series circuits for the water coolers were examined and it was determined that, with 4 coolers operating, a difference of 64 % could occur between the best and the worst cooling capacity values.

The characteristic values for face coolers determined during the SAFETECH research project improve calculating the cooling capacity. By calculating it in advance, suitable water circuiting of the coolers as well as an optimum fitting position can be achieved.

The results of the investigation of the heat and gas flow in relation to face position and the further development of the calculation programme allow the mine ventilation engineers of deep mining companies to optimize the cooling system and/or the cooling capacity of mining operations. With an improved cooling system mining workers reach better working conditions.

1.4 WP 4 - Climate Studies of Arduous Working and Emergency Activities (MRSL)

Task 4.1 - Literature Review
The objective concerned a scoping and industry review with specific consideration given to the challenges posed in respect of the development of laterally extended working in deeper mines. Consideration was given to how perturbations in the mining process and design changes to mine environment control measures could affect local mine climate and introduce short-term but excessive work temperatures.

Task 4.2 - Development of Improved Monitoring and Visualisation Techniques
The specific task objectives were to investigate:

- The scope for the introduction of a hand held, intrinsically safe, instrument to measure Basic Effective Temperature (BET) and provide real time information.
- The use of computational fluid dynamics modelling software as a visualisation aid for heat loading in mine atmospheres.

The scope of has been taken to a point where the design study indicates the feasibility of a self-contained BET monitoring instrument. The work has demonstrated the validity and usefulness of computational fluid dynamics modelling and the harnessing of the visualisation of the results for emergency, escape and rescue purposes. Temperature stratification has been shown to be a potential problem and has highlighted the requirement for a hand held BET monitor.

Task 4.3 - Development of an Integrated Thermal Risk Assessment Methodology
The specific task objectives were:

- Development of an Environmental Chamber which provides a range of simulated and controlled conditions of heat and humidity.
- Subsequent Environmental Chamber trials to develop a methodology for heat tolerance screening, and to examine core body temperatures of rescue and other workers in exact replication of mine temperature conditions.
- Development of a risk management methodology which integrates information and data from each task of this work package.

The Environmental Chamber was designed, constructed, successfully commissioned and became fully operational. A wide range of trials were carried out and provided data for the development of a heat management procedure in terms of rescue work and heat tolerance screening methodology. The environment chamber, which was developed to facilitate the research carried out within WP4, is situated at mines rescue station at Rawdon, UK. A second unit is under construction at the new mines rescue station at Kellingley, UK.

Task 4.4 - Practical Measures to Improve Emergency Intervention in High Heat Stress Conditions
The specific task objectives consisted of:

- Potential solutions to breathing apparatus face visor condensation problems
- Use of personal microclimate cooling jackets
- Development of rehydration systems
- Transport systems in terms of emergency and rescue.

The problem of breathing apparatus face visor condensation has been resolved with cost effective and readily available surfactant liquids. The wearing of microclimate cooling jackets was found to be of limited value for rescue workers. The most important part of this task was the successful development of a rehydration methodology for use with breathing apparatus. The results showed that core body temperature can be stabilised by this method. A realistic modification to underground transport systems, for use in rescue and refuge situations, has been described.

Task 4.5 - Presenting and Publishing of Results
The **results of the research** have been utilised to provide:

- Improved health and safety of persons involved in rescue and emergency operations.
- Provide personnel risk profile by heat tolerance screening to potentially prevent health problems or safety failures of persons subjected to high heat stress conditions.
- Newly developed training courses for optimum safe working methodology and understanding of the problems associated with high heat and humidity in the mining industry and other industries where high heat stress conditions are evident including, paper, fire fighting, police, rubber/plastics, food and drinks.

1.5 WP 5 and WP 7 - Reduction of Costs Associated to Gas Control in Caving Faces (AITEMIN and SA HVL / HUNOSA)

This project part of SAFETECH developed a methodology for methane caption in the previous cracking area, located before coal exploitations. The advantages of this method is to increase the security conditions underground, especially in coal 'ragging-off' operations, because after having drained part of the methane, the potential for sudden emissions of gas will be decreased.

Task 5.1 Seam characterisation
The objective was to characterise the coal seams within the chosen areas of the mines to be researched. On determination of a suitable seam, samples of coal were taken and analysed mainly for methane content, desorption speed and ash content. The parameters to be studied were defined and present in both SA HVL (Hullera Vasco-Leonesa) and HUNOSA mines.

Task 5.2 Self-combustion characterisation
The objective was to obtain samples from the coal seam where the project was to be developed and determine in the laboratory, among others, the Index of self-combustion risk, Flash point and Flammability. The task was not realized in SA HVL due to the change of mine. In HUNOSA it was not realized because the area selected to carry out the project was not historically subject to the self combustion phenomena and was not determined as necessary to study those characteristics.

Task 5.3 Methodology design
The main objective was to develop a methodology to capture and measure the methane extracted from a coal massif during the seam development. The main results were:

1. Definition of a basic methodology for a methane drainage installation including an optimum drilling pattern, extraction pipe network, measurement modules, pumping equipment gas and diffusion area design; and
2. The design of an automated process control system with advanced measuring capability.

Task 5.4 Methane capture boreholes
This objective was concerned with the development of a battery of methane caption drillings prior to layer exploitation. These drillings had to be adequately sealed up to the point where the exploitation influence is noticed. In relation to the boreholes, it was necessary to apply the following parameters: a) Control of the gas pressure, b) Methane concentration control, and c) Control of the temperature and other gases. In the SA HVL facilities a drilling campaign has been designed in the Santa Lucía Group, especially in the foot wall drift in the 6th level and in the main crosscut in the same plant. Within HUNOSA, two areas were identified to develop the investigation. During the research several drilling tasks have been achieved in the two exploitations.

Task 5.5 Environmental parameters
The main objective of this task was the control of environmental variables in the zone of caption and validation of the methodology. The following parameters, among others, to be analysed included: induced pressure control in the drillings, aspiration flood control and aspired gas concentration control. As a result in SA HVL, a real test in open air was developed to check the caption installation. In HUNOSA, work continued until the end of the project to monitor variables in both exploitations.

Task 5.6 Development capture
As an interrelation with the previous task, the main objective was to develop a methane borehole capture system, with the pressure and flood characteristics adapted to the production. The available technologies in the industry were used. Different and successive modifications were made in the design and as a consequence of the results obtained, the characteristics of wall stress or the massif pressure for each exploitation were determined. The main results were: (1) The conjunction of three factors; knowing the values that characterize the coal seams, planning adequately the exploitations of the mine, and applying forced aspiration in the drillings, will produce a high degasification in the coal seams before being ex-

tracted. (2) The wall stress is a fundamental variable and (3) the exploitation sequence in HUNOSA pit has been changed.

Task 5.7 Industrial application
An estimation of the energy value and utilisation potential of the available methane has been made. It is also considered that improvements in mine safety will be established.

The results of the research are summarised:

- A basic methodology has been defined, as well as the general principles of the methane caption installation: caption drillings, driving pipes, measurement modules, aspiration pump and captured gas dilution area.
- The automatic control of all the process has also been designed through an advanced measurement station, with enough treatment capacity and using the flood, methane, pressure sensors and others referred to the parameters to be measured.

The mentioned methodology is a necessary proceeding for any other mine belonging to SA HVL, HUNOSA or other mining companies, using the sublevel caving method.

1.6 WP 6 Improved Gas Capture, Control and Management within High Performance Workings (UNIV NOTTINGHAM)

Task 6.1 - Geotechnical Modelling
The objective of this task was to establish geotechnical relationships between the caving characteristics of retreat longwall workings and the release and control of methane gas. The major achievements and deliverables of this work package were: (1) the development and application of a Coal Measure Classification (CMC) system applicable to UK coal measures; and (2) the development of field validated 2D and 3D geomechanical models to represent the geomechanical deformation of strata due to the presence of active UK retreat longwall workings.

Task 6.2 - Stress Permeability of Coal Measure Rocks
The main aim of this task was to determine the stress-permeability relationship for representative UK Coal Measure Rocks. The results of a laboratory test programme was able to deliver a permeability data base of both intact and fractured UK Coal Measure Rocks

Task 6.3 - Fracture Flow Modelling
The two objectives of this task were:

- to identify the bulk volume permeability of the rock mass surrounding a longwall working panel created by the location and geometry of the stress patterns and fractures created within the rock mass due to mining; and
- to identify improved tailgate support methods with which to maintain the life and capture efficiency of the drainage boreholes.

The project was able to produce the following deliverables:

(a) a methodology to produce 2D maps across working longwall goafs, showing the preferential fracture planes, and fractured rock permeability that allow gas to migrate from adjacent coal seams to the mine workings;
(b) a novel mechanical model to predict the stability of a borehole subjected to the varying shear force applied by the collapsing strata to the borehole as the face retreats; (3)the field validated models confirmed that the application of roof support in the tailgate road immediately behind the face line greatly influenced the longevity and capture efficiency of the methane boreholes in this area.

Task 6.4 - Gas Drainage Trials

The two objectives of this task were,

- to identify improved drainage stand pipe length and orientation and spacing to maintain/improve the life and capture efficiency, and
- to develop combined geotechnical and environmental planning and operational guidelines to improve the life and the capture efficiency of methane drainage systems on rapid retreat longwall panels.

The methodology adopted to achieve the set objectives produced the following deliverables:

(c) The geotechnical modelling studies confirmed that the length and inclination of the boreholes employed at each of the mines studied were confirmed to intersect the predicted predominant interconnected fracture flow paths formed above and below the workings as the face retreats. This finding recommends the use of the geotechnical modelling strategy detailed above to be used in the planning and design of the methane drainage systems of future longwall panels;

(d) The probing surveys conducted on boreholes to the rear of the face line confirmed that the use of immediate roof support in the form of props and cribs greatly increased the longevity of the effective drainage of the boreholes;

(e) For the selection of case study mines studied, it was concluded that the optimum spacing distance between adjacent boreholes was dependent of the strength of the roof in the tailgate. It was concluded that the use of additional roof support could therefore allow for both the increased spacing of boreholes as well as increasing the effective drainage life of each borehole.

Dissemination of Results

The development of the coupled geotechnical and environmental assessment tool allow the mine ventilation engineers of deep mining companies (UK Coal, Tower Colliery, in South Wales, or others) to improve the optimal orientation and spacing of the boreholes and hence maximise the capture of the strata gas released from the identified major coal bearing horizons above and below the seam being mined. The improved efficiency of capture achieved by such optimised gas drainage systems will consequently lead to improved control of strata gas entering the mine airways. This will principally improve the safe operation of these workings, as well as maintain coal production, and ensure the maximum extraction and utilisation of strata gas for energy generation.

Publications resulting from this research study

Lowndes I S, Whittles D N, Kingman S, Jobling S and Yates C, 2006, The influence of geotechnical factors on gas flow experienced on a UK longwall coal mine panel, International Journal of Rock Mechanics and Mining Sciences, 43, pp 369-387

Whittles D N, Lowndes I S, Kingman S W, Yates C and Jobling S, 2006, Geomechanical factors that affect the installation and long term stability of a surface goaf well within a deep UK coal mine, Archives of Mining Sciences,51, 2, pp 197-23

Whittles D N, Reddish D J and Lowndes I S, 2007, the development of a coal measure classification (CMC) and its use for prediction of geomechanical parameters, International Journal of Rock Mechanics and Mining Sciences, (Article in Press)

Whittles D N, Lowndes I S, Kingman S W, Yates C and Jobling S, 2007, The stability of cross measure gas drainage boreholes, International Journal of Coal Geology (Article in Press)

2 SCIENTIFIC AND TECHNICAL DESCRIPTION OF THE RESULTS

2.1 WP 1 - Highly sensitive system for measuring and analysing gas (DSK)

Introduction

Project Objectives

Background: The critical volume ratio of CO development underground at which a fire must be assumed is around 10 l CO/min. With air volume currents of 2,000 m³/min and more the increased concentration of CO in the air current is merely 2 ppm. However, most underground mines already have a higher air mass, with a trend towards higher volume currents.
The gas measuring equipment at present being used guarantees a reliable detection limit of +/- 3 ppm CO. This makes it more difficult to reliably detect fire underground at an early stage.

Objectives: Hence, the aim of the DSK-project contribution is to develop a highly sensitive system for measuring and analysing gas, one that is capable of remedying the deficiencies mentioned. This involves not only a more sensitive instrument for measuring the concentration of CO, but also an alternative method for the early detection of fire and for determining the composition of the air underground.
The subject matter of the project is the development of a reasonably priced and reliable system for monitoring the underground workings and the composition of the mine air on a wide-scale basis.

Delayed start of research activities based on the subcontract works

The subcontract between DSK and the University des Saarlandes (UoS) was signed on July 2005. The subcontractor work was absolutely essential to establish a basis for the research project of DSK. Due to the delayed start of the subcontract works, the special attention in the remaining time had to be put on the basic research of the University of Saarland in combination with underground tests. Thus the tasks 1 to 5 had to be adapted in regard to the time and content wise to the new basic conditions. The adjustments are displayed in the programme bar charts.

Before this delayed start of the subcontract works, studies of literature were the main activity in WP1.

Based on the technical appendix of the subcontract between Deutsche Steinkohle AG (DSK) and Saarland University, Laboratory for Measurement Technology (UdS-LMT) WP1 was further subdivided in the following packages:

1. Under-ground experimental phase
   - 1.1. Reference measurements with FTIR spectrometer
   - 1.2. Selection of an appropriate set of gas sensors
   - 1.3. Hard- and software design of the test system
   - 1.4. Realization of the test system
   - 1.5. Under-ground field tests
2. Realization of an under-ground prototype fire detection system
   - 2.1. Interpretation of field test data
   - 2.2. Adaptation of the test system hardware
   - 2.3. Design and implementation of software algorithms
   - 2.4. Synthesis of the hardware and software prototype systems
3. Under-ground test of the realized prototype system
4. Design of a fire detection system for permanent use under-ground

2.1.1 Task 1.1 - *Preliminary study* – (Under-ground) experimental phase

Under-Ground Atmosphere

Based on an extensive literature survey target components for early fire detection could be identified. These are changing CO concentrations as a general indicator for emerging fires (production of CO > 10 l/min) and especially ethene (C_2H_4) for early fire detection. The emerging of ethene indicates that the fire source has a temperature higher than 150°C and by observing the ratio conc(CO) / conc(C_2H_4) the change-over from oxidation to fire can be detected. Former investigations of DSK [1, 2] show that in case that this ratio exceeds a value of 100 a fire alarm should be given.

In normal mining operation the CO concentration is in the range of a few ppm in open airflow. Considering the above ratio, ethene concentrations lower than 100 ppb need to be detected. This fact places tight requirements on the selection of suitable measurement equipment almost only leaving semiconductor gas sensors to take when being urged to fulfill price conditions at the same time. Especially in mining, the background methane concentration in the order of up to 1% and the strongly varying relative humidity additionally introduce cross-sensitivities which have been investigated carefully.

In order to be able to carry out measurements under-ground, a pressure-resistant housing fulfilling the required safety criteria has been designed and its realization has been organized. Figure 1 shows the design and the realization. This design also accounts for the parallel operation of the FTIR spectrometer with up to 3 of the test or prototype systems and their remote control and measurement data read-out from an above-ground control room. This remote control possibility is based on a fiber-optic connection and a device converting the optical signal to a standard Ethernet connection. This Ethernet connection, in turn, is now converted into a RS232-bus signal by means of a special serial server (for controlling the spectrometer) and into standard USB signals using an Ethernet/USB-server. Since the spectrometer requires specific operating conditions in terms of ambient and gas sample temperature and needs frequent calibration, the aerated housing was equipped with a water cooling system, a gas probe cooler and, in addition, a remote-controlled case containing the calibration gases.
However, in the course of this project, the FTIR spectrometer turned out to be not suitable for measurements under-ground. As a result, under-ground atmosphere samples have been collected and analyzed above-ground. The results confirmed above investigations about the atmosphere and are in conformity with samples evaluated by gas chromatography.

Figure 2 shows the resulting gas concentration profile developed to simulate relevant situations in the under-ground atmosphere and taking into account for lab measurements and their interpretation within the whole project [3]. These lab measurements have been carried out in order to characterize the used sensors with regard to the application thus accelerating the actual under-ground field test and its interpretation.

In table 1 [3] all relevant situations have been outlined and assigned to the situations non-fire, warning and fire as tested in the profile shown in Figure 2. Besides relative humidity and methane concentration change (these compounds have always been present as background), hydrogen and NO_2 have been tested as additional interfering components.

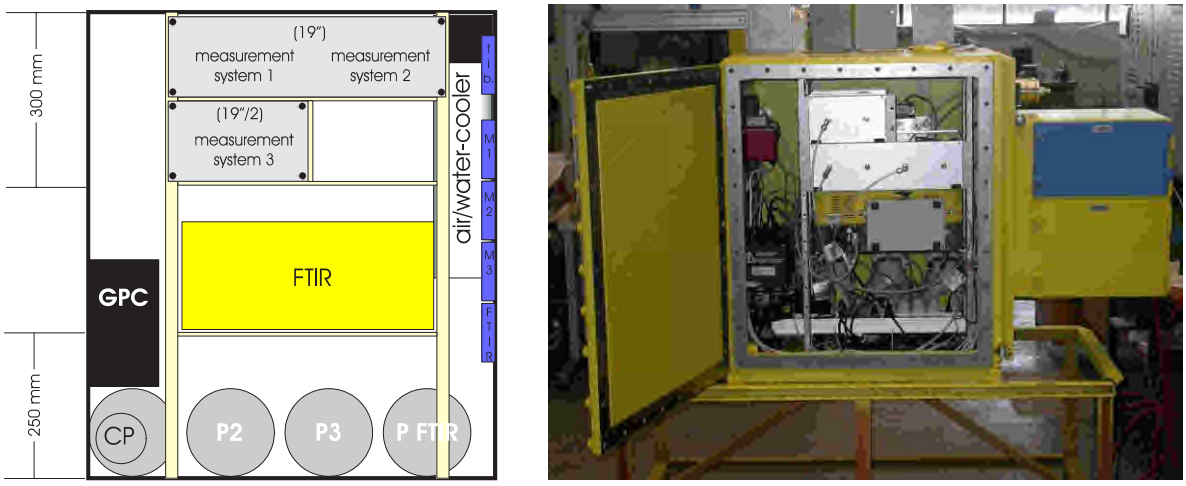


Figure 1: Front view of the designed housing (left) and the realization (right).

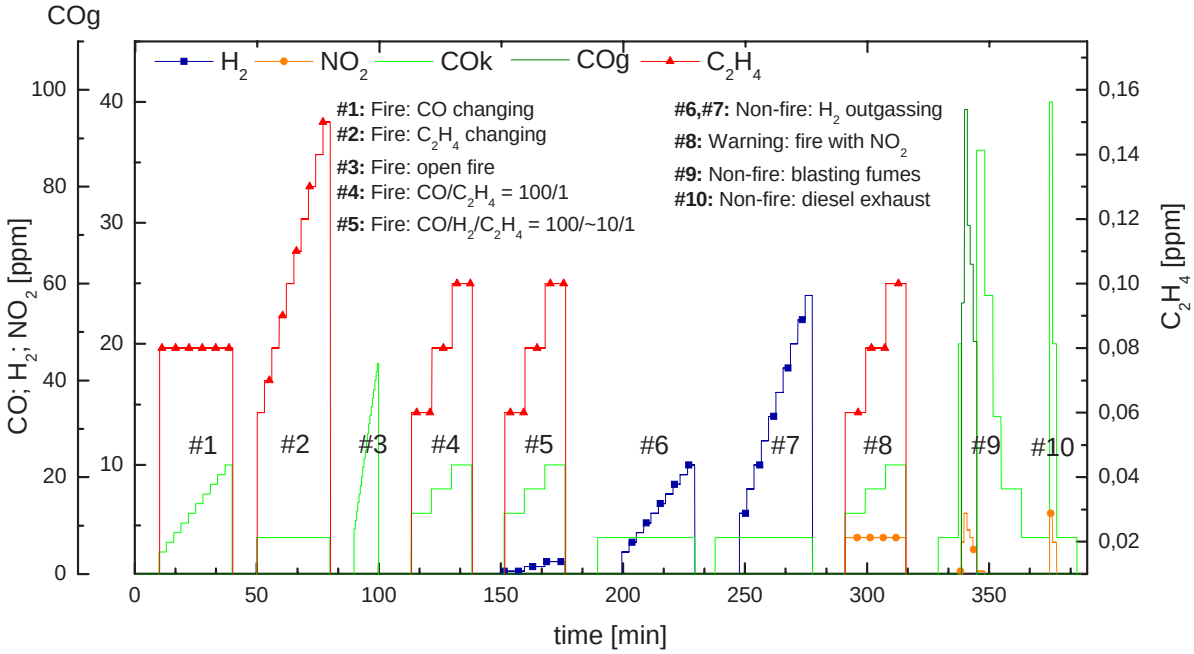


Figure 2: Concentration profile simulating relevant under-ground situations always taken into account methane and r. H. in differing concentrations as background (not shown). As perturbation compounds, NO_2 and hydrogen have been considered. Especially for simulating blasting fumes, a higher CO concentration (COg) has been used. COk denotes the smaller concentration range.

Situation		Label		Profile
Fire/Alarm	Open fire	1	Exponential CO increase	#3
	Smouldering fire	2	$CO/C_2H_4 = 100/1$	#4
		3	$CO/C_2H_4 \approx 100/1$	---
		4	$CO/H_2/C_2H_4 = 100/\approx 10/1$	#5
		5	CO and C_2H_4 increase	#2
		6	CO increase @ C_2H_4 = const.	#1
	Fire with pertubation	7	Pertubation gases, e. g. NO_x	#8
	Inertization	8	Fire or non-fire @ reduced O_2	---
Non-Fire/Non-Alarm	Changing atmosphere conditions	9	changing CO atmosphere	---
		10	CH_4 in the range of 0-1 %	altogether
		11	Changing relative humidity	altogether
	Blasting fumes	12	Fast CO, NO_x increase followed by a small decay, approx. 30 min duration	#9
	Diesel exhaust	13	Instantaneous CO, NO_x – increase/decay	#10
	Battery charging station	14	Hydrogen outgassing	#6, #7
	Water adsorption on coal	15	Hydrogen emission, r. H. change	#6, #7
	Under-ground work	16	Outgassing of among others solvents, adhesives, strength agents	---

Table 1: Relevant under-ground atmospheres classified in fire and non-fire situations [3]

Sensors

Due to their robustness and their well-establishment on the market (especially in fire detectors in coal-fired power plants) sensors from the German company UST have been selected. This company offers several sensor types including broad-band sensors as well as sensors selectively responding almost only to a specific target component. The investigated types are 1330, 2330, 3330 and 7330 [4]. By means of the gas mixing system (GMS) the sensors have been exposed to different gases and compounds (all parts of the overall profile of Figure 2) while investigating their reaction on different temperature levels (static temperature investigations). These static measurements provide first details for optimum detection temperatures. Nevertheless, primarily to increase the selectivity, but also for robustness, reliability and stability reasons, not only single temperature levels but rather temperature cycles have to be applied to the devices [5]. Since the performance of the sensor set to be chosen also depends on the applied temperature cycle (T-cycle), first, investigations and optimizations regarding the cycle itself have been done. The result of this optimization is presented in Figure 3. This cycle not only accounts for the target scenario (CO and ethene to be discriminated against a high methane background) but also for reproducibility aspects (i.e. controlled T-ramps instead of temperature step changes) and for stability purposes (significantly improved by the high-temperature step at the end of the 65-sec-cycle).

Based on this specific T-cycle, the selection of the sensor UST-1330 and UST-3330 sensor devices has been verified by discriminating between $CO+CH_4$, CH_4 and $CO+CH_4$+ethene (Figure 2, profile #4). Apart from that, the sensors UST-2330 and UST-7330 have also been assessed using this T-cycle but did not show results better than that obtained from the above-mentioned devices and have thus been excluded from the use within the project.

Hardware test system

The application of the specific T-cycle and the investigation of sensor reactions regarding to the profiles outlined in Table1 have mainly been carried out based on our lab hardware [6] with little adaptation. These measurements show that a high dynamic range of the sensor resistance acquisition is required and that the adaptation flexibility of this platform was not sufficient for under-ground use. Thus, a former design of a new platform has been further developed [7] and first printed circuit board (PCB) realizations have been tested. These tests reveal several aspects to be optimized and a new version of the hardware had to be built up.

This was the reason why the field tests planned to be carried out in a relatively early stage of the project work has been replaced by lab measurements (mainly consisting of the profile of Figure 2) which have been interpreted in the next section. Due to this small change in project organization, an efficient use of time and results relying on lab measurements and being transferable to the field test phase could be obtained.

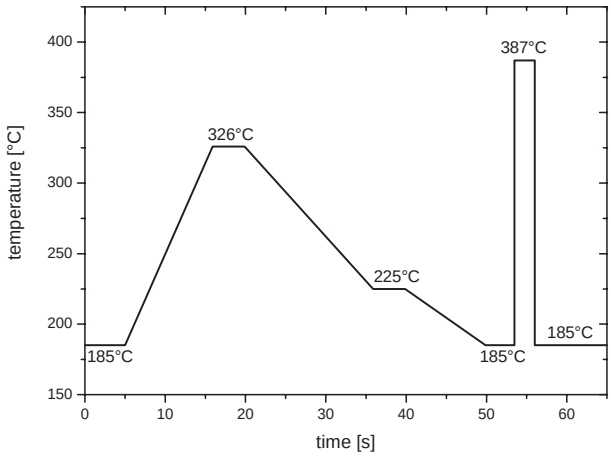


Figure 3: T-cycle used for early fire detection. This cycle incorporates optimum temperature levels and also accounts for detection reproducibility by using T-ramps and stability by applying a high temperature step.

2.1.2 Task 1.2 - *Development and production* - Realization of an under-ground prototype fire detection system

The development process for evaluating semiconductor gas sensors within the project work is schematically shown in Figure 4. This procedure basically corresponds to the subsequent steps in the system design of Figure 6 but pays more attention to the actual working steps and optimization loops which have to be run through in order to find optimum components and data processing techniques.

Signal Processing

Referring to Figure 4 the following tasks have been carried out based on the raw data obtained from the hardware platform:

- Data normalization: the raw data have been pre-processed in order to reduce drift phenomena of the sensors.
- Feature extraction: features like for instance slopes or mean values of the respective temperature levels or ramps have been extracted from the normalized data and serve as input for the decision process (see Figure 5 for details)
- Linear discriminant analysis (LDA) has been performed to assess the potential of the respective features for situation classification.

According to [8, 9] the linear discriminant analysis is a signal processing method based on feature extraction and relying on data grouping (for instance: all signals measured in air are grouped to the group "air"). By means of an eigenvalue calculation procedure the minimum number of rows in the feature vector is determined (i.e. the features for best group discrimination are chosen) thus performing a dimension reduction of the later result (i.e. discriminant) functions. An algorithm then maximizes the distance between the data of different groups (for instance between air and CH4) and at the same time minimizes the distance between the members of the same group yielding the LDA coefficients vector. In a last step the values of the discriminant functions are calculated as the result of multiplying the features vector with the LDA coefficients vector. This leads to a dimension-dependent number of discriminant functions being visualized in LDA plots up to dimension three. Higher dimension plots are available by applying projections, typically into several (2-dim) planes. The different positions in these 2-dim plots can then be assigned to the respective groups defined at the beginning.

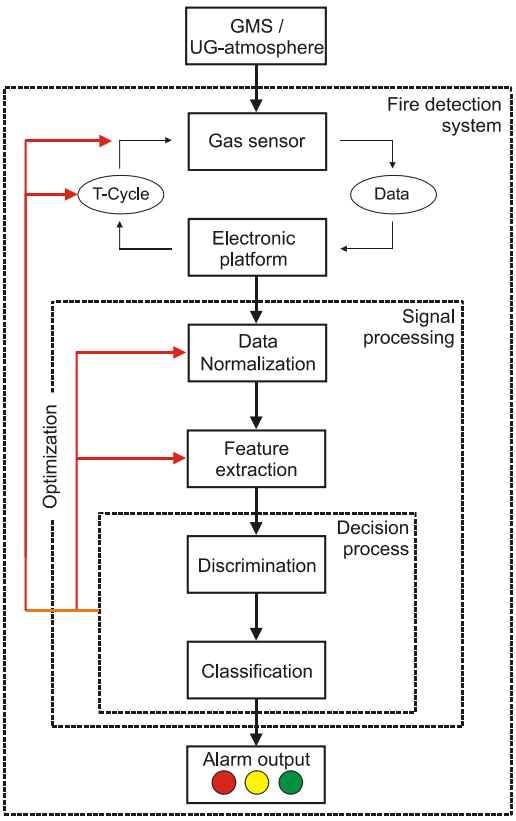


Figure 4: Work flow in the course of the project especially paying attention to optimization loops. This procedure is independent of the fact whether the atmosphere is generated by a gas mixing system (GMS) or provided by reality (under-ground atmosphere).

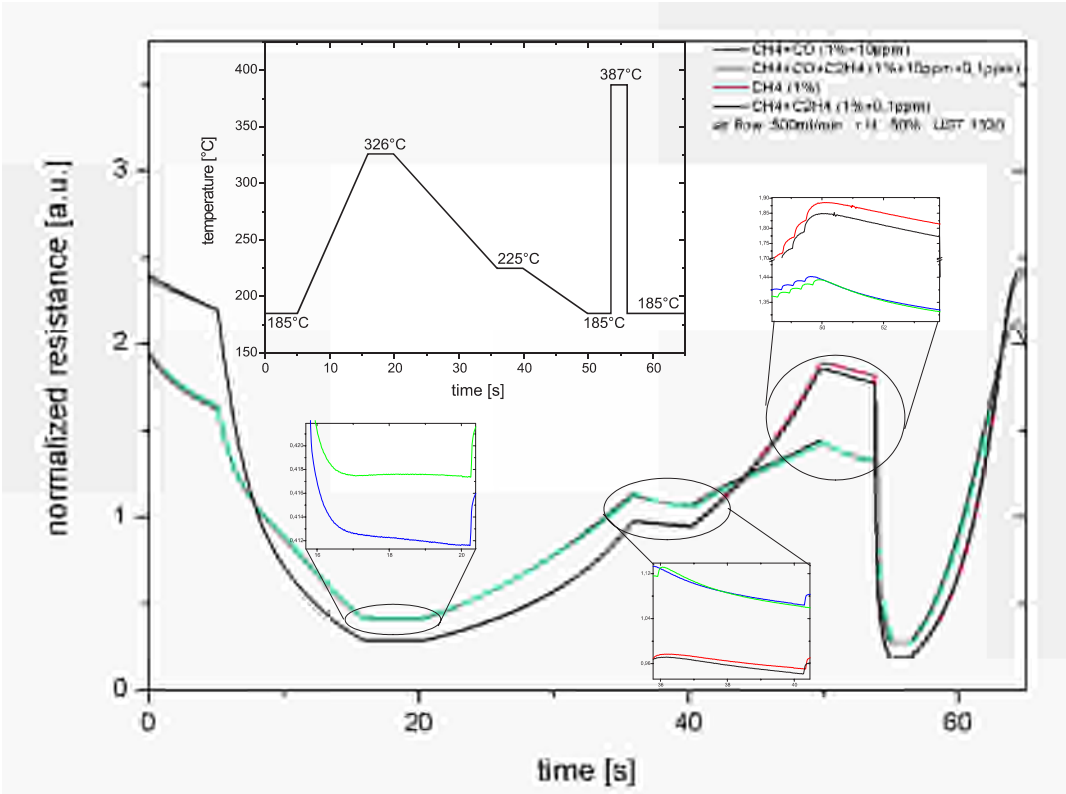


Figure 5: From the normalized raw data (divided by the cycle resistance mean value) features like mean values and especially slopes are extracted.

Finding this coefficients vector optimally suited to classify the considered situation is done during the LDA training or calibration phase. In the verification phase, i.e. application, the coefficients vector gained from calibration is multiplied with the feature vector extracted from the measured data and the result is depicted in a typically 2-dim plot with the respective positions being characteristic for the respective groups. Among others, the position of the respective groups is detected by an algorithm in order to perform the classification of the compound under consideration.

A separation or discrimination between fire and non-fire situation in presence of all considered interfering gases using only one single LDA step is not possible. Therefore, after evaluating the lab measurements with the profile from Figure 2 and iteratively passing through the signal processing optimization loops of Figure 4, the hierarchical discrimination algorithm in Figure 6 could be identified. This algorithm provides the discrimination between alarm and no-alarm taking into account perturbing situation. Hierarchical discrimination means that based on the result of a precedent discrimination step the coefficient vector for carrying out the current discrimination is selected. The discrimination algorithm established within the project (Figure 6, right) consists of up to 4 steps: At first, the relative humidity level is determined by just one dedicated coefficient vector. This result leads to an appropriate selection of coefficients which are multiplied with the feature vector to determine the methane concentration. Based on the methane concentration detected, the coefficient vector for the next step is defined. The following (3rd) LDA qualifies the presence of interfering gases and a set of coefficients dependent on the presence of hydrogen (group 1) on the one hand and NO_2 or “no interference” (group 2) on the other hand is selected. The abbreviation “X” denotes either “ethene” or “nothing” and therefore carries the information for the alarm / no alarm decision through the discrimination process. In the last step (4th LDA), the decision between alarm and no-alarm situation is done where the situation CH_4, CO, ethene and NO_2 is classified as warning since we saw that diesel exhaust gases could also contain ethene.

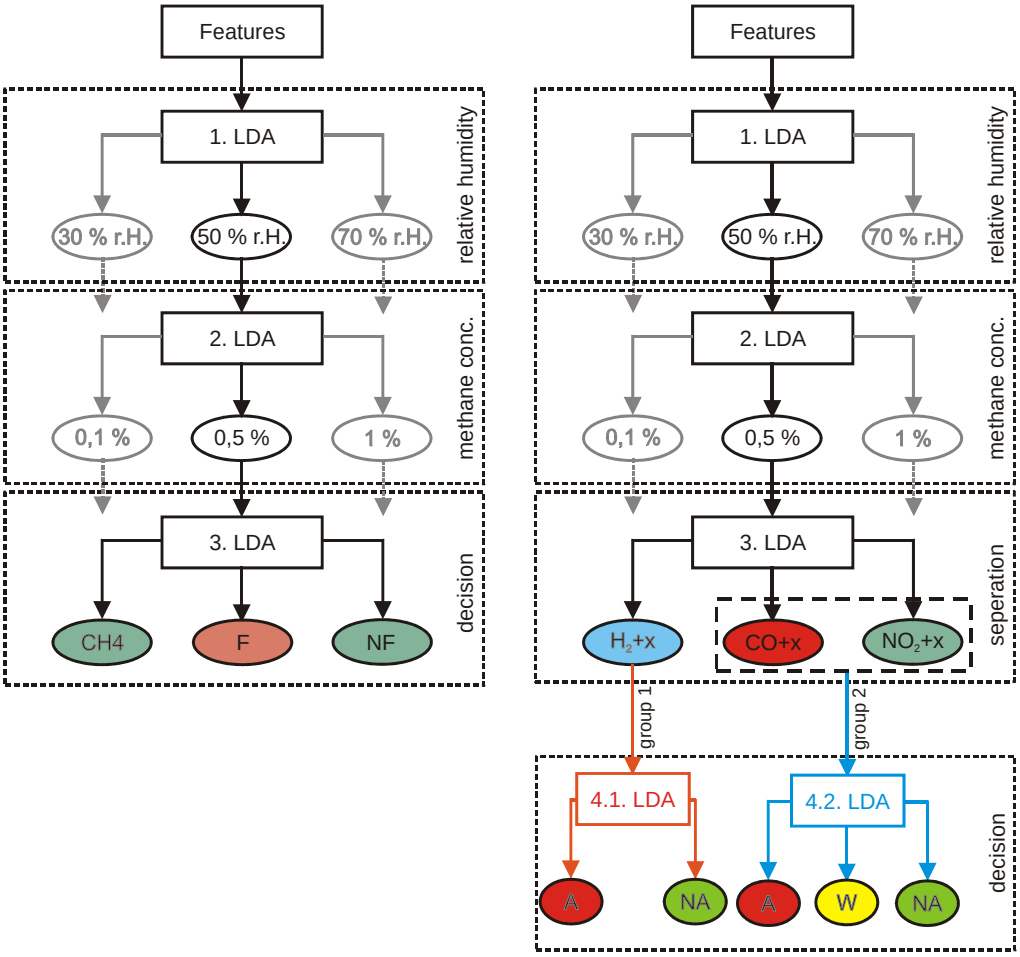


Figure 6: Hierarchical Discrimination: left without investigation of perturbing gases; right: consideration of H_2 and NO_2 as disturbing components. X denotes either "ethene concentration" or "no additional components"

In order to illustrate the above discrimination algorithm, intermediate results of the lab verification phase have been depicted in Figures 7 to 10. The exemplary decision process is carried out for the following situation: 50% r. H.; 0.5% CH_4, 10 to 25 ppm H_2, no NO_2 (situations #6 and #7 from Figure 2). First, the r. H. level is determined. It is not necessary to do this value-continuously but rather in coarsely quantized groups (e.g. 30%, 50%, 70%) as shown in Figure 7. The above compound would yield points in the 50% area of Figure 7 (left). Then, the corresponding coefficient vector (the vector for 50% r. H.) is selected and applied to the features extracted from the above compound. This would produce points in the 0.5% area of Figure 7 (right). Again, a corresponding coefficient vector is selected for the determination whether hydrogen is present (then the projection would yield points in the area of group 1) or not (what is equivalent to a point position in the area of group 2). This is shown in Figure 8. The exemplary compound would produce a point in group 1 leading to the final classification step of Figure 9 where dependent on the presence of ethene, an alarm is given or not.

In the case that NO_2 is present instead of H_2, the point calculated based on raw data feature extraction would fall in group 2 leading – after application of the corresponding coefficient vector – to the LDA plot of Figure 10. As explained above, the situation warning ($CO + NO_2 + C_2H_4$) has been additionally considered.

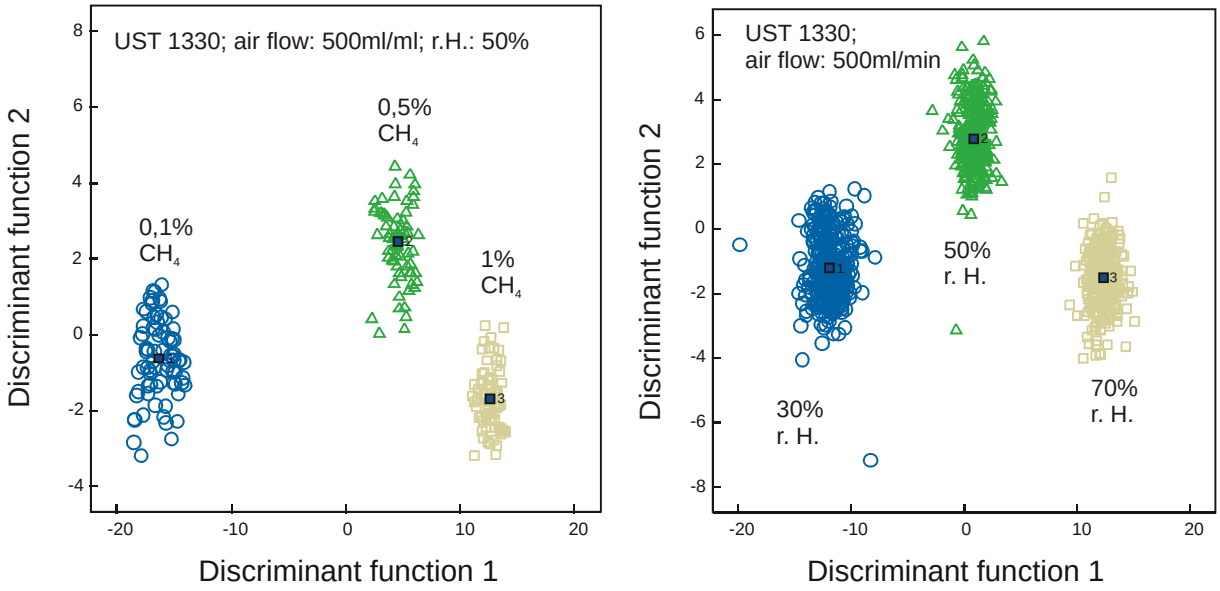


Figure 7: Left: discrimination possibility between different methane concentration groups, also in order to select the appropriate coefficient vector for the next step
Right: discrimination possibility between different r. H. groups sufficiently well quantized for coefficient vector selection.

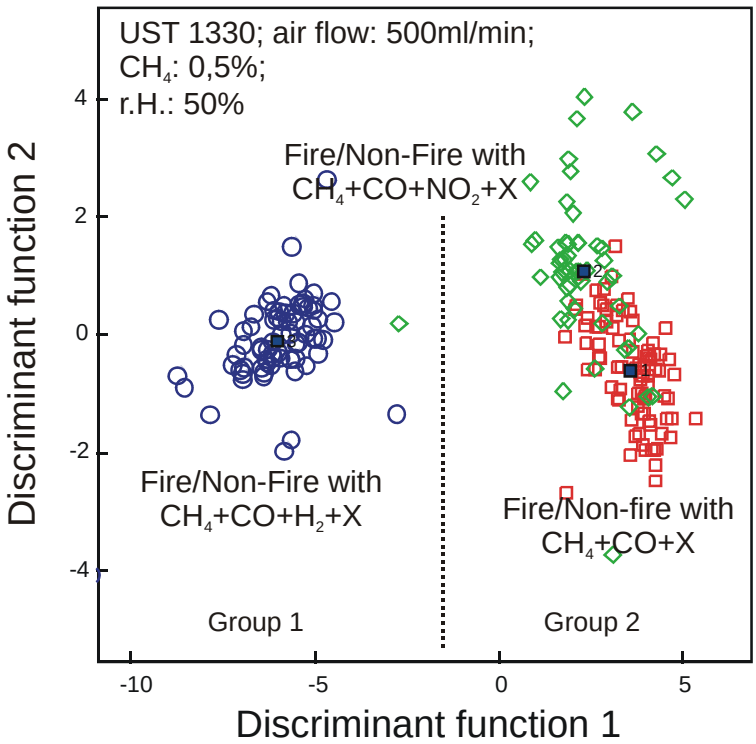


Figure 8: Discrimination of group 1 (CH_4 + CO + X plus hydrogen) and group 2 (CH_4 + CO + X and CH_4 + CO + X plus NO_2). Based on the group membership of the compound, an optimum coefficient vector is selected for further discrimination.

The combination of H_2 and NO_2 present as interfering components has not been tested with the algorithm so far.

Since the detection of the relative humidity level is of great importance, the final design provides a possibility to interface a combined humidity and temperature sensor. Also, the methane and / or the CO

concentration could be obtained from the already installed measurement equipment under-ground revealing the systems ability either to be operated stand-alone or to be easily incorporated in the existing infrastructure.

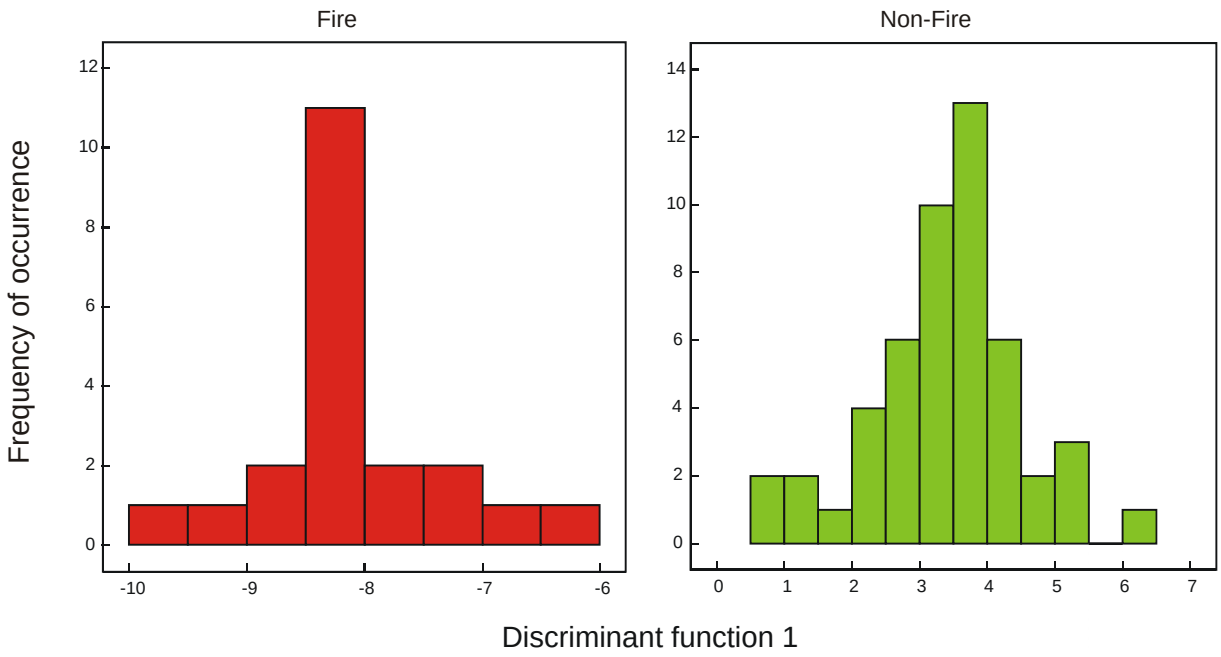


Figure 9: Final discrimination obtained by an LDA: Fire situation (CH_4 + CO + H_2 + ethene) and non-fire situation (CH_4 + CO + H_2) could be discriminated. This LDA is based on coefficients selected due to "group 1" membership in the precedent step (visualized in Figure 10)

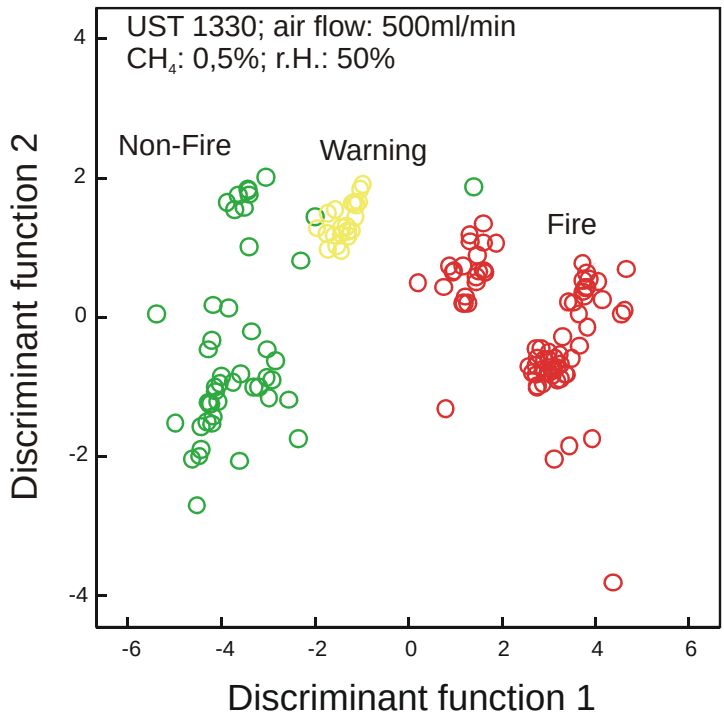


Figure 10: Final discrimination between fire (CH_4 + CO + ethene), non-fire (CH_4 + CO + NO_2) and warning (CH_4 + CO + NO_2 + ethene) based on "group 2" membership of the precedent step.

Hardware platform

Based on a predecessor platform [5] with insufficient measurement range and adaptation flexibility a more powerful and versatile hardware platform has been designed [7]. First lab measurements based on the design realization showed the need for optimization and for adaptation to the specific application. As a result of adaptations and optimizations within the project time, the development system PuMaH (Pulse-width-modulated Measuring and Heating Unit) has been realized which largely relies on the use of digital, i.e. pulse-width-modulated (PWM), signals for temperature control and sensor read-out (Figure 11) [10]. These signals are provided by a high-performance MCU allowing a high amount of flexibility due to easily applicable software changes. This software variability is complemented by the system hardware functionality being realized with a significantly reduced amount of peripheral electronics, especially regarding analog components. So, the almost fully software-based adaptation of the overall system ensures a high versatility. The key features of the platform are a temperature control accuracy of better than 1°C being established within a few milli-seconds (for use with microstructured gas sensors), a sample rate of 1 kHz and a large measurement range of up to 26 bit. Another key feature of PuMaH is that the system automatically detects the sensor device to be handled and adapts to it in terms of heater and sensor resistance range, applied heater voltage and control speed.

The above system has been realized on PCBs, with an individual board for each sensor. The prototype detection system consists of up to four semiconductor gas sensors which are provided with individual T-cycles and which can be read-out in parallel. A PCB and the overall system set-up are shown in Figure 12. This overall system together with a dedicated measurement chamber incorporated in the same housing has been installed under-ground. For the application of under-ground fire detection, especially the feature of a large measurement range is of great importance. Another key feature for the field tests is the possibility of incorporating a sensor for relative humidity and temperature. Lab measurements confirmed that the current electronics yield the same raw data (sensor resistances) as the older platform used as test measurement system, i.e. the calibration data obtained during the lab tests can be used as reference for the field test data.

A dedicated graphical user interface for the sensor raw data acquisition has been developed accounting for the communication infrastructure (USB, Ethernet, Fiber-optics) and allowing to store the incoming data to dedicated files. These data files serve as raw data input for the subsequent signal processing, i.e. test data are evaluated off-line.

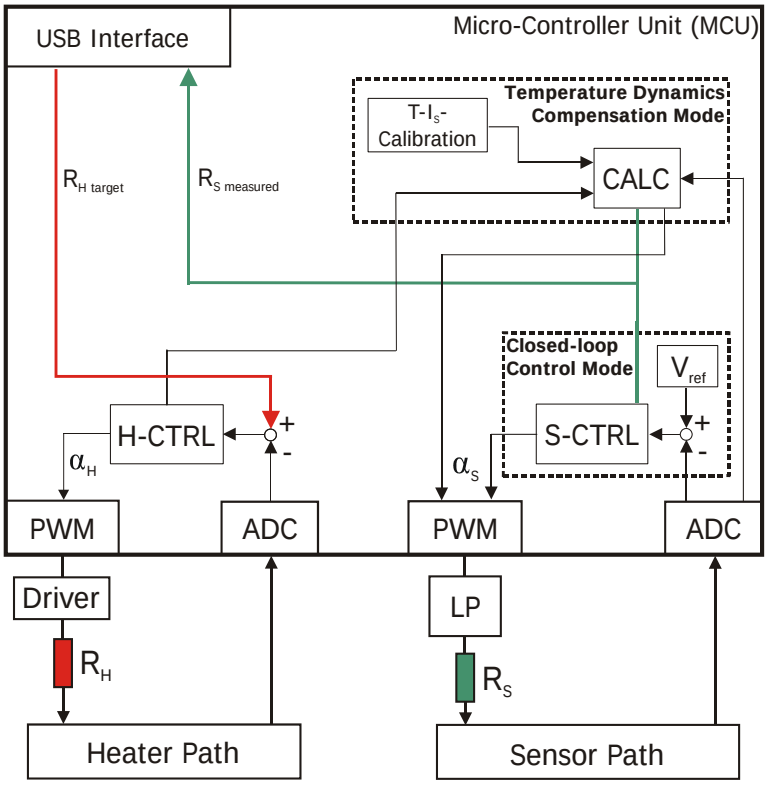


Figure 11: Final Design of the hardware platform used within the project. This design is based on results from lab measurements and subsequent optimization and adaptation steps.

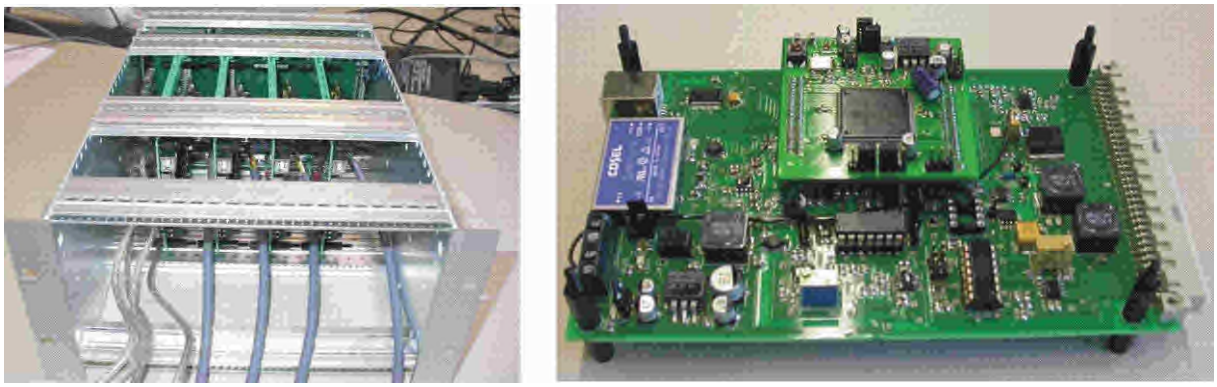


Figure 12: Realized overall system (left) consisting of a to four sensor read-out circuits on a PCB basis (right). Also, a sensor for relative humidity detection can be incorporated in the modular system realization.

2.1.3 Task 1.3 - *Development and construction of electrical Equipment* - Under-ground test of the realized prototype system

The field tests and the test of the realized prototype system started in the last third of the last project period. A detection system has been installed in the housing and the data have been transferred to a PC above ground. Exemplary field test data are shown in Figure 13.

The upper depiction of Figure 13 shows the output of regularly installed measurement equipment which serves as a reference for evaluating the data of the prototype system. The detected CO peak clearly indicates a digging-induced signal that belongs to a non-alarm situation. This peak could also be acquired with the prototype system yielding a characteristic pattern in terms of slopes as shown in the

lower depiction of Figure 13. This figure confirmed that CO is present due to its reaction at lower temperatures or higher sensor resistances, respectively.

During the field test period, there was no alarm situation so that only patterns for a non-alarm situation could be acquired.

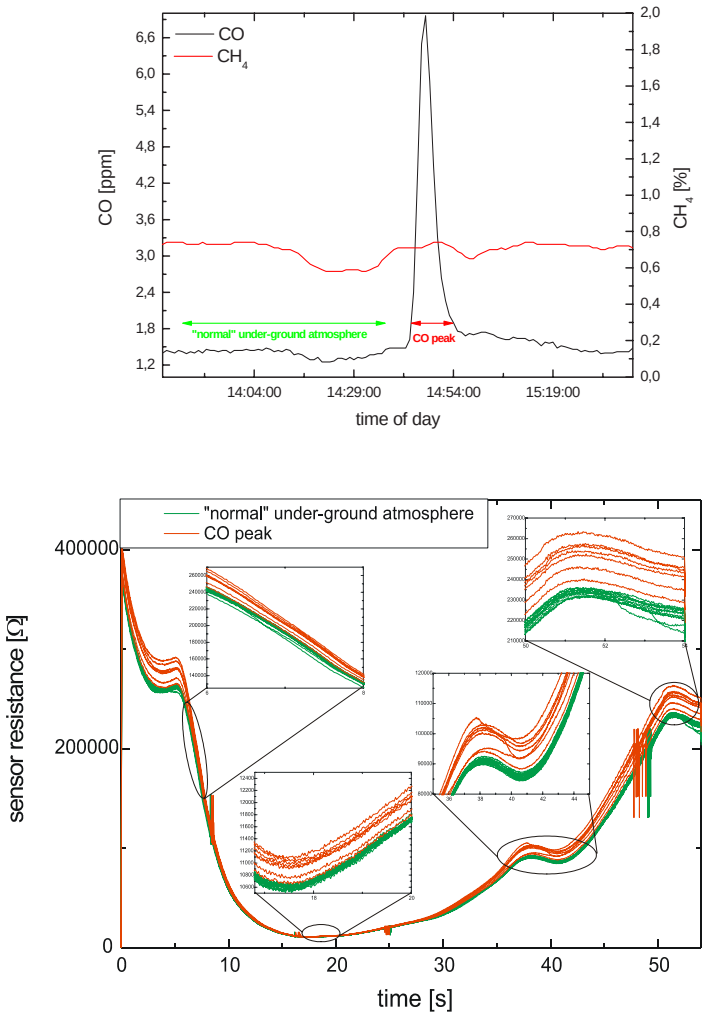


Figure 13: First results of field test data. Top: results from already installed CO measurement device. Bottom: Typical CO shape in the data acquired with the new detection system.

2.1.4 Task 1.4 – *Testing and certifying ATEX* - Design of a fire detection system for permanent use under-ground

Figure 14 shows a system design for early fire detection under-ground based on semiconductor gas sensors as a result of the project work. This design includes appropriate sensors (UST-1330 and UST-3330), a dedicated electronics being able to apply an application-specific temperature cycle to the semiconductor devices and to read out the gas dependent response profile and a signal processing block consisting of data pre-processing, feature extraction and classification.

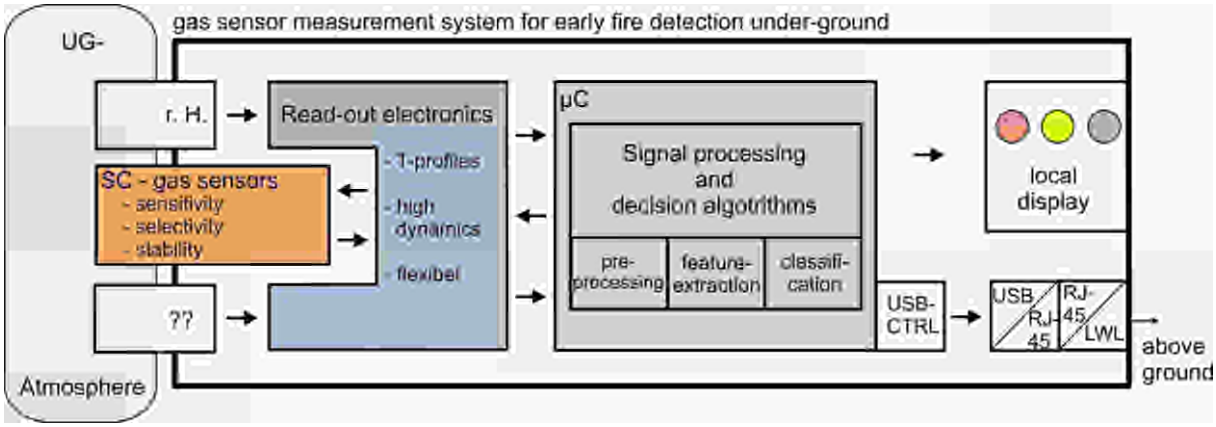


Figure 14: Design of the overall system

The current system provides the following performance which also needs to be guaranteed by a later fire detection system for permanent use under-ground:

- An internal sampling time of at least 25 ms and a measurement range of at least 16 bit in order to acquire the signal shapes appropriately
- A temperature set-point accuracy of better than 5°C
- An implementation of the above algorithm incorporating a multi-stage discrimination process
- A graphical user interface visualizing the measurement results in an appropriate manner
- A simple up-to-date data connection realized via USB

Regarding the above prototype system as a starting point, some aspects have to be optimized and some features have to be added:

- A sensor for relative humidity has to be incorporated in the hardware section.
- The possibility to consider already existing measurement signals or any additional data input (marked with "??" in Figure 14) should be provided.
- For a stand-alone system, a powerful processor has to be used or additionally incorporated in the system (a standard micro-controller will not be able to handle the calculations to be done).
- The data processing section of the system should provide a possibility to reduce the time needed for a calibration process. This could be achieved by finding a mathematical transformation mapping the LDA coefficients determined for one specific humidity level to LDA coefficients needed to carry out discrimination at other humidity levels.
- A local display indicating the classification of the current situation as alarm, warning, non-alarm.

Requirements for the design of an auditable fire detection system

The German test laboratory for mine ventilation, EXAM, made an assessment how the existing test standards which are currently applied to CO gas detection apparatus for use in hardcoal mines can be applied to such apparatus with new techniques of signal processing (evaluation of the signal of a virtual sensor array). Proposals for the extension and modification of the current European standards concerning the performance requirements and test methods for apparatus for the detection and measurement of toxic gases (EN 45544 series) were formulated. This EXAM-Report is added as Annex.

2.1.5 Task 1.5 – Underground Testing and Optimising

The research work for developing a highly sensitive system for measuring and analysing gas will continue and it is planned to test the prototypes in underground trial run. Because of the later beginning of the research work and different difficulties (duration of the certification the pressure-resistant housing and the under-ground tests of the measurements equipment) this task 1.5 could not be carried out in the SAFETECH research period.

Further measurements and developments

The remote calibration mechanism will be equipped with mixed gases in order to be able to adjust the sensors underground.
The university is developing a demonstrator which integrates the individual software packages (measuring, evaluation and visualisation) into a total package.
The overall system is a further development of the lab model for evaluation of the fire risk by means of continuous comparison with the field test data.

In a subsequent project, further research and development should be carried out regarding the construction of inherently safe measuring equipment with remote calibration and the software packages from the above demonstrator. To this end, it is a precondition that the measures described above can be realized, i. e. early and safe detection of coal mine fires is possible.

2.1.6 Conclusions

Advantages for the security of the workers

The use of special semiconductor sensors makes it already possible to measure concentrations of gases within the ppb range. This means that coal mine fires can be detected during the initial stages and suitable counteractive measures can be early introduced. Risk to employees by toxic carbon monoxide concentrations or explosive gas concentrations in the mine can therefore be minimised.

2.1.7 Exploitation and impact of the research results

Actual applications

DSK and the Saarland University have agreed that the prototypes which have been developed will continue to operate at least until the end of 2007. Three test measuring systems are operating at the underground measuring station of the Prosper Haniel coal mine, and an additional system is in use as a reference system in the laboratory of the university. Further field test data will be collected in order to verify the developed model to detect fire risk.

Dissemination of Results

The results achieved to date cannot be used by a third party, neither from the technical nor practical point of view. It is only possible to employ the prototype for recording the measurements underground with special permission. The overall software as a combination of the software packages for measurement, evaluation and visualisation is currently under development by the university (demonstrator). Currently, there is no interface between the measuring system and the computer at the central mine control station using the approved communication standard for coal mine air data.

References:

[1] Guntau, A., „Entwicklung und Erprobung neuer Verfahren zur Verhütung und Bekämpfung von Selbstentzündungsbränden im Steinkohlebergbau“, Abschlussbericht 01 HD 018, Bundesministerium für Forschung und Technologie, September 1994

[2] Fuchs, E., „Ein Beitrag zur Erhöhung der Sicherheit bei der Grubenbrandbekämpfung im Steinkohlebergbau durch die Entwicklung eines Brandgas-Analysesystems und neuartiger Explosionsdiagramme“, Pirrot Verlag & Druck, Saarbrücken, 1996

[3] Reimann, P., “Entwicklung von Auswertealgorithmen zur effizienten Branderkennung unter Tage”, Diploma Thesis, Saarland University, 2006.

[4] http://www.umweltsensortechnik.de/index2.htm, 03/2006

[5] Schütze, A., Gramm, A., Rühl, T., „Identification of organic solvents by a virtual multisensor system with hierarchical classification”, IEEE Sensors Journal, Vol. 4, No. 6, Dezember 2004

[6] Kammerer, T., Ankara, Z., Schütze, A., “GaSTON – a versatile platform for intelligent gas detection systems and its application for fast discrimination of fuel vapors,” Proc. Eurosensors XVII, Guimarães, Portugal, Sep. 22-24, 2003.

[7] Conrad, T., Hiry, P., Schütze, A., „PuMaH – a temperature control and resistance read-out system for microstructured gas sensors based on PWM signals“, IEEE Sensors Conference 2005, Irvine, California, USA, Oct. 31 - Nov. 3, 2005.

[8] R. Gutierrez-Osuna, „Pattern Analysis for Machine Olfaction: A Review”, IEEE Sens. J. 2 (3) (2002), pp. 189-202.

[9] K. Backhaus, B. Erichson, W. Plinke, R. Weiber, “Multivariate Analysemethoden”, Springer, Berlin, 2006

[10] Conrad, T., Fricke, T., Reimann, P., Schütze, A., „A versatile platform for the efficient development of gas detection systems based on automatic calibration”, Proc. Eurosensors XX, Gothenburg, Sweden, Sep. 16-19, 2006

2.2 WP 2 and WP 8 - Preventive Maintenance of Sensible Electrical Installations Used in Mines Susceptible to Explosive Atmospheres or Firedamp (CdF and INERIS)

Introduction

Some installations are indispensable for safety and remain without human supervision over several shifts. This is the case for high voltage supply stations, exhaust water pumping stations, ventilation, automatized processes. The detection of the degradation of some components in these installations is difficult prior to failure, an incident, and in the case of a fire.

The principal objective of this study was to find devices usable in firedamp mine able to anticipate the failures. Some of these devices still exist but cannot actually be used in firedamp mines.

Charbonnages de France (CdF, previous HBL, WP 2) worked with INERIS (WP 8) to investigate the best technical solution applicable in mines subjected to potential explosive atmospheres. The research methodology comprised of the following:

- early detection of degradation and overheating of electrical components
- air analyse
- infrared thermograph
- camera and acoustic monitoring

and was completed together with preventive maintenance and periodic inspections.

The project work of CdF and INERIS had been finished mainly in 2005 how planned and in this case earlier than the others work programmes.

2.2.1 Task 2.1 - Listing of mine installations

Charbonnages de France listed every mine underground installation, equipment and infrastructure, essential for safety, from the end of exploitation to the definitive closure, and potentially after this closure. The most important items to be considered were:

- general air monitoring system with essentially measurement of methane, CO, and air velocity
- high voltage electrical installations
- dewatering stations
- open galleries and pits
- underground transportation

It was necessary for CDF to ensure the best level of safety with the minimum human means including areas that may be unmanned during several shifts. A problematic situation arose where some pits have been closed for several years and the associated difficulties with inspection potential under a shaft seal/closure. After consultation of the remaining responsible underground mines, the following installations were selected:

- high voltage distribution
- dewatering
- surveillance of mine atmosphere
- pit and gallery inspection

2.2.2 Task 2.2 - Foreseeing of possible failures

The intention with this component of the work was to ensure the highest level of safety with fewer employees and a continuous lost of internal competence. In this regard, the following areas were selected which may be considered as potential problem areas:

- With respect to every item of electrical equipment - How to prevent a future failure of a component leading to a heavy break down?
- In our remaining open galleries - How to detect easily a hot point?
- How to ensure efficient underground atmosphere surveillance with actual classical sensors used in industry, and with a possible external subcontracting?
- How to inspect a closed pit not fully filled?

Every solution has to be usable in gassy underground mines and if some equipment still existed on the surface of the mine, it was necessary to consider if there was a requirement to modify and certify it, in accordance with the mine regulations.

The choice of the solutions and the required modifications were proposed by Charbonnages de France (CdF, previous HBL). In the examination of the final specifications, the equipment evaluation and the final certification of each item of apparatus was the responsibility of INERIS.

2.2.3 Task 2.3 - Seeking for devices

2.2.3.1 Infrared Camera

This device is able to carry out temperature measurements at distance and to register them. By comparison with initial measurements, it is possible to detect a hot point with accurate temperature determination in an electrical installation or in underground coal walls. Several types of cameras were available on the market. The model considered the most useful for this project was the small and portable E4 produced by FLIR in Sweden. This device does not consume a very high level of energy and may perhaps be modified to an intrinsically safe device able to be certified M2 after examination by INERIS.

2.2.3.2 Video Camera for Deep Pit Inspection

In order to visit closed pits a camera was required which was able to:

- go through thin pipes : diameter 150 mm and more
- go down up to 1000 m
- provide illumination to 8 m
- inspect with an adjustable head
- register the whole distance.

A device was chosen which was produced by HYTEC in France. The camera was watertight to a depth of 1000 m, with infrared light capability and an external supply through an electric feeder at 300V.

For the purpose of this component of the project, CdF and INERIS carried out modifications to this device in order to obtain an M2 product rating in accordance with the ATEX directive 94/9/EC.

2.2.3.3 Monitoring System for Surveillance of Mine Atmosphere

The main objective of this component of the work was to use modern classical sensors using low energy, bounded to a standard modular and efficient monitoring system which still existed, in order to replace the complex actual CGA (general atmosphere surveillance system). The CGA is considered to be very expensive to maintain. Indeed, we have lost a lot of internal technical competence due to the continued closure of the mining industry and the actual system includes old sensors not in accordance with the ATEX 94/9/EC directive thus not allowing this equipment to be placed on the market. After consultation, a suitable item of equipment was found that was produced by OLDHAM (France). The apparatus consisted of the following:

- a monitoring system referenced MX 62, still used on surface and which could be installed without certification out of the mine ,
- a methane sensor low energy : only certified for group "ia" II C
- a carbon monoxide sensor : OLC … Z0 "ia" M1
- an air velocity sensor TROLEX TX 5923 "ia" M1

It was necessary to find suitable interfaces between the monitoring system and the sensors installed in the mine at less of 1600 m and then to build a whole underground system M1.

2.2.4 Task 2.4 - Testing of devices

2.2.4.1 Infrared Camera

Feasibility Study on an Infrared Camera

CdF (HBL), the partner of INERIS for this work package, required an infrared camera for contactless temperature measurements. After some investigations, a suitable certified product was not found but an uncertified infrared camera manufactured by a Swedish company was found to have potential. An investigative visit was made to the company with a view to examine the product for suitability.

It appeared that the current version of the camera was not directly certifiable. Certain parts of the camera had to be removed with the supply to the circuit being established with an external source together with the video exit and the internal circuit for charging the battery. The detector had to be encapsulated in accordance with standard EN50028 owing to the fact that manufacturer did not possess the drawings, and did not have the control of manufacture and the required power necessary to make the detector function at 2,5 W. The other parts of the apparatus can be considered in intrinsic safety in accordance with standard EN50020 in so far as:

- the capacity number is reduced,
- that each tension is protected using zener diodes assembled in element shunt,
- that the circuits "Backlight", "Detector" and "Logic" are separate from/to each other.
- The supply must have a limiting device of current with semiconductor of 3,3 W. The separation of the circuits "Backlight", "Detector" and "Logic" will have to be to realise by distances conform to table 4 of standard EN50020
- The power resulting from the supply will have to also be separate in each circuit via fuses
- A resistance will have to be placed in series with the inductance of 10µH, another with the memory battery present on the chart "Logic" and another with the motor.
- A limitation of the heating allowing the classification in temperature will have to be carried out via thermal fuses placed on the FBGA of the chart "Logic" and that of the detector.

Figure 1: FLIR Infrared Camera

Research of a Company Able to Modify an Infrared Camera

A company was identified that repairs the intrinsic safety products used in coal mines. This company is certified according the quality system called SaqrATEX. Such certification attests the competency in the field of ATEX products. The following modifications were made during the third period of the project:

- Remove some unnecessary modules as the charger
- Encapsulation of the Infra red detector in accordance with EN 50028
- Adding of some limiting circuits
- Increased distances between modules
- Adding of thermal fuses for temperature limitation.

At the completion of the modifications an assessment was made of the product in accordance with the directive 94/9/EC directive as a M2 product. The assessment of the conformity was made in accordance of the annex IX related to unit verification.

2.2.4.2 Video Camera

Pre-Feasibility Study on a Video Camera

Due to the closure of coal mines some mine shafts will be disused for the transfer of coal and workers but these shafts must be maintained for several requirements. To avoid surveillance of electrical cables in all the length of the pit by trained people, a video camera is required. CdF had acquired a video camera that works under the sea and is used by the oil industry but unfortunately this camera is not adapted for ATEX.

An initial meeting was held with the manufacturer in order to establish the required specifications. For the aspect of safety, an M1 product was required which conformed to the ATEX 94/9/EC directive and a special agreement given by the ministry of industry.

It was considered difficult to use directly a recognised standard such as EN 50014 and EN 50020 so it was proposed to the manufacturer, using two independent principles of protection against explosion. The manufacturer proposed a method of inertisation with nitrogen and depression with surveillance of the pressure.

Due to technical reasons, it was considered impossible to use batteries inside this camera, therefore an external supply was necessary. The electrical insulation and integrity of the electrical cable supply must be permanently checked and the Manufacturer and CdF investigated this problem.

Assessment of the Video Camera

A suitable video-camera was found and adaptations on a standard product were made in order to fulfil the protection principles imposed for M1 products by the 94/9/EC directive, which are:

- two independent systems of protection or,
- protection against explosion risk guaranteed even in the case of two rare faults.

Figure 2: HYTEC Camera and power supply/control unit

Figure 3: Inspection of an abandoned shaft

The manufacturer provided a video camera without a power supply inside the camera and the unit was protected against explosion potential by including the following parameters:

- One unique steel enclosure
- Inertisation with nitrogen
- The pressure inside the enclosure is lower than the atmospheric pressure
- If the pressure inside the enclosure increases, the external power supply is switched-off.

A co-axial single conductor cable connected to an external power supply located at the ground level supplies the camera. The conductor carries energy, video-signal and control/command information. Insulation between the inner conductor and the protective armour is permanently checked. If a fault occurs, power supply is switched-off.

During the assessment process it was indicated that there was a difficulty to have, without important modifications on the camera, the two fully independent systems of protection required by the ATEX 94/9/EC directive. For this reason, an M2 certificate was issued and it was decided to present the product to a special commission for the safety in mines in order to get a special agreement.

The DTR 65MPX camera is now certified as category M2 equipment under the certificate number: INERIS 04ATEX7016X. For using as such equipment even in presence of firedamp, French mining regulations required a ministerial approval called "arrêté HNS" (HNS means high level of safety). A report has been produced and presented with success, to a special ministerial commission. This report shows that the camera could be used above the permitted concentration of firedamp.

Addition of a Lighting Device

After functioning tests in a large pit were carried out, it appeared that the lighting capability was not sufficient and an extra set of lights to be included with the apparatus was requested. The manufacturer provided a new lightning device in an independent enclosure. This enclosure is depressurised in the same way as the camera. Assessments tests were carried out at the beginning of 2005 (at CdF Lorraine and DSK Saar) and an extension to the certificate was issued.

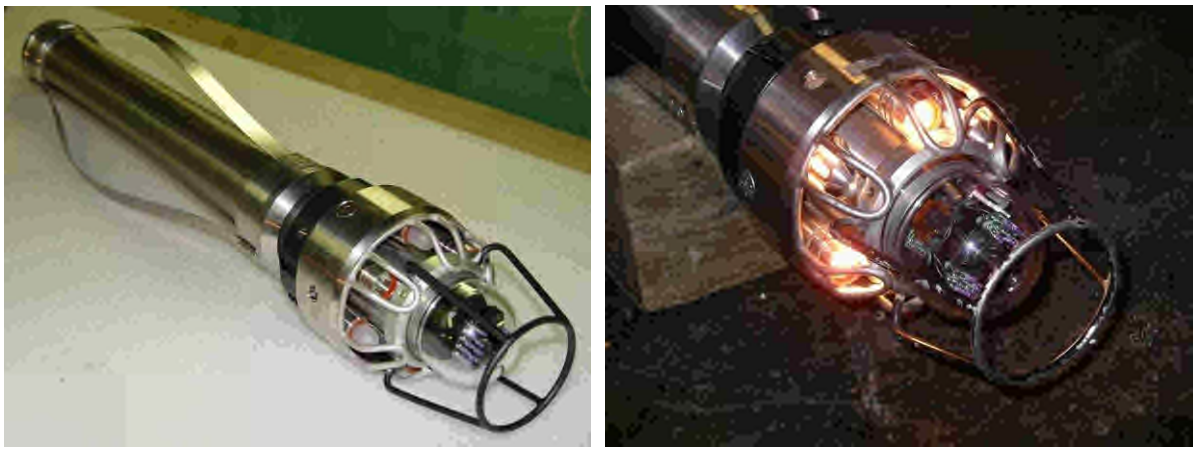


Figure 4: HYTEC Camera with extra set of lights

2.2.4.3 Monitoring System for Surveillance of Mine Atmosphere

System Pre-Study

According to the regulations, electrical equipment used underground must be protected against explosions (M2 according ATEX directive) and they shall be de-energised if an explosive atmosphere occurs. Some sensible M2 electrical apparatus are used in places where gas detectors will be absent after the closure of mines (actual sophisticated system will be removed). To avoid a M1 certification of these apparatus it is necessary to have a monitoring system for the mine atmosphere in question.

A feasibility study was carried out in order to use a standard monitoring system as currently used above ground. It was considered possible to use the equipment but the manufacturer had to find suitable interfaces between the monitoring system and gas detectors.

The feasibility study to use a standard monitoring system dedicated to surface was completed. The whole system was assessed as an “intrinsic safety system”. The system comprised of the following:

- A monitoring system MX62
- Limiting devices between sensor and monitoring
- Methane sensor CEX 300 certified “ia” M1
- Carbon monoxide sensor OLCT40 certified “ia” M1
- Air velocity sensor TX5923 “ia” M1

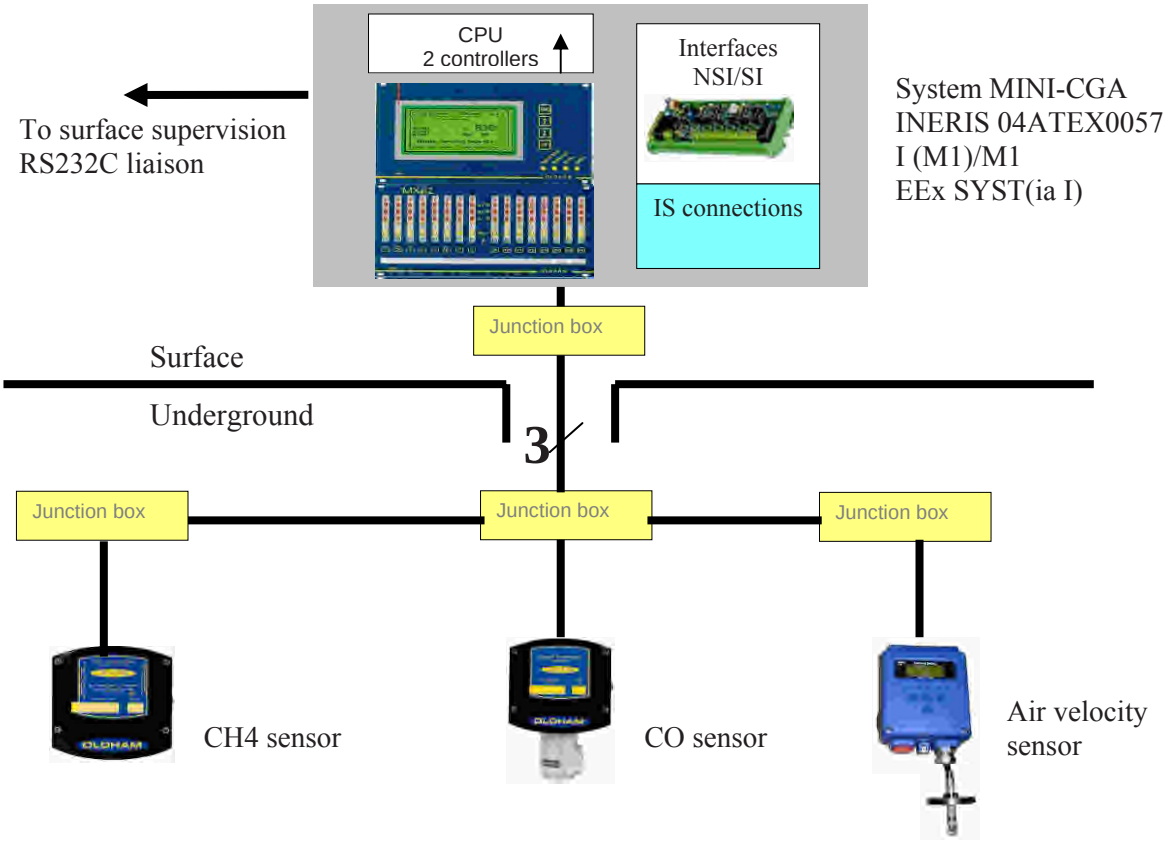


Figure 5: Monitoring system for atmosphere

- Measurement and alarm unit for 16 channels *(7 methane, 6 carbon monoxide, 3 air velocity)* configurable as required to max 64 channels.
- Measurement processing redundancy
- Direct link, network and loop

Figure 6: Monitoring System OLDHAM MX62

The system is described in a descriptive system document and the following principles are fulfilled:

- Each piece of equipment are safely connected to the monitoring system
- Intrinsic safety character of each piece of equipment shall be maintained
- Mixing of different circuits shall be avoided.

In supplement due to each item of equipment being intrinsically safe, the interconnection of the whole equipment had to design in accordance intrinsic safety system requirement for group I. Many intrinsic safety barriers existed to currently available equipment on the market and they were only designed to conform to group II requirements and the output parameters were not defined for Group I connection. Therefore, the output parameters for group I connection had been re-defined. This was especially the case for one of them, where a new intrinsic safety shunt limiter was designed which allowed the ability to connect more than 1600 m of cable.

All necessary documents have been written by CDF Lorraine and the system has been certified under the certificate number: INERIS 04ATEX0057. The category of this system is M1.

The system MX62-CEX300 has been also evaluated in accordance with performance standards of EN50054-58 series.

2.2.5 Conclusions

With the mentioned devices it is possible to anticipate an accident or incident and to avoid damage. It is possible to ensure the best level of safety with the fewer employees (means also unmanned during several shifts) and with the continuous lost of internal competence. A problematic situation is solved where some pits have been closed for several years and the associated difficulties with inspection potential under a shaft seal/closure.

2.2.6 Exploitation and impact of the research results

Actual application

Charbonnage de France (CdF, former HBL), the partner of INERIS for this work package, will liquidate on 31/12/07. After that the new company BRGM (Bureau de Recherche Géologique et Minière) will continue the work of CdF. A special department named DPSM (Département Prévention Sécurité Minière) will establish for observing the abandoned shafts. The infrared camera and the video camera will furthermore use for examining the shafts and boreholes in France (approximately 20 to 30 measurements a year). For example the video camera with the accessory units like the power winch or the monitoring and recording system is the sole system to observe abandoned shafts because the illumination has a range to 8 m and this until 1000 m depth.
Only the monitoring system MX62 is not in use since the closure of mine Vouters.

Technical and economical potential for the use of the results

The use of the existing video camera in Germany or other countries is occasionally possible. Moreover the developed and described devices as copies could also used in other mines of all European countries if the tasks and the conditions are comparable. The equipments are developed based on the demands of CdF and INERIS. The original manufacturers could recreate the devices in according to the ATEX directives.

2.3 WP 3 – Improved Gas Capture and Climate Control within High Performance Workings (DMT)

Introduction

For economic reasons, great efforts are being made in European coal mining to reduce production costs. Amongst other things, this is made possible by installation of more efficient workings with ever higher face output quantities. In German coal mining, this increase in output has only been possible through the constant development of the extraction and conveyance technology at ever longer faces. Due to the enlargement of the installed capacities and the higher rock temperatures at increasing depths, staying within the climatic thresholds becomes ever harder, especially in the final third of the face. Even with massive pre-cooling of the air before entering the face and subsequent cooling with face coolers, under extreme loads the limits of what can technically be realised and of climatically acceptable working conditions are reached. In order to be able to achieve better planning certainty for the face cooling system, extensive measurements on face coolers were conducted and cooler calculations made on the basis of these.

An as yet inadequately investigated influencing variable for the climatic loading of face air is the heat that reaches the open face area through the fugitive air from the abandoned workings. It is known that around 20 to 30 % of the air reaching the face does not pass through the open face area but through the goaf near the face. The heat absorption in the abandoned workings is significant, as the heat is transferred to the air over a very large surface area in the fracture zone. As the fugitive air can significantly contribute to the climatic load, investigations of this problem were conducted at workings within the framework of this research project. In these investigations, the influence of the face positioning and climate windows was to be central to the observations.

The work conducted by DMT can primarily be divided into two main work blocks. The first block concerned itself with the influence of fugitive air on the face climate. This block included investigations for the following tasks:

Task 3.1 - Investigation of heat and gas flow in relation to face positioning

Task 3.2 - Investigation of heat flow reduction by means of climate window configuration

Task 3.3 - Formulation of a method of calculation to predict the heat reduction as a result of a certain face positioning and/or a specific climate window configuration

The second block concerned itself with measurements of and calculations pertaining to face coolers. This area handled these tasks:

Task 3.4 - Improvement of the infrastructure and the capacity of heat exchangers specifically in thin seams

Task 3.5 - Investigation of the potential reduction of the coolant volume stream at the face

The working package 3 handles mainly with the climate situation underground. Working package 4 will attend to the ergonomic aspects on worker and mine rescue services.

2.3.1 Task 3.1 - Investigation of heat and gas flow in relation to face position

2.3.1.1 Calculations of the effects of climate windows and coal face positioning on the climate at the coal face

The following theoretical calculations show what effects the face positioning and the climate window have on the climate values in the face. The calculations were conducted for a plough operation with a bulk output of 6200 t/d at a rock temperature of 48 °C.

Climate windows and face positioning influence the proportional fugitive air stream through the abandoned workings. The following scenarios were considered:

- Straight face without climate window (1)
- Face centre advanced (so-called bulge positioning) without climate window (2)
- Straight face with climate window (3)

These scenarios were assigned different fugitive air streams, based on experience:

- (1) the first 250 m (in direction of air) 20 % and the last 50 m 5 %
- (2) the first 200 m 30 % and the last 100 m 10 %
- (3) the first 250 m 20 % and the last 50 m 35 %

The calculations were conducted with the DMT climate prediction programme. The results of these calculations are summarised in the following graph. The increase in specific enthalpy over the length of the face in the direction of the air is shown.

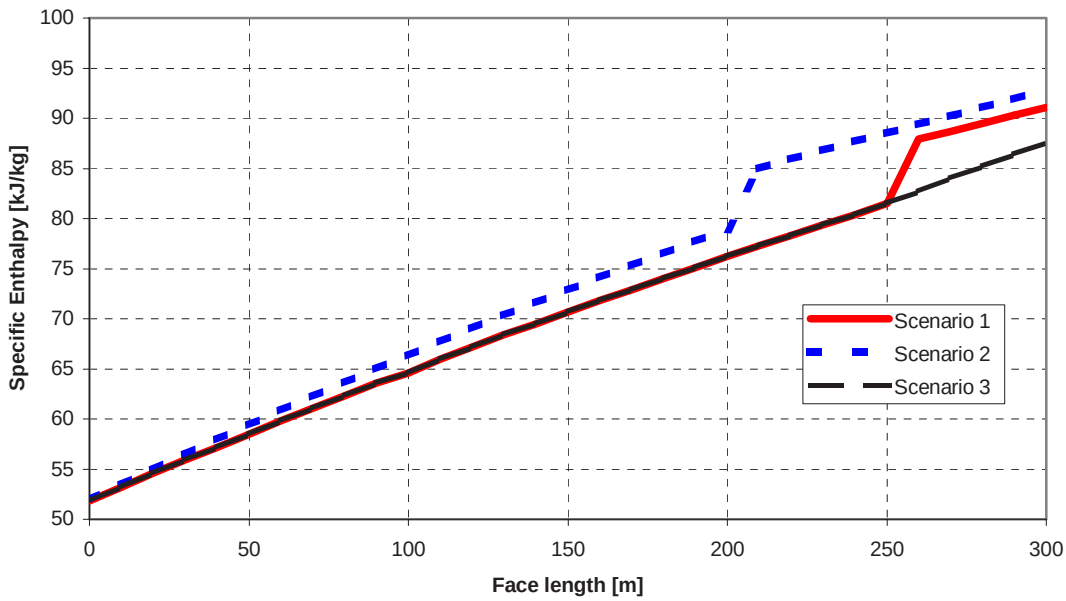


Figure 1: Specific enthalpy of the face air relative to face length

The curve clearly shows the increase in temperature at the face end typical for a straight face (scenario 1). This jump can be compensated for with a climate window (scenario 3). On closer examination of this curve, however, a rising tendency towards the face end becomes visible. This tendency becomes stronger as the fugitive air stream becomes larger and actual quantity of air available to the face becomes smaller. The heat emission from the electric equipment is independent of the ambient conditions. Uniform heat emission thus leads to greater heating effect as the volume of air becomes smaller. If climate windows are dimensioned too large, this may even result in a worsening of the climate conditions.

The curve in figure 1 for scenario 2 is likewise typical for faces with advanced centre. The final third of the face is mostly a physically very limited area with strong inflows from the abandoned workings. This leads to lasting worsening of the climate. To get from the climatic condition of scenario 2 (the highest climatic value) to the condition of scenario 3, a cooling capacity of approx. 350 kW would have to be installed, without ventilation measures. These theoretical considerations show what potential lies in the ventilation measures to be considered.

2.3.1.2 Investigations in workings

Investigations were conducted in two workings with differing climatic boundary conditions. Up to 16 measuring devices were used for the climate measurements. The power consumption was recorded with the workings' own measuring devices. The cooling capacities during the measuring period were recorded by manual measurements and with the aid of built-in measuring devices.

2.3.1.2.1 Description of the workings

The two workings investigated are schematically represented in the following figures 2 and 3. In both cases regeneration takes place in countercurrent flow via the coal haulage road. The significant data on the two workings are compared in table 1, below.

	Working 1	**Working 2**
Face length	401 m	299 m
Thickness including dirt band	1.60 m	2.38 m
Depth	App. 970 m	App. 1450 m
Original rock temperature	40.3 °C	64.2 °C
Average air flow to the face via tail gate	28.3 m^3/s	38.8 m^3/s
Average air flow via coal haulage road	20.0 m^3/s	16.0 m^3/s
Nominal installed capacity for the shearer		2 x 420 kW
Nominal installed capacity for the plough	2 x 800 kW	
Nominal installed capacity for the face conveyor	2 x 1000 kW	3 x 800 kW
Nominal installed capacity of the crusher and conveyers in the roadways	App. 1300 kW	App. 1000 kW

Table 1: Main characteristics of the working

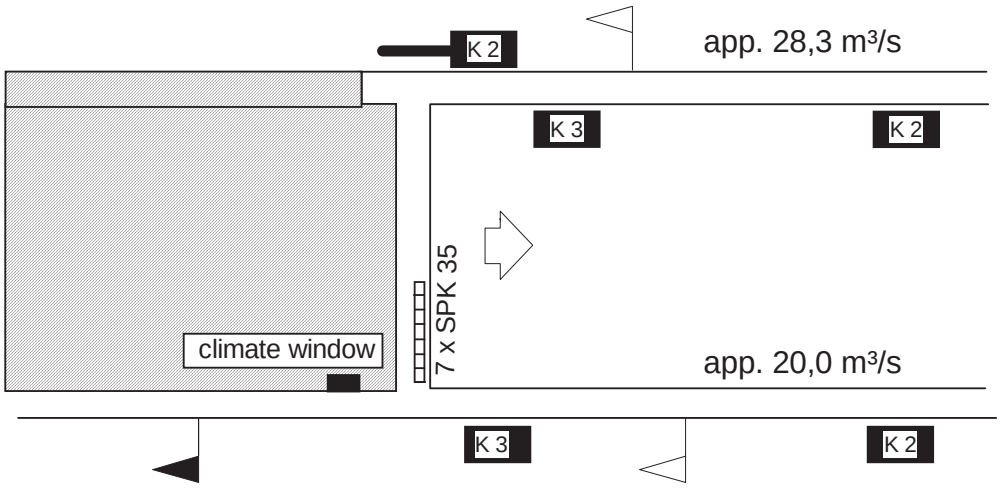


Figure 2: Schematic representation of the first working

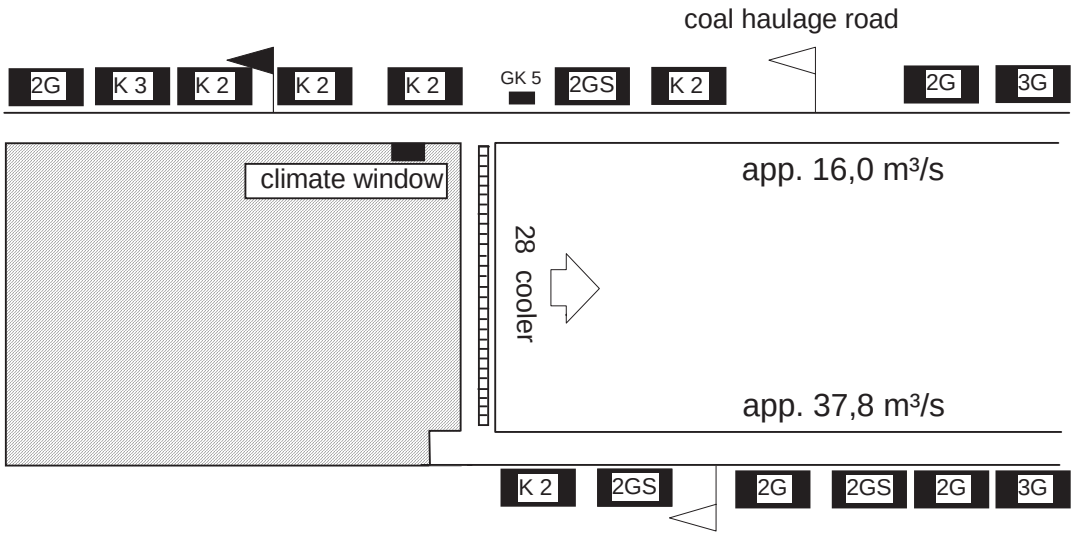


Figure 3: Schematic representation of the second working

The comparison of the significant working data shows that very different climatic boundary conditions predominate in the two workings. The high rock temperatures in the second working especially required significant expenditure on air conditioning. While in the first working 5 roadway coolers and 7 face coolers were sufficient to control the climate conditions, the second working had 15 roadway coolers and 28 face coolers installed. For the first working, the effective cooling capacity during extraction in the roadways totalled approx. 660 kW and in the face approx. 200 kW. For the second working, an effective cooling capacity in excess of 5000 kW was recorded during the measuring period. At the same time up to 1000 kW was used in the face.

The heat absorption for the face in the second working is shown in figure 4. It is shown that despite a very high face cooler capacity during extraction, more than 3000 kW of heat is transferred during extraction. Despite the high output capacity, the heat absorption in the face of the first working did not exceed 1500 kW.

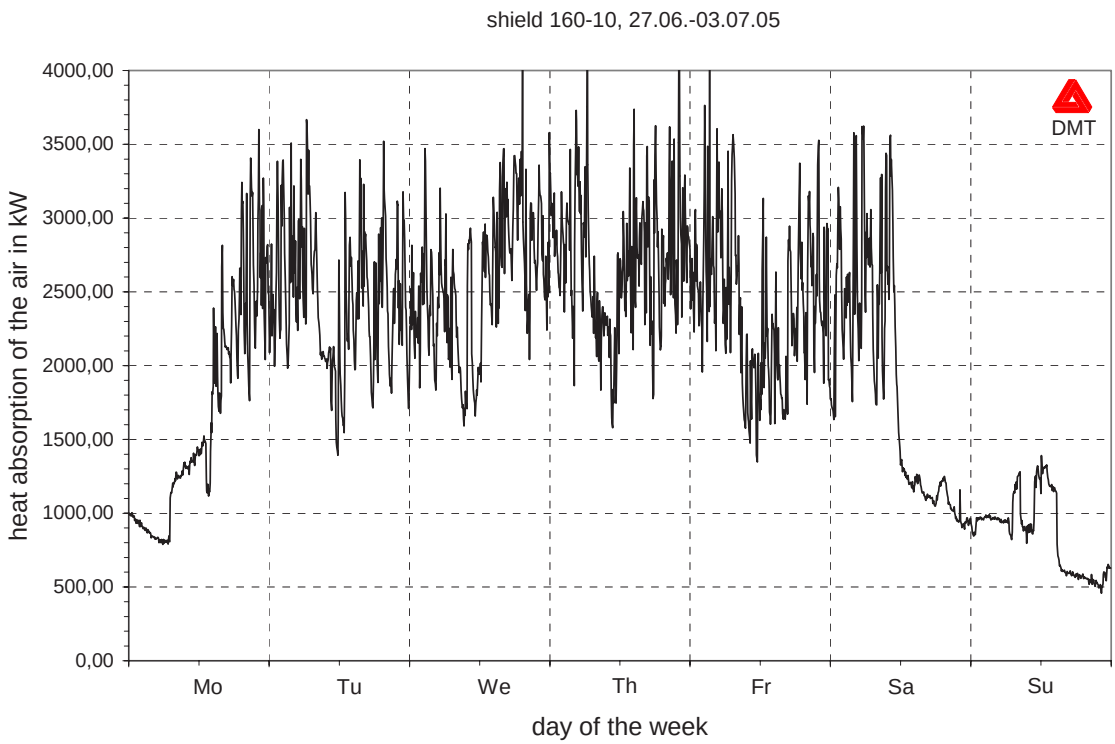


Figure 4: Heat absorption in the face (second working)

During the measuring periods, the following significant outputs were achieved in the two workings. The average useable output measured 6290 t/d in the first working and 4350 t/d in the second working. Including the respective dirt fraction, a bulk output of 7790 t/d was achieved in the first working and an average bulk output of 5440 t/d in the second working.

2.3.1.2.2 Influence of the face slip on the climatic values

The investigations were also aimed at determining the influence of the face slip on the climatic air condition. For evaluation purposes only the boundary conditions occurring during regular production could be used.

The profile of the coal face for the first working (plough operation) during an extraction day is shown in figure 5. The maximum distance between the furthest advanced shield and the main drive measured approx. 11 m, the maximum distance to the auxiliary drive approx. 18 m. During extraction, the coal face was advanced uniformly such that the differences were only very slight during the measuring period. A comparison of the values measured during extraction thus did not promise success.

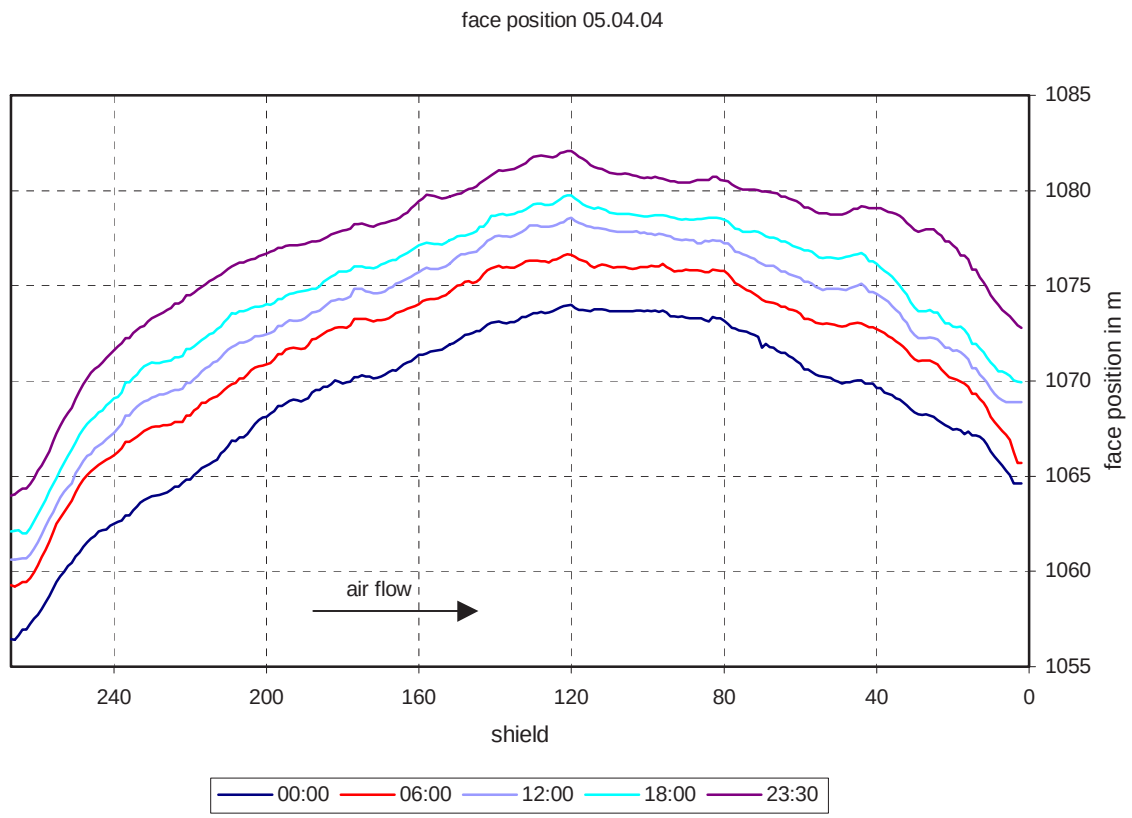


Figure 5: Profile of the coal face on an extraction day

As clearer differences in the face positioning appeared at two weekends, the measurement values for these days were considered in greater depth. The relative face positions for the comparison weekends are shown in figure 6. The heat absorption in the face between shields 262 and 11 can be found in figures 7-8.

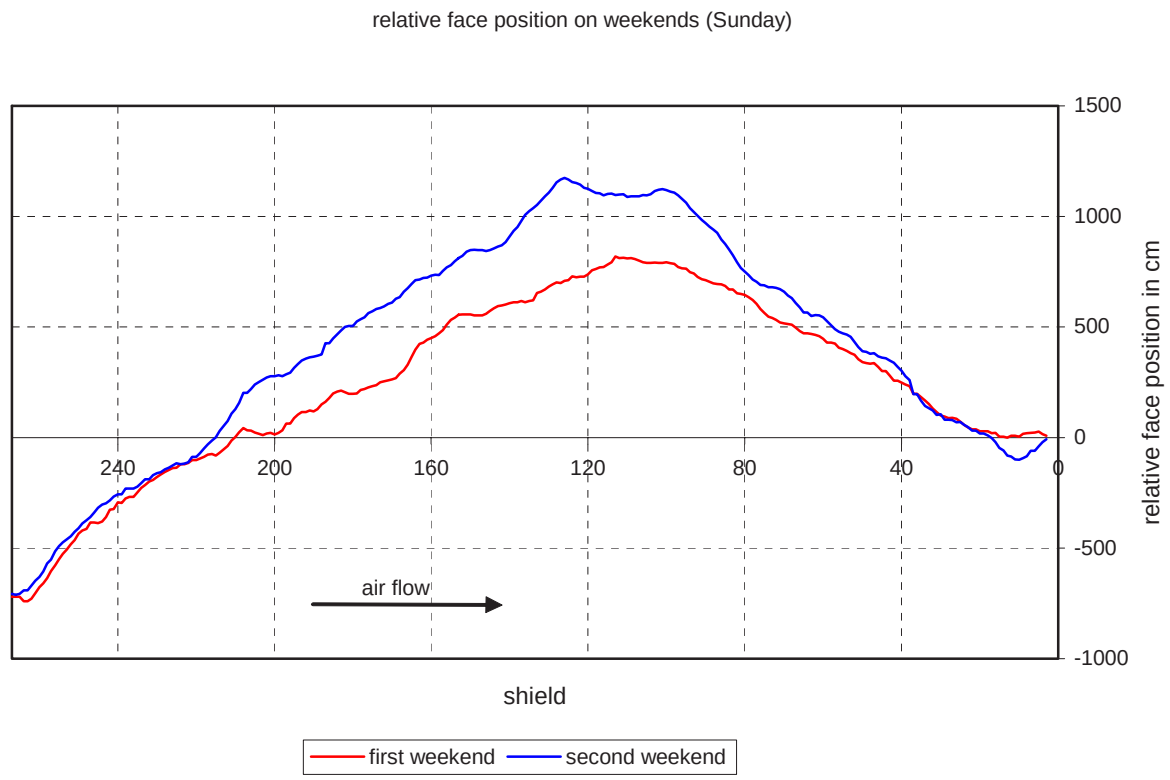


Figure 6: Relative face position at two consecutive weekends

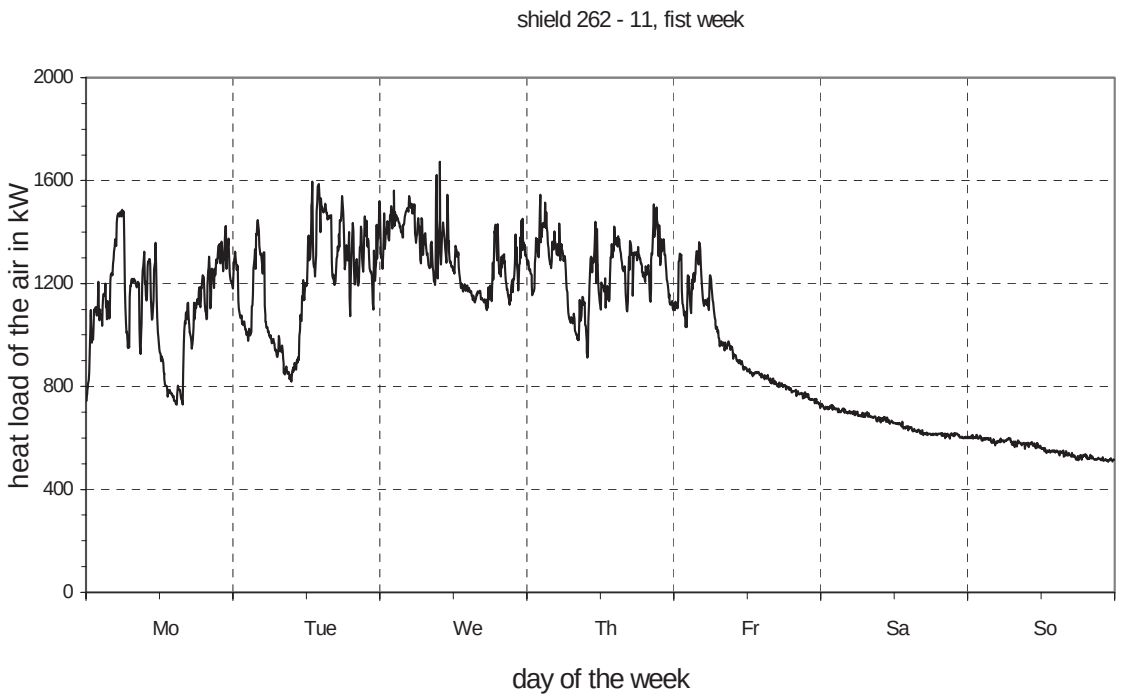


Figure 7: Heat absorption between shields 262 and 11 in the first week

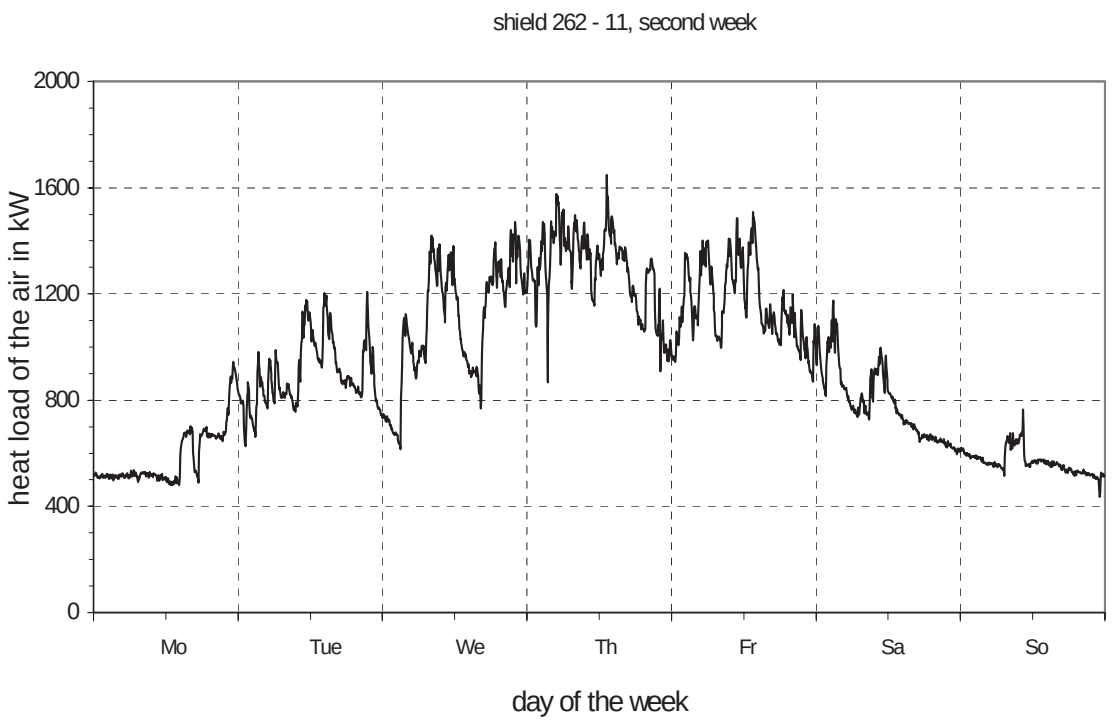


Figure 8: Heat absorption between shields 262 and 11 in the second week

Both weeks, the auxiliary drive was set back relative to the main drive by the same dimension, approx. 7 m. By the second weekend, the coal face had moved forward approx. 3.6 m further than at the first weekend. Although the face position at the second weekend means greater heat absorption should be expected, almost identical heat was measured in the face on both Sundays. With a result like this it must be taken into consideration that not only the face position but also the condition of the abandoned workings determines how much fugitive air influences the climate condition in workings. It was established during the inspections of the working that the roof break in the abandoned workings was uniform at these times. The relatively small cavities led to a reduced proportion of fugitive air, which consequently had less of an influence on the climate conditions in the face.

The face of the second working (shearing operation) was run without significant bulge during the measuring days shown. Measurements in this working did not therefore yield relevant results for this point of investigation.

It can be determined that the other influencing factors, such as heat emission from the equipment, heat emission from the rock and the run-of-mine coal, and the influence on the fugitive air streams of the slope of the roof in the abandoned workings mask the influence of the air inflow, dependent on the face positioning. Partial air streams from the abandoned workings, found repeatedly in the face, exerted only local influence on the climatic air condition. The extent of these inflows is also influenced by the condition of the abandoned workings. Relative to the other influencing factors, the face positioning has a negligible effect on the face climate. Findings to date show that advancing the central face area by around 10 m would not have any demonstrable adverse effect on the climate.

2.3.2 Task 3.2 - Investigation of heat flow reduction by means of climate window configuration

Climate windows serve to prevent any additional adverse effect of the air inflow from the goaf at the face end on the highly loaded air conditions in this area. Extraction of highly heated air through the climate windows, however, leads to an additional load on the return air flow.

Multiple measurements for the first working revealed air streams from the climate window averaging 1.3 m³/s. Relative to the incoming air stream, this constitutes 5 %. The outlet temperature at the climate window was close to the rock temperature, averaging 38 °C dry and 37 °C moist. Due to the low volume proportion and the relatively low temperature values, the air from the climate window was not able to significantly affect the climatic condition of the total air.

Several measurements were conducted at the climate window in the second working. The variations on the number of windows originally planned for the measurements could not be conducted for operational reasons. The average air flow rate was 3.8 m³/s. Based on the density of 1.23 kg/m³ determined for the discharge temperatures, a mass flow rate of 4.7 kg/s was obtained. This volume corresponds to 9.1 % of the overall face air quantity. For the climate window outlet, dry temperatures of 54.0 °C were measured at a wet bulb temperature of 51.0 °C.

Average depth	-1455 m
Face length	300 m
Seam thickness	2.4 m
Installed electrical power in the face	3240 kW
Overall face air volume	37.8 m³/s
Face cooler capacity	800 kW
Rock mass temperature	64.2 °C
Dry temperature at face inlet	24.7 °C
Wet bulb temperature at face inlet	20.8 °C

Table 2: Basic data for the climate forecasts

To determine the influence of this air flow on the face air conditions, data obtained from measurements was used for climate comparison forecasts. Data used for calculations is listed in Table 2.

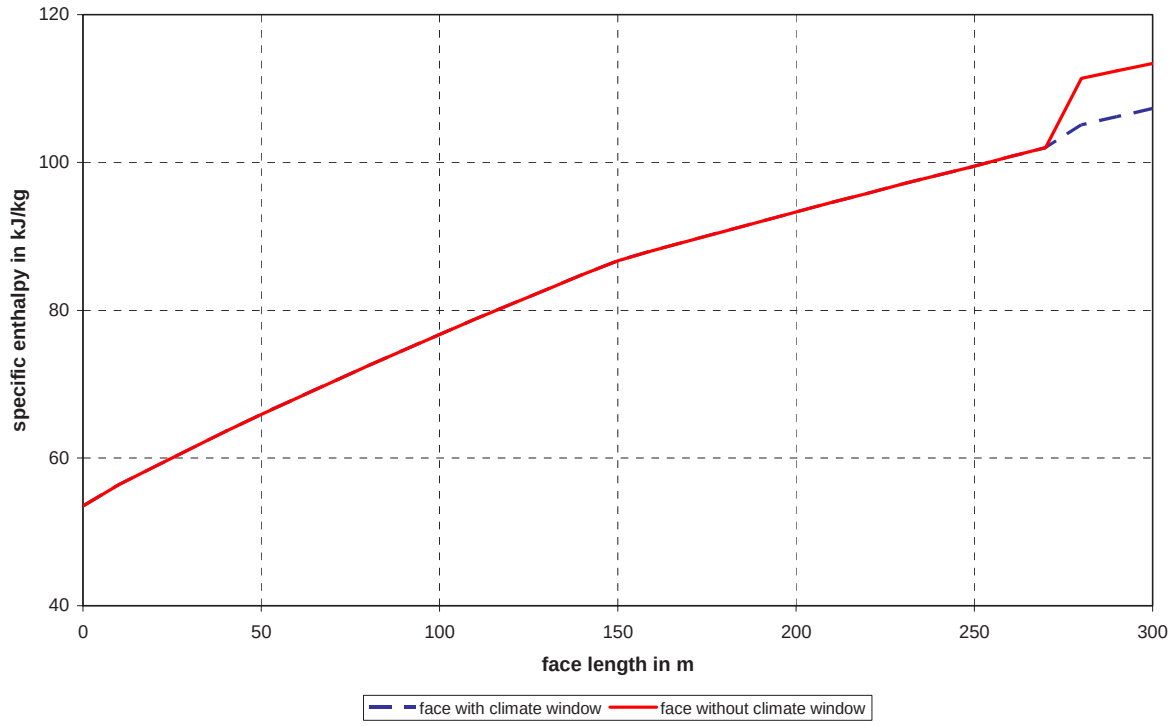


Figure 9: Specific enthalpy curve in the face with and without climate window

Figure 9 shows the specific enthalpy curve for the face. In the absence of a climate window, the final face area is clearly influenced by the additional inflow from abandoned workings in the face. The result is an enthalpy increase from 107.3 kJ/kg to 113.4 kJ/kg. This means an increase in the effective temperature of 1.4 K. This increase may result in the permitted climate thresholds no longer being complied with, especially in workings with high climatic loading. Given the face air at the face outlet after it mixes with air streams from abandoned workings, both calculation examples with 118.9 kJ/kg and 119.2 kJ/kg yield almost identical values. Even if climate values in the case of an existing climate window in the return area are locally higher, the influence on the overall mixed stream is negligible. It must be concluded that climate windows significantly contribute to the reduction of climatic loading in the end area of the face.

2.3.3 Task 3.3 Formulation of a method of calculation to predict the heat reduction as a result of a certain face position and/or a specific climate window configuration.

The research work for the working package 3.1 shows that the climate influence of the face position is covered by a multitude of other effects which can't be regularly planned. Therefore a calculation method for the climate prediction as a result of certain face position will not lead to any success. For workings with high rock temperatures climate windows have a remarkable effect on the climate situation of the last third of the face. It is possible to calculate a climate situation for climate windows with a certain air flow with the DMT climate programme. The air flow situation at the climatic window can't be predicted because the rock fall as most important influence on the air stream in the goaf has no regular forecast.

2.3.4 Task 3.4 Improvement of the infrastructure and the capacity of heat exchangers specifically in thin seams

2.3.4.1 Influence variables for the cooling capacity of face cooler

Cooling capacity is not a constant. In addition to the design and the exchange surface area present, the capacity of coolers is mostly determined by the water and air-related inlet conditions. The influencing variables are listed below:

- air mass flow rate
- air inlet temperature
- air inlet moisture content
- cold water mass flow rate
- cold water inlet temperature
- degree of dirt accumulation at the heat exchange surface

The degree of dirt accumulation has a significant influence on the cooling capacity. The extent to which dirt tends to accumulate can be affected by the design of the cooler. Investigations have shown that with good servicing, the capacity loss due to dirt accumulation in mining can be below 10 %. With poor servicing and without dedusting devices, however, the losses can be more than 70 % of the original capacity. Experience shows that with adequate dedusting and regular servicing, the capacity losses can be lower than 20 %, even at heavily loaded working points.

	t_{db} = 32 °C, φ = 50 %		
Cold water inlet [°C]	Δt_{max} [°C]	Δx_{max} [g/kg]	Δh_{max} [kJ/kg]
6.0	26.0	9.3	50.2
10.0	22.0	7.4	41.5
	t_{db} = 32 °C, φ = 90 %		
6.0	26.0	21.9	82.6
10.0	22.0	20.1	73.9

Table 3: Maximum cooling potential for a cooler under various inlet conditions.

The maximum capacity of a heat exchanger is determined by the inlet conditions of the air and water. For the air, not only the dry temperature is decisive. The air moisture content has a significant influence on the capacity. The cold water temperature is the temperature and moisture sink towards which the heat exchanger moves. The water steam content that would result from the saturation of the air for the cold water temperature is calculated as the draw point for moisture. The theoretical cooling potential is determined from the maximum temperature and/or moisture content difference. In table 3, below, the differences are shown, together with the enthalpy differences for the cold water temperatures 6 °C and 10 °C and for air moisture contents 50 % and 90 % at an inlet air temperature of 32 °C.

The values in the table show that a change in the moisture content of the air results in a significant change in the specific enthalpy. If the low moisture content at 32 °C were kept constant and the specific enthalpy of the high air moisture content were to be achieved, the dry temperature would have to be raised to 63 °C.

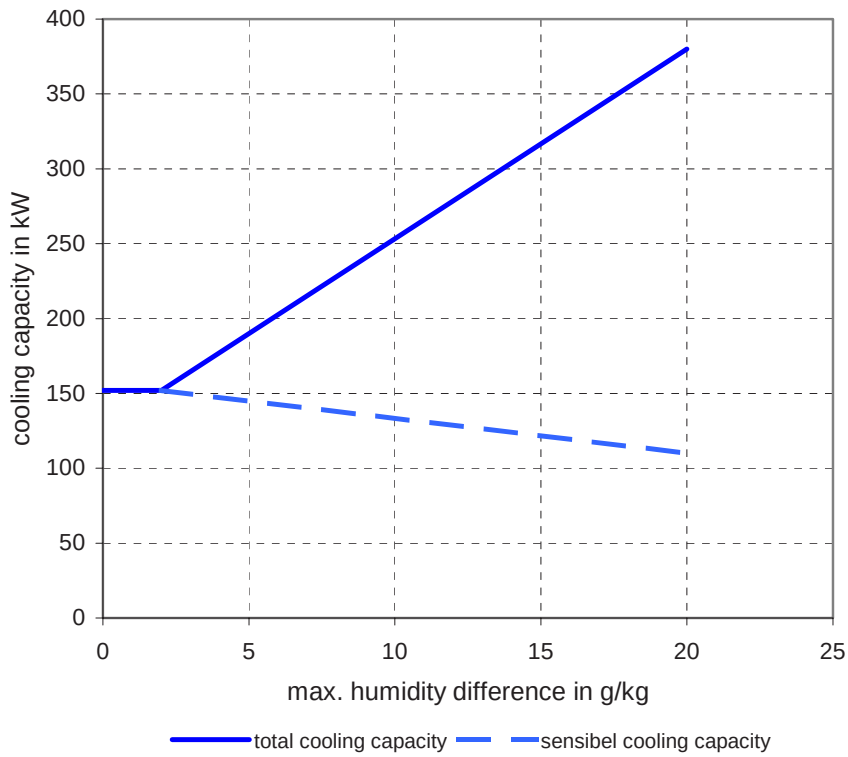


Figure 10: Performance of a roadway cooler for various moisture content differences

In figure 10, the performance of a roadway cooler is shown for various moisture content differences. In this example, the inlet air temperature was kept constant at 32 °C and the cold water inlet temperature at 10 °C. It is shown that changing the moisture content difference has a significant effect on the cooling capacity. After a short constant area, a steep increase in cooling capacity begins. The shape of the curve can be explained as a dry heat exchange takes place at low moisture contents, solely because of the temperature difference. As the moisture content difference increases, the heat exchanger surfaces become covered in condensation. The condensation leads to an increase in the heat exchange surface area. The drops of condensation also lead to greater air turbulence and thus better heat exchange on the surface. When the temperature on the heat exchanger surface falls below the dew point, the evaporation enthalpy of the water is used for the heat exchange such that a significant increase in cooling capacity can be achieved. The total cooling capacity is thus composed of a sensible portion, which results from the temperature difference, and a latent portion, caused by the condensation on the surface. The curves in figure 10 show a decrease in the sensible cooling capacity with increasing moisture content difference. Increasing condensation leads to a decrease in the dry temperature in heat exchangers, causing the sensible portion to fall because of the lower temperature difference.

Altering the inlet flow rates of the air and cold water likewise has a significant effect on the cooling capacity. At very high flow rates, the capacities approach a limit curve, where any further increase in the flow rates results only in a very slight increase in capacity.

The various factors influencing the capacity of a cooler show that the working characteristics for the coolers used must be known in order to have sufficient planning certainty for the intended deployment location with the inlet conditions expected there. In order to determine the cooler performance with an advance calculation programme, certain fundamental characteristics of the coolers must be known.

A heat exchanger enables the flow of heat from a fluid with a high energy level to a fluid with a low energy level. The driving variable here is the temperature difference between the fluids. The transfer of heat in a cooler can be described in simplified form with the flow of heat in a pipeline.

The heat exchange takes place in three stages:

- transfer of heat from the air to the outer surface of the pipe by convection
- conduction of heat from the outer to the inner surface of the pipe
- transfer of heat from the inner pipe surface to the cold water by convection

The external heat transfer has the greatest influence on the heat exchange here. The variable is determined here by the geometry of the exchange area and the turbulence and moisture on the surface. At the moisture levels encountered in coal mining, condensation generally takes place on the surface of the heat exchanger. In addition to the heat transfer by convection, a mass transfer also takes place. The evaporation enthalpy of the condensed water contributes to the cooling capacity such that a significant increase in capacity occurs with increasing moisture content difference, as shown in figure 10. The combined cooling capacity of the sensible and latent cooling capacities is termed the total cooling capacity.

For the heat exchange, the following equation can be formulated with the heat through-flow coefficient kt:

$$\dot{Q}_t = k_t \cdot S_1 \cdot (t_1 - t_2)$$

For the heat through-flow coefficient, the following applies:

$$\frac{1}{k_t} = \frac{1}{\alpha_t} + \frac{d_1 \cdot \ln(d_1 / d_2)}{2 \cdot \lambda} + \frac{d_1}{\alpha_2 \cdot d_2}$$

The markings in the representation have the following meanings:

$\dot{Q}$: heat stream in W

t_1 : air temperature in °C

α_t : total heat through-flow coefficient between air and outer pipe surface in W/(m²K)

S_1 : outer pipe surface in m²

λ : thermal conductivity in W/(mK)

t_2 : cold water temperature in °C

α_2 : heat through-flow coefficient between cold water and inner pipe surface in W/(m²K)

d_1 : outer pipe diameter in m

d_2 : inner pipe diameter in m

A heat exchanger consists of a multiplicity of pipes with various water channels and parallel paths. For ribbed pipes, further factors apply. Within an air cooler, the air and water-related conditions change throughout the heat exchange. A capacity calculation is especially only possible for moist heat transfer if thermal ratings are determined in advance by test measurements.

2.3.4.2 Determination of characteristics for face coolers

Capacity measurements for face coolers were conducted at the DMT test stand. The objects were a cooler with a diameter of 350 mm (SK3) and a cooler with a diameter of 500 mm (SK4). The inlet conditions for both coolers were altered within a wide range in order to obtain an adequate basis for determining the heat transfer coefficients. For both coolers, the relationship of their capacities to the water flow rate, to the cold water inlet temperature, to the air flow rate and to the inlet moisture content were determined.

In the following, the results of the measurements with the SK3 cooler are shown. Initially, the relationship of the capacity to the cold water inlet temperature and the cold water flow rate were investigated. For this purpose the cooler was driven with a standard ventilator ES 3.5-11 with a drive capacity of 1.1 kW. During all measurements, the air inlet conditions were kept constant at a dry temperature of 32 °C and a wet bulb temperature of 28 °C. While varying the cold water flow rate, the cold water inlet temperature was 7 °C. The varying cold water inlet temperatures were run with a cold water volume flow of 4.4 m²/h. The curves are shown in figures 11 and 12.

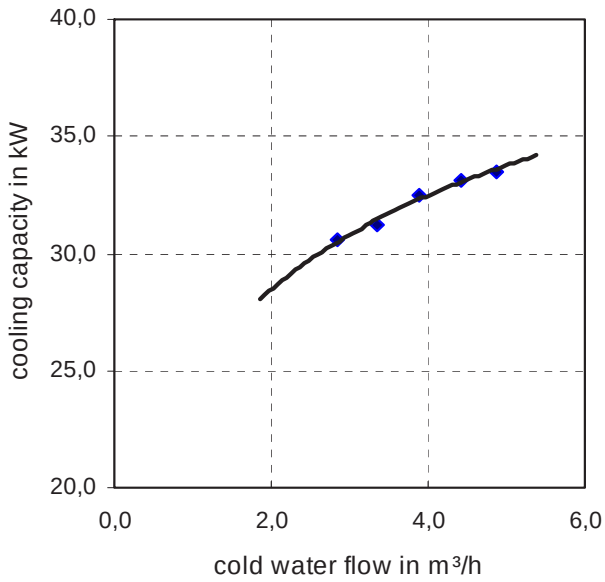


Figure 11: Cooling capacity relative to cold water flow rate

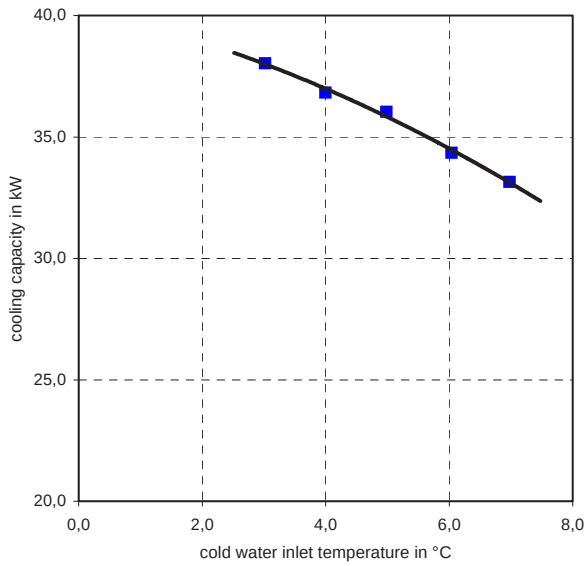


Figure 12: Cooling capacity relative to cold water inlet temperature

The characteristics of the face coolers are comparable to the performance of the roadway coolers. An increase in the cold water flow rate leads to the capacity curve coming asymptotically closer to the limit curve. The same applies to the relationship to the cold water inlet temperature.
In order to calculate the heat transfer coefficients for the air, the capacities for dry heat transfer have to be determined for different ventilation velocities. Additionally, the characteristics of the cooler for varying air inlet temperatures must be known. In figure 13, below, the measurement results for various air

flow rates during dry heat transfer are shown. The air inlet temperature was kept constant at 37 °C. The dew point here was always below the cold water temperature, so that this point was never fallen below. As regards the water, the cold water flow rate of approx. 5 m³/h and the inlet temperature of approx. 12 °C remained unchanged.

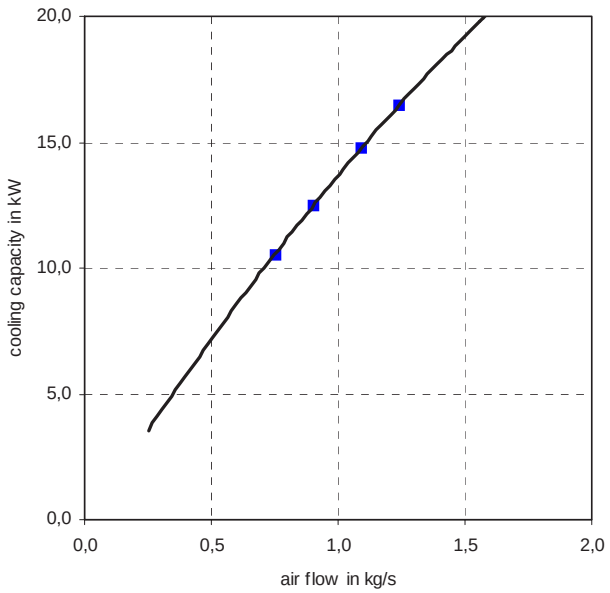


Figure 13: Relationship of dry cooling capacity to air mass flow rate

Capacity change is graphically represented in figure 13. The direct relationship of the capacity to the air heat transfer coefficient □1 can be seen here.

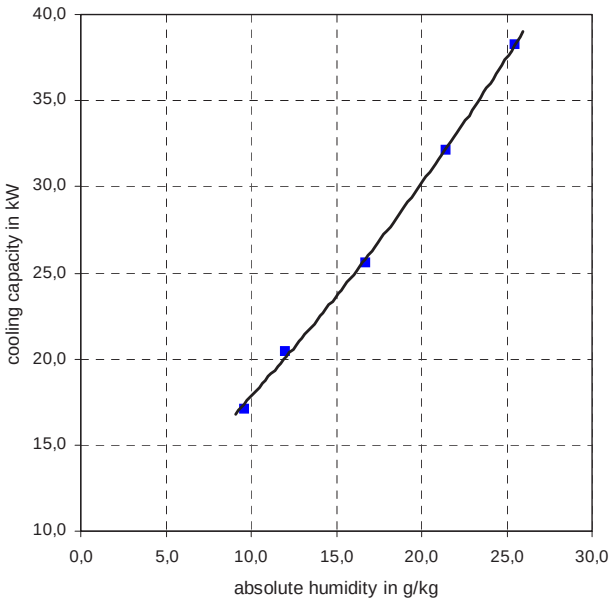


Figure 14: Relationship of the cooling capacity to the inlet moisture content

The capacity data for the measurements with different air moisture contents are represented in figure 14. The air inlet temperature was kept constant at 32 °C and the air mass flow rate at approx. 1 kg/s. As regards the water, the flow rate value of 5 m³/h and the cold water inlet temperature of 7 °C remained unchanged. An increase in the air moisture level led to a steep increase in cooling capacity.

The measurements for the SK4 cooler were conducted in a comparable manner. The performance of this cooler turns out similarly.

On the basis of the capacity measurements carried out, the heat transfer coefficients were calculated both for dry and for moist heat transfer. In figure 15, the relationships for face cooler SK 4 are shown.

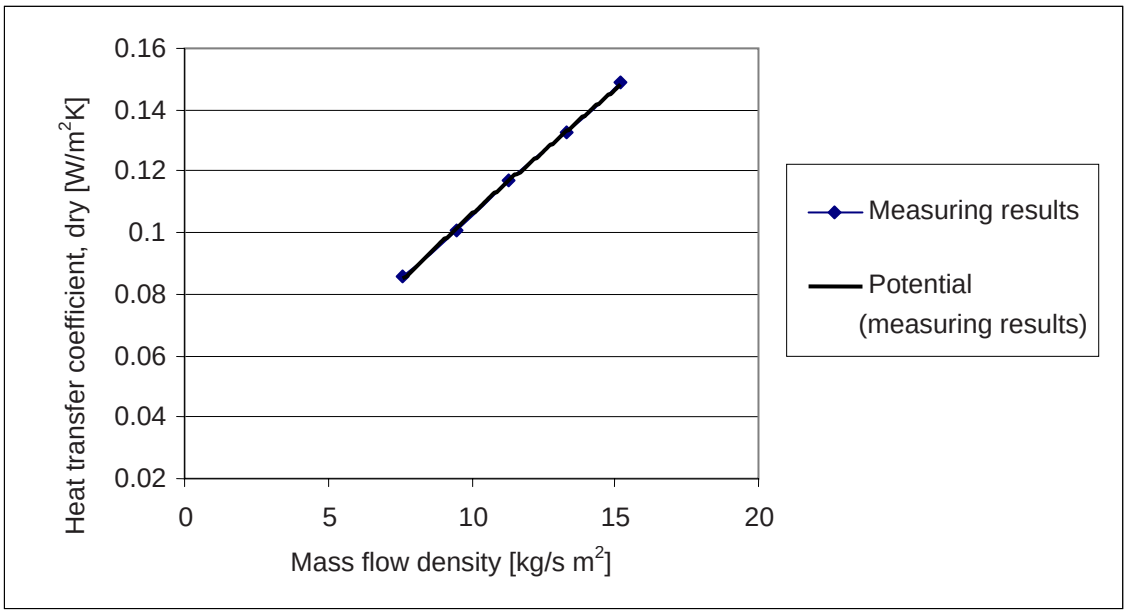


Figure 15: Dry heat transfer coefficient relative to the mass flow rate density of the water for face cooler SK 4

The heat transfer coefficient for moist heat transfer is shown relative to the ratios of total to sensible cooling capacity. Figure 16 shows the relationship of the heat transfer coefficient for face cooler SK 4 during moist heat transfer.

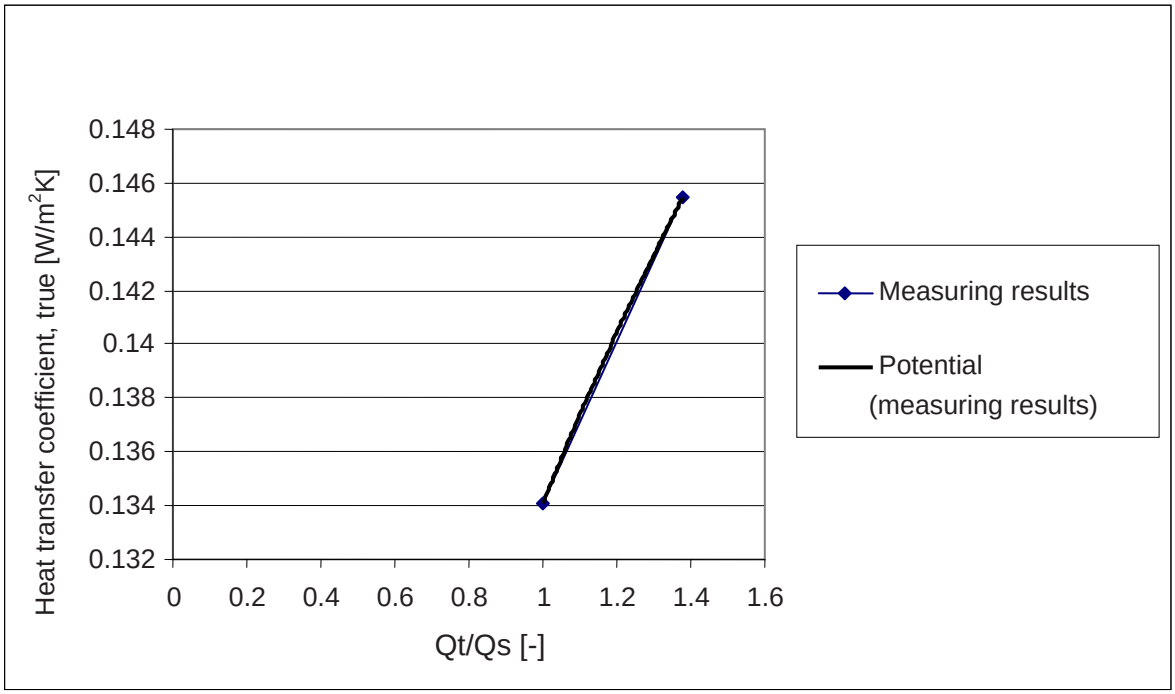


Figure 16: Heat transfer coefficient for moist heat transfer relative to the ratio of total to sensible cooling capacity for face cooler SK 4

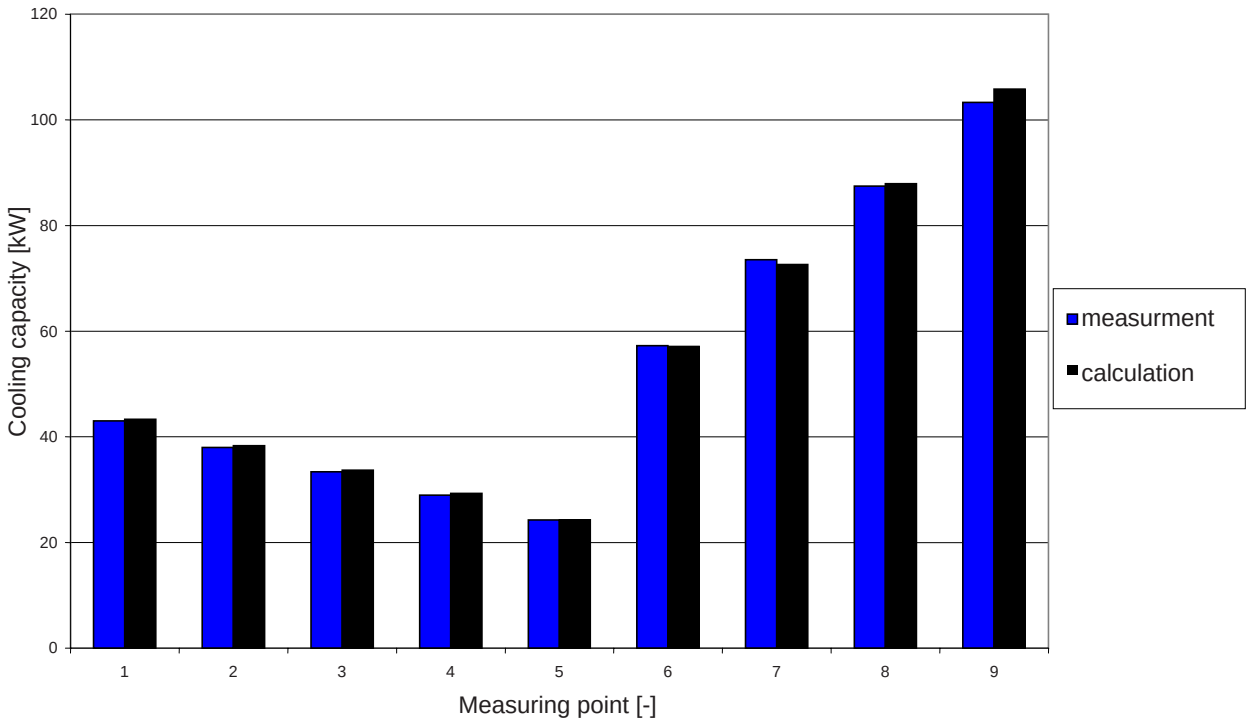


Figure 17: Comparison of measured and calculated cooling capacities for face cooler SK 4

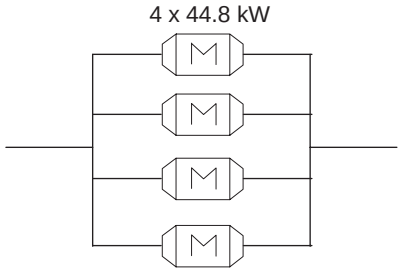
Incorporating specific data on the cooler, such as the surface area of the heat exchanger, flow cross section, pipe inner and outer diameters, number of parallel cooling pipes etc. and the experimentally determined heat transfer coefficients, the cooling capacities of coolers can be calculated in respect of the inlet conditions for air and water with the aid of the DMT cooler calculation programme. Figure 17 shows the comparison of the measured and calculated cooling capacities for face coolers SK 4. The strong correlation shows that, using the characteristic values taken, the capacities of face coolers can be determined for the widest range of inlet conditions.

2.3.5 Task 3.5 Investigation of potential reduction of the coolant volume stream at the face

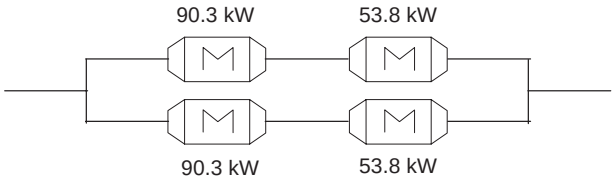
Due to the space restrictions, there are limits on the number and size of coolers in the face, but water supply lines should also take up as little space as possible. It thus follows that an optimum must be found regarding the existing cooling potential of the cold water, the number of air coolers and the kind of the water-related interconnection.

A general statement cannot be made as to the optimal configuration and combination of parallel and series connections. An optimum must be calculated here based on the individual air and water-related conditions. Under point 1.2, some limiting conditions are specified, e.g. for the minimum water quantity.

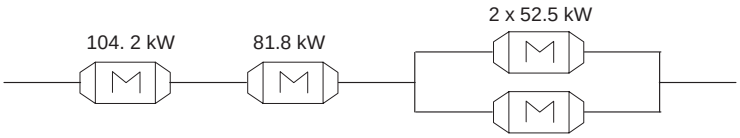
By way of example, four different interconnection arrangements for four SK 4 coolers will be investigated. The available water quantity is to be 16 m3/h and in a second series of calculations 12 m3/h with a cold water inlet temperature of 7 °C. For the dry temperature 32 °C, for the wet bulb temperature 28 °C and for the air flow rate 2.5 m³/s are assumed. The following diagrams schematically represent the circuits and the cooling capacities of the individual coolers.



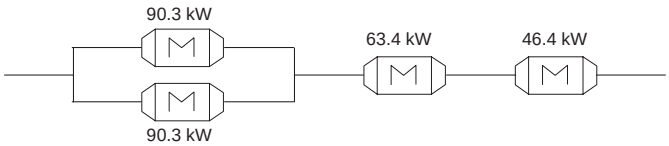
Circuit 1: Four parallel coolers



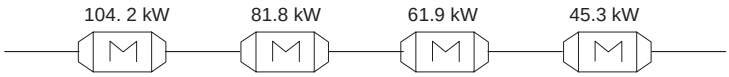
Circuit 2: Parallel connection of two coolers connected in series



Circuit 3: Two coolers in series, then two parallel



Circuit 4: Two coolers parallel, then two coolers in series



Circuit 5: Four coolers in series

Table 4 below shows a summary of the results.

Circuit no.	Total cooling capacity [kW] with 16 m^3/h	Total cooling capacity [kW] with 12 m^3/h
1	179.2	163.6
2	288.2	244.2
3	291	267.1
4	290.4	246.4
5	293.2	249.2

Table 4: Total cooling capacity with varying water-related interconnections

Through the form of the interconnection alone, the cooling capacity can be increased by up to around 64 %. It can also be seen, however, that depending on the boundary conditions, different circuit arrangements may prove to be optimal, and that a calculation taking the individual conditions into account is therefore necessary for optimal planning.

2.3.6 Conclusions

The examinations on the climatic influences of the face positioning revealed that there are no considerable influences on the face climate due to the usual advancing of the breast centre region. The measurements showed that face operations can be optimally operated within the current limits from a mining point of view without having to expect any negative influence because of the face positioning.

The characteristic values for face coolers determined during the research project allow calculating the cooling capacity. By calculating it in advance, suitable water circuiting of the coolers as well as an optimum fitting position can be achieved. This results in a technically and economically improved use of the cooling capacity of the cold water and of the face coolers to be installed.

2.3.7 Exploitation and impact of the research results

Technical and economical potential for the use of the results

The results of the investigation of the heat and gas flow in relation to face position and the further development of the calculation programme allow the mine ventilation engineers of deep mining companies to optimize the cooling system and/or the cooling capacity of mining operations.

Advantages for the security of the workers

Optimal cooling system is more important in deep hard coal mines particularly with regard of the increasing depth (and as a consequence the depth-related rise in rock temperature and the self-destruction of air temperature) and furthermore the increasing installed capacity of the electrical equipments. It is becoming ever more difficult to keep the climatic limit values. With an improved cooling system mining workers receive better physiological working conditions.

Like WP 3 from DMT the next WP 4 from MRSL deal also with the climatic condition in deep mines and with the aim to improve health and safety of persons.

2.4 WP 4 – Climate Studies of Arduous Working and Emergency Activities (MRSL)

Introduction

The principal objective of Work Package 4 is to provide an improved understanding of the ability and shortfalls of working, emergency intervention and rescue operations in deep, laterally extended mines. Subsidiary objectives include:

- Development of improved monitoring and visualisation techniques
- Development of an integrated thermal risk management methodology
- Improve emergency intervention and rescue procedures in high heat stress conditions.

The work package consists of five individual tasks and an overview of each task is given in the following sections.

2.4.1 Task 4.1 - Literature review

2.4.1.1 Specific Task Objectives and Planned Activities

The objective consisted of a scoping and industry review with specific consideration given to the challenges posed in respect of the development of laterally extended working in deeper mines. Consideration was given to how perturbations in the mining process and design changes to mine environment control measures could affect local mine climate and introduce short-term but excessive temperatures. Alternative methodologies of cooling system changes in given ventilation circumstances were also to be considered.

The planned activities were carried out as per schedule with the data obtained being utilised as necessary in each respective task within this work package. The review was carried out in two phases as described in the following sections.

2.4.1.2 Thermal Physiology and Physiological Measurement Techniques

As a background to the various MRSL work packages associated with practical intervention responses and monitoring methods, a review of literature and practice was conducted, covering the following subsections:

- Review of underground climatic conditions
- Management of heat stress in the industrial workplace
- Mining guidelines and practice
- Core body temperature analytical models
- Monitoring of core body temperature and heart rate

This part of the review surveyed underground climatic conditions and examined what constitutes practical, safe limits in physiological terms. Thermal analytical models, measurement principles and instrumentation options for determining key physiological parameters were also examined. Focus was given to the measurement of deep body core temperature, since this is considered as the most critical physiological parameter concerning physical activities carried out in hot and humid underground mine conditions.

2.4.1.3 Character Behaviour and Accuracy of Measurements

The work concentrated on determining the characteristic behaviour and accuracy of candidate measurement techniques, initially considering application in research investigations, but subsequently considering possible use in the workplace. The following sections were reviewed and analysed in detail:

- Thermal physiological measurement standards – ISO 9886: 2004
- Body core temperature measurement methods and attainable accuracy

- Application of arterial heat balance compensation methods
- Skin temperature measurements
- Safe upper limits of body core temperature
- Body core temperature instrument development issues.

Heat stress is a multi-component hazard in many workplaces and may be evaluated using environmental/psychometric parameters, together with an estimation of metabolic rate along with the effects of clothing. Typically there is high variability and exposure assessment is an approximation. Therefore, a large safety margin is required. Measurement of heat strain may include skin/oral temperature, body core temperature, weight loss from sweating and heart rate criteria. Physiological monitoring may include periodic or continuous personal monitoring. Whilst heat stress limits predicted by exposure assessment may be exceeded with caution, heat strain derived from physiological monitoring must not exceed threshold limit values or other agreed safety limits. This work formed the basis of developing an integrated thermal risk assessment methodology. Detail from the review was also used to provide some benchmark standards for carrying out the various scheduled trials in the Environment Chamber.

2.4.2 Task 4.2 - Development of improved visualisation and monitoring techniques

2.4.2.1 Specific Task Objectives and Planned Activities

The specific task objectives were:

- Investigate the scope for the introduction of hand held, intrinsically safe, instruments to measure Basic Effective Temperature and provide real time information.
- Investigate the use of computational fluid dynamics modelling software for use as a visualisation aid for illustrating heat loading in mine atmospheres.

Both of the specified task objectives were carried out and achieved within the planned time schedule. Although the visualisation aspects of using computational fluid dynamics were adequately demonstrated, it was evident that there was a requirement to provide a more in depth assessment of the visualisation technique, of modelling, validation and mine information. The work was extended to provide additional data and to investigate the use of the visualisation technique as a possible predictive aid to mine emergency escape and rescue activities. It was envisaged that this work would also benefit the other tasks within this work package.

2.4.2.2 Basic Effective Temperature (BET) Monitor

The design of a portable BET monitor using the latest commercially-available sensors and microprocessors has attendant difficulties. The design of a rugged instrument requires additional consideration of the sensors to be employed, together with confirming their long-term stability in the underground environment. Additionally, designing for ATEX M1 intrinsic safety imposes severe constraints on power consumption and the choice of electronic components. The scope of this part of the task was to outline a 'proof of concept' design that would provide a vehicle for developing the hardware and software, the human interface (display, keyboard), and its functionality (logging capability, alarms etc).

BET may be obtained from a nomogram of wet and dry bulb temperature, against differing wind speeds. For implementation in a microprocessor, it is necessary to reduce the nomogram to either a deterministic algorithm or a "*look-up*" table. The advantage of a "*look-up*" table is that it simple to implement, since it is based on a tabular version of the nomogram. However, the disadvantage is that the algorithm may still need to interpolate between data points (unless the "*look-up*" table has a very fine resolution) so that, computationally, the advantage may be lost. However, algorithms for BET are complex, so a high-resolution "*look-up*" table is the favoured approach.

As a starting point a MatLab program was used to produce a linear algebraic formula from which it was possible to create a "*look-up*" table at any required resolution. The approximations that this process involved were small, except at conditions of high wet-bulb temperature. Unfortunately, it is in this

situation that the BET monitor needs to be accurate and it is recognised that a more accurate model of the nomogram may eventually be required.

Several possible methods of measuring temperature, humidity and wind-speed were investigated. After experimentation and due consideration, some conclusions were drawn on the most favourable type of sensors. As well as considerations of intrinsic safety, decisions were based on the cost of implementation, ruggedness and reliability.

A combined humidity and temperature sensor, the Sensirion SHT75, was chosen due to the fact that a solid-state sensor had the advantages of low-power consumption and convenience of use and that, for this application, the low cost or high accuracy of the other methods was not required. The device is a single chip RH and temperature sensor module that provides a serial digital output. The device includes two calibrated micro-sensors for relative humidity and temperature which are coupled to a 14-bit analogue to digital converter and a serial interface circuit on the same chip. This is claimed to result in a superior signal quality, a fast response time and insensitivity to external interference. Each sensor is calibrated in a precision humidity chamber and the calibration coefficients are programmed into an on-chip memory. This results in a high inherent accuracy, as tabulated below for the SHT75 device. There is still some residual dependence of the digital output of the humidity sensor on temperature, but this can be compensated by applying the recommended algorithm in the user's microprocessor.

For measurement of wind speed, a silicon diode-based 'hot wire' device was chosen. The term 'hot wire' implies high power consumption but a system involving a silicon diode sensor would use only a milliwatt of power. The term also implies 'heat' but, again, a silicon diode implementation would aim only to heat the device to the same temperature as a reference device that was protected from the airflow, so there is no danger in using a 'hot wire' in a hazardous atmosphere. With a sensor based on a pair of silicon diodes, the current flowing in the two diodes could be used to derive both temperature and the degree of cooling of the sensor diode (and hence the wind speed). The disadvantage of this method is that because the principle relies on the airflow to transport heat away from the device, it would be essential to calibrate the instrument, and the calibration may require a matrix of calibration points (i.e. a look-up table) for wind-speed and temperature. However, even a large calibration with thousands of points is trivial to handle in a microprocessor, and the calibration process itself can be automated. A prototype instrument circuit design was proposed, which has the block diagram shown in **Figure 1**. The circuit diagram corresponding to the above block diagram is given in **Figure 2**.

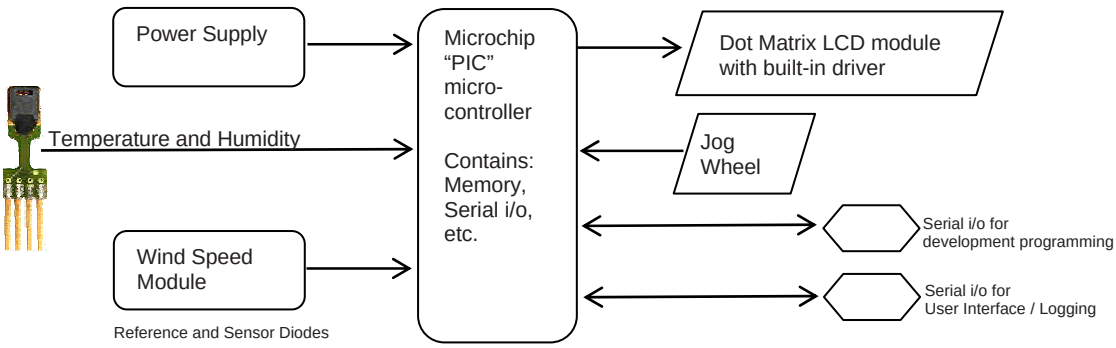


Figure 1: Functional Diagram for the BET Monitor

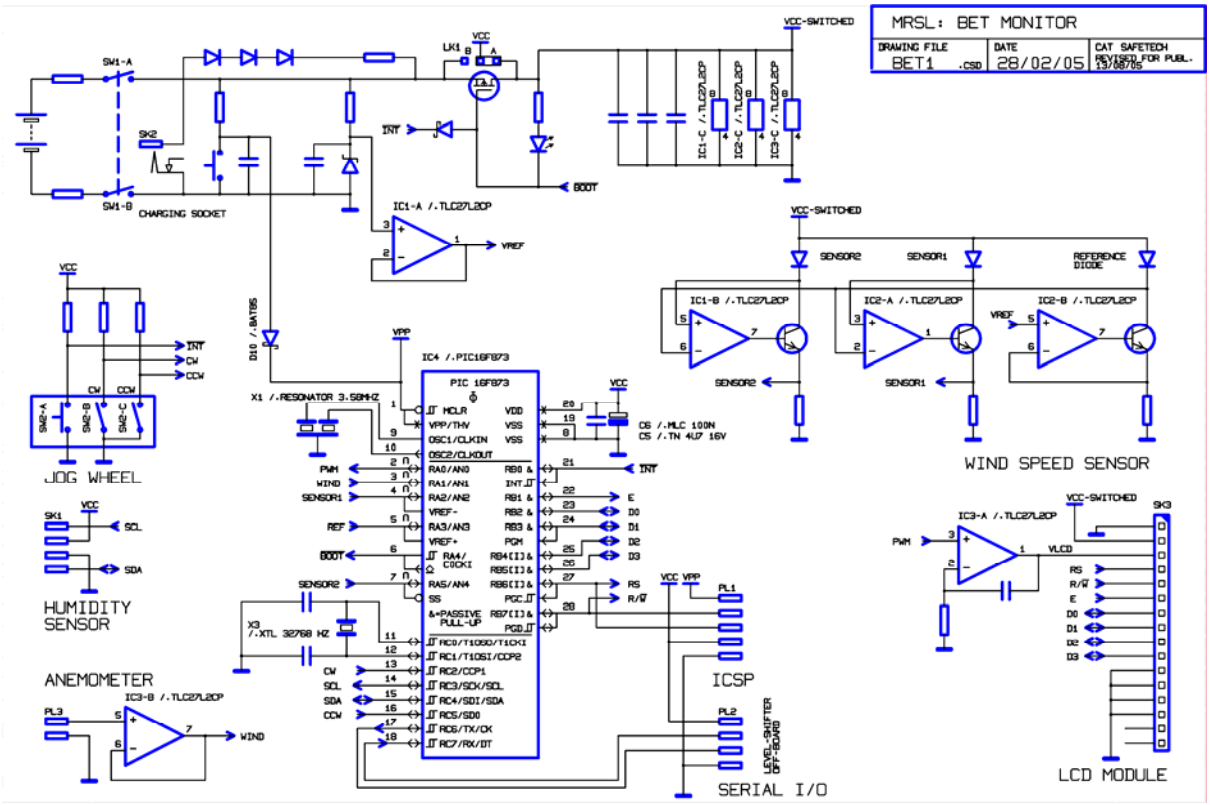


Figure 2: Circuit Diagram of BET Monitor

The power supply is intended to comprise a 3.6V 80mAh rechargeable nickel/metal-hydride battery (e.g. one of the MemPac 'memory protection' range from Varta), which measures 19 × 17 × 19 mm. The purpose of the standby condition is to reduce power consumption if the user forgets to switch the instrument off. This feature is implemented by operating the MCU in its standby mode (1µA current consumption), and powering-down any modules that consume current. The humidity sensor has a power-down mode that consumes 10µA, but the LCD requires to be physically disconnected, therefore the diagram also shows the experimental feature of a 'soft' on-off switch which is 'boot-strapped' by the MCU. It reduces the off-state current consumption to a few microamps.

The instrument has two sensors for wind-speed, largely because there is sufficient processing capability in the MCU to handle an extra input channel. There is also an input for a conventional anemometer, should this prove to be necessary. The temperature and humidity is measured using the MEMS sensor discussed above. The LCD is a standard alphanumeric dot-matrix LCD, comprising two rows of 16 characters. A wide variety of such modules are available with different back-lighting options. Most require a 5V supply, but a 3V version is available, which would be suitable for use. The unit is provided with a serial input and output that can be connected to a PC acting as a dumb terminal. The purpose of this interface is to provide additional test features during development, but this interface could be used to add data-logger functionality to the unit. The interface uses logic-level signals and may therefore require a level shifter as part of the connecting cable.

A second interface allows the unit to be connected to a PIC device programmer. In-circuit programming of the PIC MCU is a useful aid to production, as it means that programmed microcontrollers do not have to be stored as a stock item. The initial estimate is that the design will fit into an enclosure of 50 × 25 × 100 mm.

In summary, designing for ATEX M1 intrinsic safety imposes constraints on power consumption, choice of electronic components and segregation requirements between electrical subsystems.

The design presented above is a realistic development platform on which it is possible to demonstrate sensor performance. The salient point of the design is that it operates at a low voltage (3V), and a low current (40mA), and is physically small. These factors will greatly ease the problem of compliance with hazardous area certification requirements.

In addition to the possibility of achieving low power operation, the feasibility of other key aspects of the design have been confirmed. These include the adaptation of a BET nomogram to calculation by microprocessor, the use of an ergonomically versatile user interface and LCD display, and the use of solid state sensors for wind-speed, humidity and temperature.

The instrument operation and ergonomics within the design study have been given consideration, but may warrant further refinement and appraisal in a commercial instrument design. Further issues beyond the scope of the work programme include consideration of instrument maintenance, calibration and the requirement for training and instruction of mine ventilation staff. These matters are consistent with a commercial development phase.

2.4.2.3 Development of Computational Modelling Visualisation Techniques

Several mine surveys were carried out to obtain information on actual mine conditions which would be required for the modelling procedure. It was also necessary to carry out validation exercises for verification of correct modelling procedure and development of representative, predictive model outputs.

The virgin strata temperature of a section of the colliery was recorded at 21°C, which was considered unusually low when compared to other UK mines of similar depths of approximately 550m. The concept of geothermal gradient was considered to be negligible in terms of the visualisation modelling procedure. Data was obtained from parts of the mine extending to the more recent working areas and was used to update the work as required. In general, the surveys identified heat from machines and electrical equipment as being the major cause of the problem and heat measurements were determined for chosen types used in the mine. Specific examples were chosen and used to develop visualisation modelling techniques.

The modelling development was based on actual colliery dimensions, together with insitu readings of ventilation, temperatures etc of a local colliery and its working district. Some chosen equipment surface temperatures measured as follows:

- Conveyor Motor: 45°C
- Chock Pump: 30°C
- Chock pump motor: 45°C
- Face AFC Coupling: 35°C
- Shearer: 40°C
- Electrical transformer: 90°C
- FSV: 40-87°C

This data was used as a starting point to build and supplement the models with representative insitu values.

The main aim of the modelling part of the study was to analyse the air flow pattern and temperature stratification in heating conditions. The simulations were carried out with an unstructured CFX code, and utilised the shear stress transport turbulence and heat transfer models, also incorporating the Boussinesq approximation to account for buoyancy effects.

With respect to buoyancy, natural and mixed convection flows and flows in which gravity is important can be modelled with this code by the inclusion of buoyancy source terms. Natural convection refers to the case where convection of the fluid is driven only by local density variations, for example in a closed box with a heat source. Mixed convection refers to the case where convection of the fluid is driven both by pressure gradient and buoyancy forces as in a mine roadway.

The numerical results were compared with physical validation measurements. The validation was carried out by readings of wet and dry bulb temperatures and air velocity via a hot wire anemometer over a measured grid covering the required regions.

The source of heat and moisture input into the mine air from a diesel-powered free steered vehicle (FSV) was quantified within the survey. It is evident to a person, driving or standing in the vicinity of a diesel FSV, with the engine running, that it introduces considerable heat and moisture into the local environment. Although the calculations are simplified, the effect was quantified by calculating the average fuel used for the working period of the week which was taken to be 600 litres in a 5 day working period. Each litre of fuel was determined to have a total energy value of 36 x 10^3 KJ and the average rate of power transferred into the mine air was calculated as 50 KW per machine over a running period within 5 working days. Furthermore, with each litre of fuel burned an approximate value of 0.75 litres of water was assessed as being produced.

The heat output is small in comparison to the face machinery but was suggested to intensify problems locally in operational parts of the mine with hot and humid conditions. For modelling purposes, the surface temperature of the vehicle used was recorded. In terms of providing a range of scenarios, several solver runs were initiated, which covered normal conditions and potential problems such as vehicle over-heat, onset of a vehicle fire and within a range of ventilation velocities. Several solutions for each model, including the FSV model were run over the project time span.

The geometry of the generic vehicle was established to be representative of an FSV and placed within a roadway with a cross section 6m wide and 4m high. **Figure 3** shows the basic 3-D geometry where the road and vehicle longitudinal section is halved due to symmetry and therefore, halves the required processing power. The diagram is also inverted to give improved viewing of the vehicle and compacted mesh structure around the vehicle as shown in **Figure 4**. Mesh density is of importance to provide a more accurate representation in areas of interest by focusing on more nodal calculations in a region. In the general roadway area, the mesh is set with less density

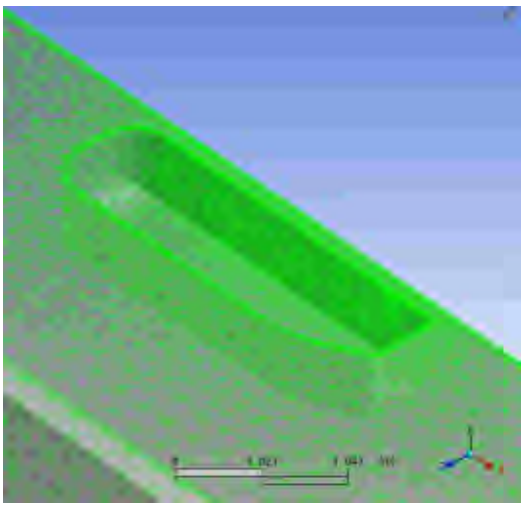


Figure 3: Halved Model Geometry

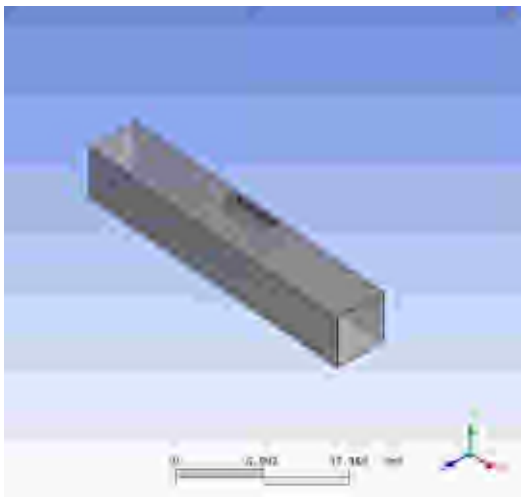


Figure 4: Unstructured Mesh

Solving the problem initially for an aerodynamic solution shows the recirculation behind the vehicle resulting from being parked in a ventilation stream of 2 m/s at 25°C and the vehicle surface temperature was set at 40°C. This is illustrated by the velocity vector diagrams in **Figures 5 and 6** which show the global temperature distribution and velocity vector detail around the vehicle respectively.

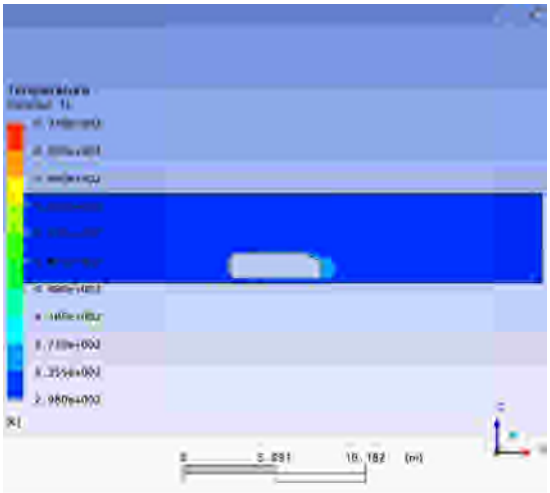


Figure 5: Temperature Distribution

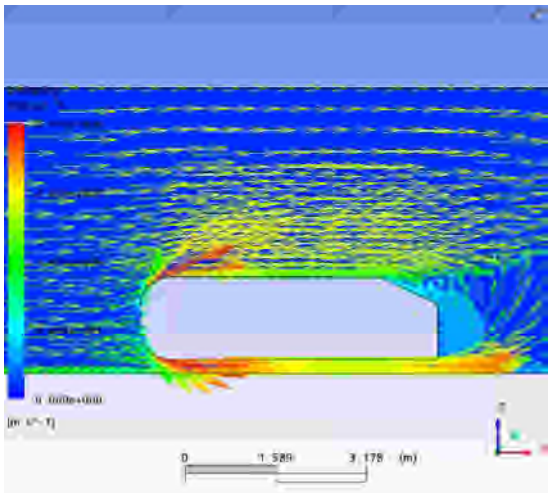


Figure 6: Velocity Vectors

As can bee seen from **Figures 5 and 6**, there is adequate ventilation to deal with the heat output from the surface of the vehicle. The velocity vectors indicate a good mixing regime and rapid flow over and around the vehicle. This example demonstrates the visualisation concept of a normal situation and the validation measurements gave good comparative results.

The model was resolved with a reduced ventilation of 1m/s and a vehicle surface temperature of 400°C to represent the initiation of a vehicle fire. **Figures 7 and 8** show the revised temperature distribution and velocity vectors respectively.

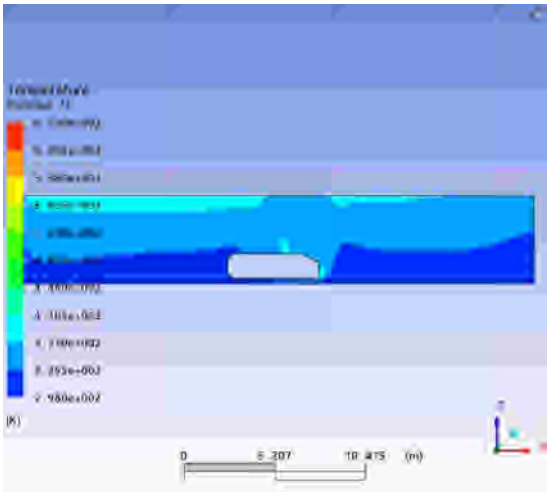


Figure 7: Revised Temperature Contours

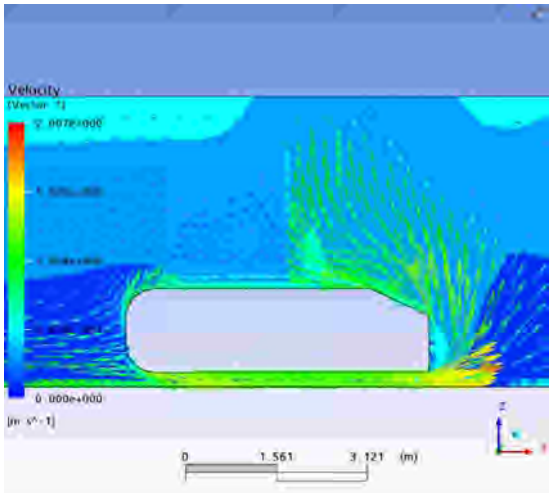


Figure 8: Revised Velocity Vectors

The revised model now illustrated a definite temperature stratification and hot air buoyancy resulting from the initiation of the fire. The initial roadway air temperature was set at 25°C. As a result of the fire on the vehicle, the thermal expansion has created a barrier to the flow direction, which in turn, results in recirculation on the upwind side. This demonstrates the formation of fire 'flashover' potential which is an extremely dangerous situation for fire fighters or mines rescue personnel who may have to fight fires to rescue trapped persons.

Temperature stratification also exacerbates problems for rescue personnel with respect to safe wearing times for rescue workers using closed circuit breathing apparatus. Wet and dry bulb temperatures are recorded by using a whirling hygrometer and safe wearing time is established from a specialised chart. Temperature readings could be taken in the lowest/coolest layer yet the upper part of the body may be subjected to a higher temperature layer. The three stratified layers shown in **Figure 7** give temperature

ranges of 25°C - 62°C for the dark blue layer, 62°C - 100°C for the mid blue layer and 100°C - 137°C for the light blue layer. The fire model was obviously not validated but has shown results which were experienced in research carried out in fire tunnels under ECSC PRO94 Fire Fighting Systems 2001 - 2004. These factors demonstrate how useful and important the visualisation concept can be, in terms of emergency intervention and rescue.

2.4.2.4 Conclusions for Task 4.2

In conclusion, the work demonstrated the validity and usefulness of computational fluid dynamics modelling and the harnessing of the visualisation of the results for emergency, evacuation and rescue use. Temperature stratification was shown to be a realistic problem in several models with equipment having surface temperatures of 50°C to 90°C in areas where the ventilation has a velocity of 1m/s or less. This phenomenon demonstrates the requirement for careful measurement techniques especially during rescue operations. Furthermore, the requirement for an accurate hand held real time BET measurement device is demonstrated. There is a requirement for the continuation and further development of these visualisation techniques and the further development of a wider range of models which represent other operational mines as a rescue aid.

2.4.3 Task 4.3 - Development of an integrated thermal risk assessment methodology

2.4.3.1 Specific Task Objectives and Planned Activities

The specific objectives of this task were:

- Development of a controlled and monitored Environmental Chamber to provide a range of simulated and controlled conditions of heat and humidity.
- Environmental Chamber trials to develop a methodology for heat tolerance screening, examine core body temperatures of rescue and other workers in exact replication of mine conditions without the requirement of working in mines which are not available to access for research purposes.
- Development of a risk management methodology which integrates information and data from each task of this work package.

Each of the specific task objectives have been investigated and achieved within the planned schedule.

2.4.3.2 Development of a Controlled and Monitored Environmental Chamber

Introduction

The development of the environmental chamber fulfilled some of the requirements of work in Tasks 4.2, 4.3 and 4.4 to provide a range of simulated and controlled conditions of heat and humidity. The chamber also provided the opportunity to develop a methodology for health screening, examine core body temperatures of rescue and other workers in exact replication of mine conditions without the requirement of working in mines which are not always available to access for research purposes. Furthermore, it is necessary to observe and monitor the condition of rescue personnel during a series of simulated rescue operations under such controlled environmental conditions.

Environment Chamber Development

The chamber consists of three separate sections - a walkway section with central vertical partition, a central confined space area and a roadway section partitioned longitudinally into four smaller roadways. The overall length of the chamber is 20m with a cross-section of 2.5m x 2.5m.

Figure 9: Completed Chamber

Figure 10: Completed Roadway Section

Figures 9 and 10 illustrate the completed chamber and a view of the completed walkway section respectively. The internal sections are linked by doors and hatches and there are four external entrances which can also be opened from the inside.

Environmental Chamber Control and Monitoring

Sensors are strategically placed within the chamber and hard wired to the control panel and production units. The temperature and humidity can be set on the control panel which, via a controller system, operate the heaters and humidifiers until the set temperature and humidity values are reached and then subsequently maintained as required. All parts of the chamber can be controlled for conditions of heat and humidity.

All parts of the chamber are monitored by eight waterproof, video cameras with DVD recording facilities. Constant viewing is available in total darkness and with conditions of thick smoke. This allows the control panel operator to view and guide operations. Communications between inside and outside are made via two way sets which are specially sheathed to withstand operations in fully saturated atmospheres.

2.4.3.3 Environment Chamber Trials

Test Methods

The trials consisted of investigating methods of monitoring the core body temperature and heart rate of rescue workers who carry out a set training procedure while wearing a BG4 rescue breathing apparatus under conditions of high heat and humidity. The test candidates were monitored for core body temperature and heart rate just prior to chamber entry and immediately on leaving the chamber.

The training test exercise consisted of a rescue team of three to four members, entering the chamber from the fresh air base, which was nominally set up just outside the main entrance to the chamber. The first action to be taken by the team was to obtain wet and dry bulb readings by using a 'whirling hygrometer' and referring to a hygrometric chart for maximum time in the given environmental condition while wearing the breathing apparatus. The test exercise continued with the team working through the chamber obstacles to locate and recover a simulation body. The team captain would then determine the time to retreat from the chamber.

A range of baseline tests were carried out to give a comparison to people subjected to the same exercise regime but in conditions of ambient atmosphere whilst wearing and not wearing breathing apparatus. The aim of these tests was to give some level of reference to a range of conditions while wearing breathing apparatus and was compared to where candidates repeated tests without breathing apparatus. The test conditions ranged from ambient temperature and humidity, moderate temperature and high humidity, to high temperature and humidity. Full time rescue workers were chosen for these tests due to

the availability of medical examination regimes, and the fact that levels of fitness must be maintained for professional purposes. A strict code of test ethics was established and maintained throughout the trials to ensure the health and safety of the test candidates.

General Overview of Baseline Test Results

Twenty-one trials were carried out with three test candidates per trial. The same three candidates were used throughout the baseline trials, clothing was also standardised to overalls and an exposure time was set at 30 minutes per trial.

The results at ambient conditions at dry bulb 21 °C and wet bulb 17°C were as follows:

- Without breathing apparatus the mean core body temperature increase was 0.3°C.
- With breathing apparatus, the mean core body temperature increase was 0.1°C

At moderate heat and high humidity (dry bulb 34°C and wet bulb 33°C):

- Without breathing apparatus the mean core body temperature increase was 0.35°C.
- With breathing apparatus, the mean core body temperature increase was 0.4°C.

At high heat and high humidity (dry bulb 40°C and wet bulb 39°C):

- Without breathing apparatus the mean core body temperature increase was 0.9°C.
- With breathing apparatus, the mean core body temperature increase was 1.1°C.

The heart rate increased with exercise and was found to be variable for each candidate throughout the range of baseline tests. The core body temperature increases gave a good indication of the increased duty on body thermo-regulation with increased heat and humidity exposure. There was a slight indication of higher level of increased core body temperature with increased age and body mass index. Although this was not unexpected, it was necessary to carry out the baseline tests to obtain results which can give an indication of trend, some level of standardisation and a method of comparison when developing testing regimes.

Test Results - Environmental Chamber Trials with Mine Personnel and Rescue Workers

Fifty-three candidates were tested, who represented full-time rescue brigadesmen, drivage workers, coal face workers, and a selection of outbye workers, officials and management. Six candidates represented non-coal-mine workers. There are several indications of potential trends based on the results obtained throughout the project time span, these can be summarised as follows:

- Heat conditioning is evident in heading workers, several comments were made that their work place environmental conditions were of a more extreme nature.
- At a dry bulb temperature 40° and saturated conditions, most of the heat conditioned workers demonstrated increased core body temperature.
- The type of clothing does appear to have an effect where wearers of overalls appear to record higher post-exercise core body temperatures of approximately 1°C.
- A greater number of miscellaneous (non-coal) mine team candidates tend to higher core body temperature increases (1°C-2°C) and increased heart rate post-exercise, compared to coal mine workers and full-time rescue workers.
- Non-face/heading workers showed generally increased post-exercise core body temperatures of 1°C-2°C.
- Generally, the full time rescue workers show less high heat tolerance than coal mine candidates employed in the coal development headings.
- There is difficulty in gauging the effect on heart rate due to the conditions and the exercise, due to the differences generally in persons and more test sets would be required.
- Weight measurements and the relationship to sweat loss showed too much variation to obtain a meaningful observation.

2.4.3.4 Development of an Integrated Thermal Risk Management Methodology

Introduction

Currently, mining legislation and associated guidance relating to acceptable limits for risk of heat stress can be considered incomplete. A number of international incidents have been identified where otherwise healthy rescue personnel have collapsed from heat strain.

Within the UK mines rescue service, a precautionary approach has been developed for assessing hot and humid conditions and for what constitutes an acceptable wearing period for using closed circuit breathing apparatus under the prevailing conditions. These "safe" wearing times were developed for use with SEFA breathing apparatus, using empirical methods, and are set out in a table which covers a range of wet bulb and dry bulb temperatures and indicates the safe wearing time for each wet and dry bulb temperature combination. The same maximum safe wearing time tables are currently being used for the Draeger BG4 breathing apparatus, which recently replaced the SEFA equipment.

Heat strain involves a complex interaction of mine environmental conditions (air temperature & humidity, thermal radiation, air velocity), metabolic heat production and the influence of clothing. Individual tolerance to thermal stressors varies widely and depends on factors such as age, fitness, acclimatisation, hydration and general health. If heat cannot be lost to the local environment at the same rate that it is produced by an individual, heat build up will occur within the body. This condition is termed heat storage. For the purpose of developing a heat management methodology, it was necessary to integrate material from literature, experience of operations, incident analysis and consideration of data obtained from the tests carried out within this project. The following points give an overview of the requirements and considerations necessary for the methodology.

Sources of Heat and Humidity

A greater understanding of the sources of heat and humidity, and their actual values in relation to all parts of every mine where rescue cover is provided, is required. Furthermore, a detailed knowledge of the mine ventilation parameters is necessary. Current rescue plans do not provide this information. Hence, the information currently available in a rescue scenario related to heat and humidity, ventilation flow and pressure detail is limited. In order to provide a more effective rapid rescue planning methodology, the following supplementary information is required:

- Wet and dry bulb readings at strategic points in districts within the mine.
- Wet and dry bulb readings outbye and inbye of locations where machinery operates and other equipment with heat outputs is sited.
- Geothermic temperature of each worked seam.
- Detail of ventilation pressure and flow at additional points within the mine areas

Risk Assessments

In order to determine the risks involved in carrying out rescue operations with breathing apparatus, in hot and humid conditions, it is firstly necessary to determine the problematic criteria involved with such a rescue. To achieve this goal, a range of questions must be considered:

- What are the heat sources, temperature distribution (wet bulb and dry bulb) and ventilation characteristics likely to be encountered for the affected part of the mine area?
- What are the distances to be traversed?
- What is the condition of the roadway to be travelled?
- What distance can be travelled, both in and back to safety, in relation to safe wearing times for given wet bulb and dry bulb temperatures?
- What is the nature of the rescue operation?
- Is there a possibility of elevated temperatures resulting from fires or an explosion?

- Is each member of the rescue team heat tolerant?
- What facilities are available for rehydration?
- Is it possible to monitor the condition of individuals during an operation?
- Is the clothing worn by rescue workers suitable?

Nature of the Emergency/Rescue Operation

Further consideration of the type of operation, in relation to the location within the mine and affected distance to be travelled whilst wearing breathing apparatus, is necessary. Previous research has shown that body pre-heating associated with walking inbye has a significant contributory effect on the core body temperature of rescue brigadesmen [Hanson and Booth 2000]. On reaching a specific location, it may be necessary to carryout work and then retreat to a safe position. Other factors may affect the safe wearing time, including the effect of exogenic and endogenic fires and the respective heat outputs. Both these parameters and the methods required to combat them, can have a drastic effect on both insitu temperature and humidity. This in turn could affect core body temperature and the heart rate of rescue workers in a very short period of time. Other factors that will place increased load on the body mechanisms include:

- Unblocking ventilation restrictions, with associated work in stagnant air with potentially increasing heat and humidity.
- Search and retrieval, including moving/cutting objects.
- Carrying injured mine personnel.
- Setting temporary support.

Heat Tolerance Screening

Although each rescue worker undergoes an annual medical examination for the determination of fitness to wear rescue breathing apparatus, this does not take into account work in conditions of high heat and humidity. For the purpose of developing a test procedure in hot and humid conditions for rescue workers whilst wearing breathing apparatus, and other tests for this project, a fully automated environmental chamber was developed. The chamber is used for heat tolerance screening while carrying out predetermined exercises in nominally set temperature and humidity.

The results are tabulated per person and can be utilised to provide an employee risk profile. It is envisaged that the test protocol may changed to offer improved methods of monitoring and exercising. It is also a consideration that this work will provide a potential methodology for heat tolerance screening during insitu mine exercises and during rescue operations. This methodology could prevent the danger of rescue workers becoming victims to an incident in hot and humid conditions. The work done to-date will offer a precursor for the methodology of such monitoring together with telemetry of data and decision making during a rescue operation.

Clothing Considerations

Underground clothing for full time rescue workers is standard. Green overalls with the rescue worker's name for identification purposes, together with underwear, safety wear and rescue apparatus. Combinations of overalls and underwear can act as a barrier to heat dissipation and reduce the effective body surface available for cooling. The tests carried out in this project have also demonstrated the effect of wearing overalls compared to shorts and T-shirts. The results obtained from these tests showed an increase in core body temperature of approximately 1°C within 15 minutes during light exercise in conditions of high heat and humidity.

Although consideration is required concerning the type of incident where wearing overalls may be considered necessary for protection, an assessment must be made regarding the relevance of clothing type

and the environment to be entered. Other considerations are to adopt the redesign of overalls with potential modifications such as removable legs and arms of overalls as prevailing conditions require. This would considerably improve the potential for body heat dissipation. Other considerations are fabric types which hold little heat and actually wick away sweat from the body. These types of fabric have been well proven for use in outdoor recreational activities and require serious consideration for application to rescue worker clothing design.

Provision of Water and Nutritional Requirements

Sweat is produced by the body to provide a mechanism of cooling by the evaporation of the sweat from the body surface. Despite this thermoregulatory protective process, profuse sweating may also result in dehydration, thus constituting a further potential threat to continued normal body function.

Current rescue breathing apparatus does not have any form of rehydration facility for the rescue worker. Once the set is worn and the closed circuit breathing facility is operated, the wearer becomes isolated from the surrounding atmosphere and cannot remove the face mask without being subjected to extreme danger. Wearing times could be considerably long periods dependent upon the prevailing conditions of heat and humidity and the type of rescue operation. Other than ensuring hydration prior to the rescue operation and rehydration on return to the safe area, no rehydration techniques are currently possible during the operation.

Research carried out within this project has resulted in the development of a rehydration system which has been incorporated within the BG4 breathing apparatus with success in maintaining core body temperature in hot and humid conditions. This methodology also has the potential to be further modified to include the addition of nutrients with such a system.

Integration of Numerical Modelling

The use of computational modelling has been demonstrated in Task 4.2. The integration of this technique would be beneficial to any heat risk management methodology. By the development of a library of models which represent real mine locations and conditions, which have been thoroughly validated, predictions can be made when conditions change. This would provide a rescue/emergency planning aid by offering likely conditions that may be encountered and therefore what actions will be required in rescue/escape/emergency situations. Once a library of models has been developed, changes can be made to a specific model which in turn can be quickly resolved to give a new estimate of a situation. Continued updates can be made as more information is obtained. The output from any model can be displayed as visual, graphical or numerical and can portray results from any part of the model in the X, Y or Z direction and with sampling points at any chosen density.

2.4.3.5 Conclusions for Task 4.3

A method for effective monitoring during training exercises in hot and humid conditions has been developed and will be utilised in all subsequent hot and humid trainings within the Environmental Chamber. The work in this chamber also serves to provide a predictive tool to ascertain expected physiological conditions of rescue and other workers subjected to actual given conditions in any mine.

The Environmental Chamber trials have shown that heat conditioning is evident in drivage and some face workers. Outbye workers and non-coalmine workers generally demonstrated the highest core body temperature increases for a given exposure period and over a range of temperature. At 40°C, all core body temperature rose at an increased rate. The results indicate a requirement to maintain safe wearing times that are currently in use for closed circuit breathing apparatus wearers. The results to-date also demonstrated a requirement for:

- Occupational health monitoring during rescue training in high heat stress conditions;
- Cooling methodologies to be examined and adopted where practicable, and

- Practical re-hydration systems to be examined and adopted.

The integrated thermal risk management methodology has been established. This work forms a basis for continued development by data acquisition from heat tolerance screening trials covering the widest possible spread of people and job categories. Combined with this, further modelling work is recommended to establish a library of models representing strategic mine locations, in as many mines as possible and covering a range of scenarios. This will serve as a predictive aid to rescue, escape or other emergency operations.

2.4.4 Task 4.4 - Practical measures to improve emergency intervention in high heat stress conditions

2.4.4.1 Specific Task Objectives and Planned Activities

This component of the work was concerned with identifying practical measures to improve emergency intervention capability in arduous climatic conditions. The work included reviewing the strategic consideration of the respective emergency response models, through to examining areas which require specific improvements, including establishment of personal and microclimate cooling facilities, improvements to protective equipment and apparatus and a subsidiary theme of transportation in emergency situations. The specific task objectives consisted of the following:

- Potential solutions to breathing apparatus visor condensation problems
- Use of personal microclimate cooling jackets
- Development of rehydration systems
- Transport systems in terms of emergency and rescue.

Each of the specific task objectives was researched and respective activities carried out in accordance with the allocated time schedule other than the work on the breathing apparatus visor condensation problems. This work was initiated at an earlier period due to the requirement of a solution to the problem which was evident in the new breathing apparatus of the UK mines rescue service operational teams.

2.4.4.2 Breathing Apparatus Visor Condensation Problems

A specific issue involving condensation problems which occurred in the mask of the breathing apparatus used by UK mines rescue workers, had been highlighted during the problem scoping phase. Previously, the UK mines rescue operational staff had used the SEFA closed circuit breathing apparatus. Since March 2004, this apparatus was replaced throughout the whole organisation with the Draeger BG4 breathing apparatus, a number of which were specifically allocated to this research study.

The BG4 mask has an inbuilt wiper blade system incorporated into the visor plate with an operating lever on the outside of the visor plate. Although the blade works well, there is still a considerable part of the visor covered with condensation, which reduces the peripheral field of view. Furthermore, if rescue workers are involved with carrying out injured workers on a stretcher, operation of the lever is not possible and full coverage with condensation occurs.

A review showed that research carried out by the UK mines rescue service to combat condensation problems with the use of cross-flow and double glazing of the face mask visor, had some limited success. The high cost of modifying and replacing all the breathing apparatus sets was impractical. Two demisting products were tested in surface trials along with the 1% soap solution. The anti-misting agents include Draeger's own product, and 'Care4Vision', which is a product used by aqualung divers.

Each of the de-misting agents worked well after a 15 minute hard march by the mask wearers with the treated visors, with very little condensation being evident after each march. No use of the manual wiper

was required. The same result was recorded for the wearers with the 1% soap solution but the solution had to be re-applied at least once during the 15 minute exercise. All non treated visors resulted in constant use of the wiper and some vision being reduced within minutes of the initiation of the exercise period. The investigation was extended to trials in a local mine where a more realistic test condition was encountered. The Draeger anti-misting agent was found to be the most effective in underground trials.

One final test was carried out in the environmental chamber where BG4 apparatus was used in training tests and 10 visors were untreated and 10 visors treated with the Draeger antifogging agent. Each wearer carried out a standard hot and humid training in the environment chamber. **Figures 11 and 12** illustrate the results of an example of a treated and untreated visor after the training exercise respectively. It can be seen that the surfactant successfully dealt with the fogging problems encountered previously. The anti fogging agent is now in standard use with the BG4 and no further work was required.

Figure 11: Treated Visor Post-Exercise

Figure 12: Untreated Visor Post-Exercise

2.4.4.3 Personal Microclimate Cooling Jackets

When considering such systems for mineworkers and, with increasing importance, for mines rescue workers, any equipment or clothing must be robust, simple and reliable. Tethered systems are less suitable for mining applications unless the wearer is static as in the case of a vehicle driver or possibly a machine operator. The use of gel packs has some potential for use by rescue workers but careful consideration must be given to any jacket/vest/holder design. The rescue worker may have to wear breathing apparatus and carry other equipment such as reviver sets, stretchers etc. For this reason there are difficulties in utilising tethered systems or anything which is bulky, heavy, complex, fragile and unreliable. In this part of the project, an investigation of the feasibility of using a simple cooling system such as the passive gel pack system was made.

Experimentation with Personal Microclimate Cooling Systems

A Draeger cooling vest (and later, for the underground trial, a Lakeland 'Ice' Jacket) was obtained to carry out a trial to compare its capability to that of the rehydration method. The trials with these jackets were limited in number and only gave an indication of functionality and effectiveness. **Figures 13 and 14** show the jacket and gel pack respectively.

Figure 13: Cooling Vest **Figure 14:** PCM (Phase Change Material) Pack

The cooling vest functions by aiding the user to reduce the increase of body temperature at work in hot environments. The system consists of packs of PCM (Phase Change Material) made up from a blend of salts (sodium sulphate decahydrate – soda crystals), which absorb the heat given off from the body. The heat absorption continues until the PCM's become more plastic and soft. It is claimed by the manufacturers that the major advantage with this cooling vest is that it gives a cooling effect at a point when the body requires it. The PCM elements recharge automatically in a temperature lower than 24°C and the process requires less time in lower temperatures. An important consideration for use by mines rescue personnel with these jackets is the weight of 2.25Kg, complete with gel packs.

Environmental Chamber Tests

Five trials were carried out in the environmental chamber under the same procedure as set out in Task 4.3. The measured wet bulb and dry bulb temperatures were 26°C and 28°C respectively which allowed an exposure time whilst wearing breathing apparatus of 83 minutes. The wearers of the Cooling Vests showed an increase in core body temperature of 0.5°C to 1.0°C with a similar increase demonstrated by the test candidates with standard BG4 apparatus and no cooling aids.

Underground Tests

Three individual mine trials were carried out, comparing core body temperature and heart rate of rescue workers carrying out a traverse of a production district over a 65 minute period. The maximum readings of wet bulb and dry bulb temperatures were 33 and 30°C respectively. Teams of rescue personnel wore a mix of standard unmodified BG4 apparatus, BG4 modified with rehydration system, and two types of personal microclimate cooling system jackets.

Core body temperature and heart rate were monitored for each team member to obtain pre-test, mid test and post test values. In all cases, the heart rate increased with exercise and the rate of increase was variable between all test candidates. The personal microclimate cooling jackets, which comprised 3 individual tests on each jacket type, showed no benefit and core body temperature increased 1 to 2°C with a similar trend to test candidates wearing standard BG4 sets. As a comparison, the core body temperature results for the rescue personnel wearing the modified BG4 sets with the rehydration system all demonstrated little or no increase. Eight results showed increases from 0 to 0.3°C and one result showed an increase of 0.6°C. These test results concur with all other results of tests carried out in the environmental chamber.

Microclimate cooling jackets were also tested for general underground use at another UK colliery which experiences problems with high heat and humidity. Test candidates reported that in general use, the jackets offered some cooling protection for between one and two hours. No core body temperatures were taken and the results were based purely on wearer observation.

There appears to be some level of protection offered by the cooling jackets especially for general underground use for short periods of time for approximately two hours. In terms of use by mines rescue personnel who wear breathing apparatus, the extra weight and non-stabilisation of core body temperature, appear to be limiting factors. Further work, in a greater range of temperatures, especially those encountered when fighting open or spontaneous combustion fires, is required. In greater localised temperatures, the short term protection may be of benefit.

2.4.4.4 Development of Rehydration Systems

Bladder Drinks System

Three BG4 face visor masks were adapted to hold a mouthpiece of a bladder rehydration unit within the mouth area of the inner mask cup. This allowed the wearer to take a drink of water, as required, throughout a test exercise.

One litre of water was placed in the pre-sterilised unit and three such units were used in this trial run. An examination of the BG4 backpack revealed that it was not possible to incorporate the bladder without potentially interfering with the workings of the apparatus. It was also decided that to suspend the bladder on the outside casing of the backpack was not practicable and would result in damage to the bladder, rendering the system unusable. The bladders, for the purposes of this trial, were suspended on the test candidates via neck straps and placed within the overalls. The trial using the bladder rehydration units was carried out in the Environmental Chamber under nominally set conditions of 40°C and 90% humidity. The wet and dry bulb temperatures were recorded at 39°C and 40°C respectively. The same test procedure of search and retrieval was used and three trials were carried out with three teams, each consisting of three personnel.

The results show that each test candidate generally maintained core body temperature with only a small fractional increase of 0°C to 0.2°C at the end of the exercise. This was compared to other test results using unmodified breathing apparatus, under the same conditions, with increases in core body temperature of 0.5°C to 1°C at the same level of exposure. The heart rate showed considerable increase as expected in carrying out an arduous exercise in high heat stress conditions.

The benefits of rehydration in high heat stress conditions are illustrated in the results of this trial. Rehydration aided the body in maintaining thermo regulation whilst wearing breathing apparatus. Each test candidate also stated that they experienced improved work capability and improved physiological condition during the test, when compared to previous exercises without drinking water. Negative feedback was given in regard of the bladder being worn on the body and was considered awkward during the exercise. No further trials with this method of rehydration were undertaken. It is considered highly important that the rescue workers are not distracted or burdened with extra and/or uncomfortable equipment.

Development of the Modified BG4 Rehydration Unit

Rehydration is possible whilst wearing breathing apparatus by providing a drinking tube within the mouth area of a face mask of the BG4. Tests carried out to-date have shown the potential of rehydration to help maintain core body temperature over an increased time period for a given condition of high heat and humidity. Equally important is the fact that the rescue worker has enough weight to carry with current equipment and the potential of having to carry/assist casualties out from a mine. Furthermore, some operations may have to be undertaken in damaged and confined areas and extra equipment, weight and/or awkward items may give rise to further problems.

It was decided to investigate the potential for using melted ice from within the heat exchanger unit of the BG4 as a method to re-hydrate the apparatus wearer. The heat exchanger unit is located within the backpack of the BG4 and has an ice block placed within it prior to any use. The heat exchanger cools the breathing gas supply to the face mask. The heat exchanger end cap has a breather hole to ensure

pressure build up does not occur during the heat exchange process and allow the end cap to 'blow-off' and lose the ice/melt water.

To obtain melted water from the heat exchanger unit, the ice block was placed directly into the heat exchanger. A tube was inserted into the heat exchanger with the end placed under the ice block to facilitate drawing off melted water on activation of the apparatus. The tube passes through the end cap via a drilled hole and held in position with a rubber seal as shown in **Figure 15**. An important factor to note at this stage is that, by establishing the drinks tube in this manner, a potential for ambient atmosphere to leak into the face mask is created. It would be necessary for the ice block to be placed in a thin collapsible membrane and placed into the heat exchanger unit, then coupled to the tube by a sealed quick-fit coupling device. This would prevent any leakage into the face mask and provide a sealed drinks system. For the purpose of the trial, the tube was fitted and tied to the outside of the right hand breathing tube of the BG4 (**Figure 16**) and the drinking end was placed in the mouth area of the inner cup of the BG4 facemask.

Figure 15: Drinks Tube in Heat Exchanger

Figure 16: Drinks System Operational

Test Method

In this trial, the methodology was changed to simulate a rescue team walking into a mine district from the shaft bottom. The distance to be simulated was 8 kilometres, representative of that in a local mine. The Environmental Chamber was set to 28°C and humidity 90%. These conditions allowed the test to be carried out over 80 minutes and be reasonably representative of the true mine condition. The main objective was to determine the condition of rescue workers travelling this distance prior to entering a production district and carrying out an operation. It must also be remembered that the rescue team need to be in a fit condition to make the return journey. It is important to attempt to predict the condition of rescue workers in the event of worst case scenarios to prevent potential casualties in real rescue operations.

Each of the tests was carried out with one team member using the modified BG4 set. The remainder of the team used standard BG4's. The weight of the test candidates was measured pre and post exercise in order to give an indication of fluid loss during such an exercise and comparison of any benefit from the rehydration unit.

Results for Modified BG4

The drinking system of the modified BG4 unit functioned extremely well and the first sip of water became available at 5 to 12 minutes from the test initiation. In each case where the test candidate utilised the modified kit, a stable core body temperature was generally maintained throughout all the tests at a range of temperatures between 28°C and 40°C nominally set dry bulb and a humidity of 90%. The pulse rate increased in all candidates during the exercise and those who wore the standard BG4 sets recorded increases in core body temperatures ranging from 0.5 to 1°C under the same test conditions.

In each case where the modified set was used, the wearer took a sip of water at regular intervals and the amount of water consumed ranged from 250 to 600 grams. Initially, the ice blocks weighed between 770 to 877 grams. **Figures 17 and 18** show examples of the remaining ice and melt water remaining in the heat exchanger on examination immediately after the exercise.

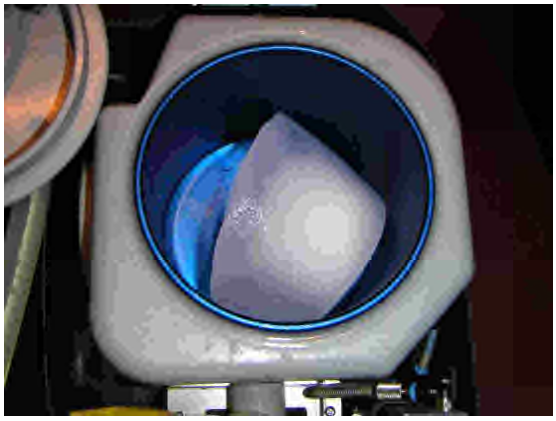


Figure 17: Heat Exchanger Post Test (1)

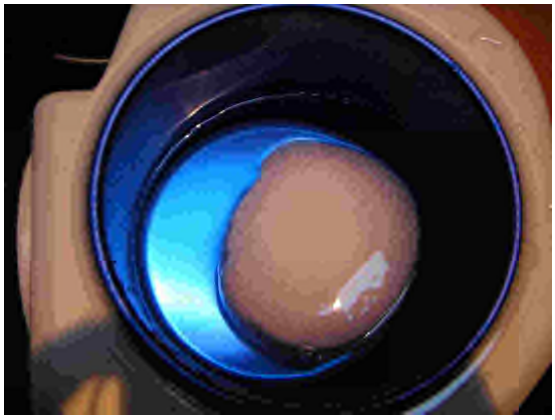


Figure 18: Heat Exchanger Post Test (2)

The process of gas cooling and drinking could have continued, if necessary, in the event of further operations being required and for the return journey. The body fluid loss did not give any trend and it would be necessary to continue monitoring this with all subsequent training exercises to obtain a larger database and provide a more informed insight into dehydration and rehydration.

Each test candidate that used the drinks system indicated that this was extremely useful and there was a vast improvement in their condition during this set of exercises when compared to previous trials and wearing standard BG4 sets. All the rescue workers were impressed with the fact that there remained ice and melted water in the heat exchanger unit at the post test point. They felt confident that with careful use of the available water, this would allow them to continue drinking over a considerable period of wearing time.

Other Considerations

It must be noted that this system would require the ice block to be placed in a sealable jacket and the tube to be strategically placed within the jacket via a sealable snap coupling. This would be necessary to prevent any ambient atmosphere entering the BG4 facemask via the drinking tube. The heat exchanger is not a sealed unit where the end cap has a breathing hole.

The drinking tube could be passed through the inside of the breathing tube and into the mask via the valve system through a purpose built tapping. This does not fall within the scope of this project but is put forward as a recommendation. The objective of the trial was to demonstrate the effectiveness and feasibility of such a drinks system being incorporated into current rescue breathing apparatus. Other modifications would be to provide a purpose made mouth piece on the drinking end of the tube with a non-return valve to aid drinking. It was also mentioned by the test candidates who used the modified BG4, that to suck on the mouth pipe and extract water from the heat exchanger was never problematic and water flowed with ease.

Underground Trials of Modified BG4 and Microclimate Cooling Jackets

Three mine trials were carried out, comparing core body temperature and heart rate of rescue workers carrying out a traverse of a production district over a 65 minute period. The maximum readings of wet

bulb and dry bulb temperatures were 33 and 30°C respectively. Teams of rescue personnel wore a combination of standard unmodified BG4 apparatus and BG4 modified sets for the trials.

Core body temperature and heart rate were monitored for each team member to obtain pre-test, mid test and post test values. In all cases, the heart rate increased with exercise and the rate of increase was variable between all test candidates. The core body temperature results for the rescue personnel wearing the modified BG4 sets with the rehydration system all demonstrated little or no increase. Eight results showed core body temperature increases from 0 to 0.3°C. Core body temperature increased 1 to 2°C for all the test candidates wearing standard BG4 sets. These test results concur with all the results of tests carried out in the environmental chamber. The test candidates wearing the modified BG4 with the rehydration system also reported major benefits in their well being, during and post exercise, by having the ability to drink water whilst wearing their breathing apparatus.

2.4.4.5 Emergency and Rescue Transportation

Available information regarding transportation during emergency and rescue operations was found to be limited. Collieries have a CEO (Colliery Emergency Organisation) scheme in place where transportation within the mine will be made available within the confines of legislation. In terms of an irrespirable atmosphere and requirement for rescue operations, the rescue teams would have to walk from a fresh air base to the specific destination while wearing breathing apparatus. The problem area is the extended lateral distances that rescue teams may have to travel prior to actual search/rescue operations. This, combined with restrictions of safe breathing apparatus wearing times under specific wet and dry bulb temperatures, is a major factor in determining the effective outcome of a rescue operation. The time constraint may be further exacerbated by slow movement due to lack of visibility. Rescue teams can be sent into an area of the mine but can they return safely to their fresh air points after their given operation.

A potential solution to this problem would be to use a form of transportation which would be dedicated to emergency, escape and rescue. There are several design considerations for such a vehicle and would require a study outside the scope of this project. Nevertheless, there is a growing need for such a facility and some possible basic design requirements are considered in this section.

Underground Emergency, Escape and Rescue Vehicles

These could be linked by using a vehicle system such as the Eimco 880 Explorer (utility vehicle). The 880 was designed to be a versatile load carrier where space to manoeuvre is a problem. A ‘skid steer’ system allows the machine to turn within its own length, which would be of great advantage in emergency/rescue situations. The vehicle is powered by a flame-proofed Perkins 1004 diesel engine which can develop 60 horse-power. The dimensions of this vehicle are 1.66 metres in width and 4.62 metres in length and **Figures 19 and 20** show the Eimco 880 with a pod and personnel carrier respectively (for up to 20 people). The vehicle can also be fitted with a crane which could be of use for certain rescue operations including roof falls and removing machinery for access etc.

Figure 19: Eimco 880 with Pod

Figure 20: 880 with Man rider

The Eimco Pod, as shown in Figure 19, could be converted into a refuge chamber or rescue pod. Such a conversion would require the pod to be sealed and self contained regarding atmosphere and other factors relating to survivability. The pod or pods, as required, could be strategically placed at any position within the mine and moved as necessary.

The use of such a pod as a rescue aid, would allow rescue teams to be transported closer to a place of operation without the need to wear their breathing apparatus. This would provide a major benefit in allowing more time to be spent on the actual rescue operation for a given time in hot and humid conditions. Less time would be lost in travelling in and out of an affected area.

Other Design Considerations

Mobile diesel powered machines are the main choice for mining companies to provide transportation solutions. The machines are generally used in the form of Free Steered Vehicles, Load Haul Dumpers and Locomotives. Although these diesel vehicles can perform for long periods of trouble free service, they have an inherent design flaw which can be problematic under certain conditions. There have been several instances where mine vehicles have continued running despite the fuel supply being shut off in the switch off procedure, due to combustible components in the local atmosphere. This can also further emphasise the problem of high heat and humidity in cases of emergency escape and rescue.

Due to the ever increasing depth and lateral distances in coal mines along with the associated increase in heat and humidity, transportation becomes increasingly important for productivity, emergency, escape and rescue. This component of the project was to consider areas for improvement for emergency intervention in high heat stress conditions. A complete design project would be required including a cost analysis, which did not fall within the scope of this project.

2.4.4.6 Conclusions and Recommendations for Task 4.4

From the work undertaken and the results obtained in this section of the project, the following conclusions may be drawn:

- Use of currently available surfactants, provide a cost effective solution, in most cases, to the problem of breathing apparatus visor fogging.
- High specification alteration would not be cost effective and would increase the cost of the current apparatus for limited effectiveness.
- Microclimate cooling jackets did not demonstrate any advantage in maintaining stability of core body temperature of breathing apparatus wearers in temperatures up to 40° C and fully saturated conditions within the environmental chamber.
- In some instances they can be advantageous for general underground use in conditions of high ambient heat and humidity for between one to two hours.
- There appears to be a difference in the level of protection offered by the cooling jackets based on work rate. The jackets would be more suited to FSV drivers rather than rescue workers who

could be further affected by the extra weight of the jackets above the apparatus to be worn and carried.

- Rehydration was shown to be the most effective method of maintaining core body temperature in arduous conditions of high heat and humidity. The prototype drinks system which was incorporated into the BG4 breathing apparatus successfully demonstrated the practicality of using melted water from the ice within the heat exchanger. The ability of the wearer to drink this water whilst wearing the apparatus in hot and humid conditions and carrying out activities stabilised the core body temperature in all tests. Physiological and psychological benefits of such rehydration systems were demonstrated.
- A potential methodology to enhance and combine transportation, emergency activities, refuge, escape and rescue in high heat stress conditions was identified. Modifications to currently available equipment could enhance the emergency preparedness of any mine, offer a high potential to save lives and protect/aid rescuers.

Recommendations for further study include the following:

- Full incorporation of drinks system into the BG4 breathing apparatus. The re-design should include a self-contained ice pack holder and water pipe to prevent possible contamination from the ambient atmosphere into the face mask via the breathing hole of the heat exchanger unit cap. Effective porting of the drinks tube through the breathing hose and valve block from the heat exchanger to the face mask.
- Further tests with the microclimate cooling jackets to verify longevity of the cooling effect in a range of duties with core body temperature measurement.
- Full development of a dedicated rescue vehicle to operate in adverse mine environmental conditions. Development would include the rescue pod and refuge chamber systems that could be utilised together with the rescue vehicle.
- Mine vehicle cab acclimatisation systems.

2.4.5 Task 4.5 - Presenting and Publishing Results - Conclusions

The principal objectives of WP4 have been to scope the challenges and specific response measures when undertaking emergency interventions, primarily rescue operations in deep, laterally extended mines. Consideration was also given to how perturbations in the mining process and mine environment affect the local mine climate and introduce short-term, excessive temperatures. The work has highlighted the thermal physiological dimension as a fundamental limitation in rescue. The ability to monitor conditions continuously has been partly addressed by the provision of an instrument design which would have the capability of monitoring effective temperature. This has been augmented by quantitative modelling, analysis and visualisation tools to define heat build-up in critical areas of mining operations. These tools have provided primary inputs into a thermal risk assessment methodology. Mechanisms to address the thermal risk posed to workers and rescuers have been appraised, concentrating on the role of rehydration of rescue staff and reducing the impacts of pre-heating by having appropriate emergency transport systems available. Cooling interventions have been touched upon, but further work is required here.

The research is anticipated to present significant opportunities for the transfer of best practice. There is a large and active network between mine operators, mines inspectorate, unions and mines rescue service providers in regard to most emergency management matters. Development of guidance codes in conjunction with direction from relevant Mines Inspectorate will be important, alongside the development of training materials, in order to ensure effective transfer of practice to coal mining and other industries. Towards these aims, the results from this project have been disseminated initially within the UK Mines Rescue Service which operates six rescue stations strategically serving the mining industry. At these establishments the project outputs will be of immediate use. Further distribution is then planned via Company intranets and web sites. Discussions are being held with Draeger who manufacture the BG4

breathing apparatus with respect to the use of the test results from the rehydration trials and possible application of the rehydration system. The possibility of reporting the research and its findings at the 3rd International Mines Rescue Conference in mid-2007 is also being actively investigated.

Regarding observations on further work, one key conclusion from the work within SAFETECH is that the assumption that long distance evacuation (and rescue) can be effected in severe conditions of heat and humidity is now being challenged around the world. The demands of providing escape respiratory protection and thermal management strategies appropriate to extreme conditions are leading to a fundamental reappraisal of escape strategy. There is a recognised research 'gap' concerning the practical limitations associated with self-escape under high physiological stress conditions. This will need to be the subject of further research. One strategic response is to implement a staged evacuation process, where underground refuges (safe havens) are a central component of mustering and rest and recovery actions, and offer a place where the mine workforce can await rescue or exchange respiratory protective devices and continue their evacuation. The successful introduction of staged evacuation strategies depends on developing a thorough understanding of risk issues, refuge respiratory support and thermal management and other support measures. The work within SAFETECH has facilitated a significant head start towards the evaluation and development of thermal management strategies which protect all personnel and against a wide range of anticipated circumstances.

2.4.6 Exploitation and impact of the research results

Actual applications

The environment chamber, which was developed to facilitate the research carried out within WP4, is situated at mines rescue station at Rawdon, UK. A second unit is under construction at the new mines rescue station at Kellingley, UK. Both Rawdon and Kellingley stations have replaced older stations and it is intended that a similar modernisation process, with inclusion of such facilities, will take place at other stations within the UK mines rescue service.

The three modified rescue breathing apparatus sets, which provided a rehydration system are used at the Rawdon rescue station for training purposes and continued monitoring of core body temperature stabilisation. Contact has been made with the manufacturer of the breathing apparatus, via the UK network and the relevant details of the research regarding the rehydration tests have been offered to them. All items of equipment such as the microclimate cooling jackets are being utilised for training purposes. The application of the chambers an other equipment developed or use within this research, is in allowing the mines rescue service to offer specialised training courses in operations in high heat stress conditions to the mining and other relevant industries.

Technical and economical potential for the use of the results

The principal objectives of WP4 have been to scope the challenges and specific response measures when undertaking emergency interventions, primarily rescue operations in deep, laterally extended mines. The work has highlighted the thermal physiological dimension as a fundamental limitation in rescue. The ability to monitor conditions continuously has been partly addressed by the provision of an instrument design which would have the capability of monitoring effective temperature. This has been augmented by quantitative modelling, analysis and visualisation tools to define heat build-up in critical areas of mining operations. These tools have provided primary inputs into a thermal risk assessment methodology. Mechanisms to address the thermal risk posed to workers and rescuers have been appraised, concentrating on the role of rehydration of rescue staff and reducing the impacts of pre-heating by having appropriate emergency transport systems available.

Dissemination of Results

The research is anticipated to present significant opportunities for the transfer of best practice. There is a large and active network between mine operators, mines inspectorate, unions and mines rescue service providers in regard to most emergency management matters. Development of guidance codes in conjunction with direction from relevant Mines Inspectorate will be important, alongside the development of training materials, in order to ensure effective transfer of practice to coal mining and other industries. Towards these aims, the results from this project have been disseminated initially within the UK Mines Rescue Service which operates six rescue stations strategically serving the mining industry. At these establishments the project outputs will be of immediate use. Further distribution is then planned via Company intranets and web sites. The reporting of the research and its findings will be done at the 32nd International Conference of Safety in Mines research Institutes, September 28-29, 2007 Beijing International Conference Centre, China.

Regarding observations on further work, one key conclusion from the work within SAFETECH is that the assumption that long distance evacuation (and rescue) can be effected in severe conditions of heat and humidity is now being challenged around the world. The demands of providing escape respiratory protection and thermal management strategies appropriate to extreme conditions are leading to a fundamental reappraisal of escape strategy. There is a recognised research 'gap' concerning the practical limitations associated with self-escape under high physiological stress conditions. This will need to be the subject of further research. One strategic response is to implement a staged evacuation process, where underground refuges (safe havens) are a central component of mustering and rest and recovery actions, and offer a place where the mine workforce can await rescue or exchange respiratory protective devices and continue their evacuation. The successful introduction of staged evacuation strategies depends on developing a thorough understanding of risk issues, refuge respiratory support and thermal management and other support measures. The work within SAFETECH has facilitated a significant head start towards the evaluation and development of thermal management strategies which protect all personnel and against a wide range of anticipated circumstances.

This extension of the research direction is being carried out at the Rawdon rescue station by the UK Health and Safety Executive and utilising the environmental chamber. Dissemination to the UK coal producers will be at the Rawdon station where demonstrations will be carried out for their representatives.

2.5 WP 5 and WP 7 - Reduction of Costs Associated to Gas Control in Caving Faces (AITEMIN and SA HVL / HUNOSA)

Introduction

AITEMIN, SA HVL and HUNOSA worked in collaboration on two components of the project. The first component was gas evacuation from stopes, together with achieving improved safety in the mining system. The second component of work concerned the viability of the developed methodology in a working mine.

One of the most successful extraction methods used in relatively thick coal layers is the so called "exploitation with sublevel caving method", used in Hullera Vasco-Leonesa (SA HVL) and HUNOSA. This method moves large coal masses in relatively short time periods.

One of the problems encountered in this exploitation method, when operating coal layers with a medium-high methane level, is the methane release from the holes caused in sublevel caving. When breaking upper strata, the migrating methane may affect adjacent drifts and workings, causing risks to workers and activity losses up to the moment when methane concentration levels are returned to within acceptable limits.

This investigation project attempts to develop a methodology for methane capture in the adjacent fractured area, located before coal exploitations.

This section of work dealt with the creation of a theoretic-practical exhibition of the methodology to follow, in order to undertake the infrastructure to produce a methane caption type installation with the aid of underground drillings. The prospective evaluation of the methane content in the coal layer was accomplished through the drillings mentioned in the first part of this section. A drilling campaign had to be designed previous to coal extraction. The drillings were adequately lined and connected, through a collector pipe, to an aspiration pump, and later on, to a section for captured gas dilution. In this section, it was necessary to avoid an explosive atmosphere (between 5 and 15% CH_4 concentrations), in exploitation areas as well as in all the ventilation circuit in the mine.

The main aim of this method was to increase the security conditions underground, especially in coal 'ragging-off' operations, because after having drained part of the methane, the potential for sudden emissions of gas will be decreased.

During the project, it was necessary to relocate the study to a different mine. The project continued in a mine belonging to HUNOSA which also exploits their coal layers using the sublevel caving method. For tunnelling these sublevels, HUNOSA uses different technologies: safety explosives, compressed air or road headers. Once the sublevel has been performed, it is blasted over the roof of the gallery (also with safety explosives or compressed air) caving the coal and winning it, before the sinking of the walls. This exploitation system has many advantages, for example, no miners with specialised skills are needed and extraction methodologies are relatively simplistic. The most significant risk is that of a coal/gas-outburst during drifting workings.

In February 2007, there were two exploitations being developed in HUNOSA and were used for the project. Both areas have similar properties, particularly being at the same depth and having similar geological faults. It was also discovered that wall pressures are very high so it was necessary to change the sequence of the exploitation, to dismiss the pressure that the exploitation causes in advance workings. The first exploitation was called "2º Cut", which is the previous exploitation and the second is called "4º Cut" which is the current exploitation. Both exploitations employ the same system (Sublevel Caving Method) with one incline and several crosscuts to access to each sublevel. These exploitations are joined by one gallery at the bottom level.

There was no doubt that each mine is a different case, determined by the geologic and tectonic conditions of the deposit. Depending of those characteristics, the emplacement and design of the gas caption installation had to be considerably modified.

Due to the requirements of the project the report corresponds with the tasks as planned at the beginning and as reflected in the contract. For each task explained, the activities are done in exploitations belonging to both SA HVL and HUNOSA.

2.5.1 Task 5.1 - Seam characterisation

2.5.1.1 Project Task Objectives:

The objective was to establish characteristics of coal seams where the research would be developed. Once the coal seam was determined, samples of coal would be taken and analysed in a suitable laboratory.

2.5.1.2 Comparison of initially planned activities and work accomplished

Parameters to be studied were defined and accepted in both mines SA HVL and HUNOSA. Parameters analysed were, among others, desorption speed, ash content and methane content.

2.5.1.3 Description of activities and discussion

Seam characterisation of the previously selected area of SA HVL was made. Tests have been accomplished in the foreseen drilling location area in order to determine characteristics of the coal and consequently the suitability of the foreseen location area. The results are shown in table 1.

PROF. (m)	DP(V_1) mm.c.a.	DP(V_{1p}) mm.c.a.	q (cm^3/35s)	Q_1 (cm^3)	V_1 (cm^3/35s/10g)	V_{1p} (cm^3/35s/10g)	GRISU (%)	Q_2 (cm^3)	Q'_3 (cm^3)	Q_3 (cm^3)	m (g)	C_B (m^3/tb)	CENIZAS (%)	C (m^3/t)
0,0	0	0	0,00	0,00	0,00	0,00	0,10	1,00	2,00	1,92	16,34	0,18	70,20	0,78
1,0	60	-	0,40	1,36	0,34	-	0,50	-	-	-	11,94	-	-	-
1,5	-	80	0,53	1,81	-	0,39	0,50	5,03	7,00	6,72	13,68	0,99	63,50	3,29
2,0	60	-	0,40	1,36	0,26	-	0,20	-	-	-	15,15	-	-	-
2,5	-	80	0,53	1,81	-	0,38	0,60	6,04	5,50	5,28	13,94	0,94	65,70	3,40
3,0	80	-	0,53	1,81	0,34	-	0,30	-	-	-	15,90	-	-	-
3,5	-	120	0,80	2,72	-	0,60	0,80	8,06	6,50	6,24	13,42	1,27	58,60	3,57
4,0	100	-	0,67	2,27	0,47	-	0,40	-	-	-	14,08	-	-	-
4,5	-	100	0,67	2,27	-	0,51	0,70	7,05	12,50	12,00	13,10	1,63	64,00	5,50
5,0	100	-	0,67	2,27	0,48	-	0,50	-	-	-	13,81	-	-	-
6,0	-	100	0,67	2,27	-	0,52	0,80	8,06	23,50	22,55	12,93	2,54	56,50	6,72
7,0	-	100	0,67	2,27	-	0,47	0,70	7,05	17,30	16,60	14,18	1,83	68,00	7,25
8,0	-	80	0,53	1,81	-	0,43	0,70	7,05	30,50	29,27	12,43	3,07	53,20	7,40
9,0	-	70	0,47	1,59	-	0,58	0,60	6,04	20,20	19,39	7,99	3,38	46,60	6,94
10,0	-	60	0,40	1,36	-	0,41	0,50	5,03	12,60	12,09	9,67	1,91	57,40	5,18

Table 1: Methane concentration in the coal layer

In the case of HUNOSA results are exposed as follows:
The values of the concentration of methane in coal seam and of the desorption speeds are measure by means of samples obtained in the degasification drillings. 21 boreholes were drilled to determine the firedamp content in the coal seam.

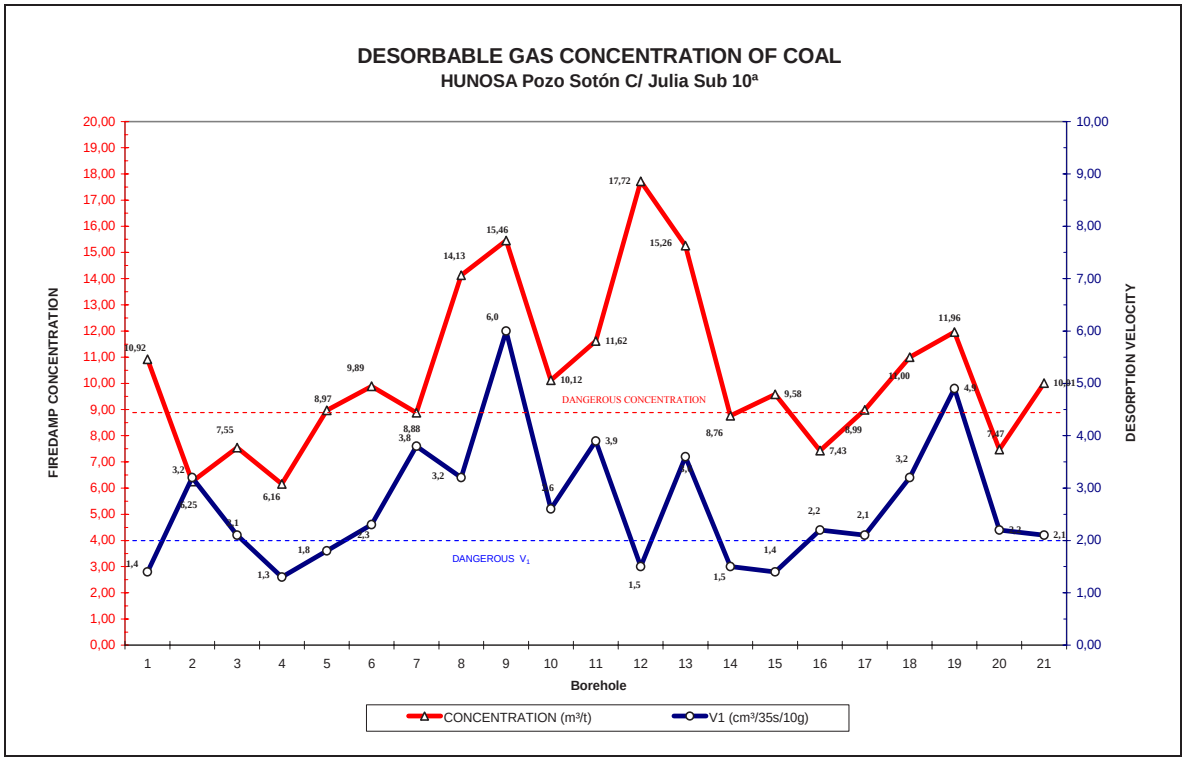


Figure 1: Firedamp concentration and V1

Figure 1 shows the average values of the concentration and of the desorption speeds obtained in the 21 different drillings of characterization of the coal seam.
The coal seam which is the focus of the research is the called "JULIA", which has an average concentration of methane of 10 m^3/ton, approximately. The desorption speed values change depending on the conditions of local working the maximum average value is of about 2,5 cm^3/35s/10g.

2.5.2 Task 5.2 - Self combustion characterisation

2.5.2.1 Project Task Objectives:

The objective of this task was to obtain samples from the coal seam where the project was to be developed and determine in the laboratory, among others; Index of self-combustion risk, Flash point and Flammability.

2.5.2.2 Comparison of initially planned activities and work accomplished

The task was not achieved in SA HVL due to having to change to a different mine. In HUNOSA it was not realized because the area selected to carry out the project was not liable to self combustion phenomena historically and it was, therefore, not considered necessary to study those characteristics.

2.5.3 Task 5.3 - Methodology design

2.5.3.1 Project Task Objectives:

To develop a methodology to capture and measure the methane extracted from a coal massif during the driving works.

2.5.3.2 Comparison of initially planned activities and work accomplished

The first step was defining the methodology to be adopted for an adequate methane capture. The research started in SA HVL facilities. Necessary steps were carried out in order to develop the infrastructure to produce a methane caption type installation with the aid of underground drillings. The prospective evaluation of the methane content in the coal layer was accomplished through a designed drilling campaign prior to coal extraction. Nevertheless, owing to some emplacement difficulties, it was necessary to previously develop a "real" test at the surface of the mine. A basic methodology has been defined, as well as the general principles for the methane caption installation: caption drillings, driving pipes, measurement modules, aspiration pump and captured gas dilution area.

Continuing difficulties were experienced in the SA HVL facilities and it was therefore necessary to change the research to another mine belonging to HUNOSA. The caption methodology developed previously was implemented in HUNOSA with some differences. There is no doubt that each mine is a different case, determined by the geologic and tectonic conditions of the deposit.

2.5.3.3 Description of activities and discussion

Stage 1: SA HVL – Definitions

In order to achieve the final aims of this WP, the following steps had to be accomplished:

1. Methane caption drillings before layer exploitation. These drillings must be adequately sealed up to the point when the exploitation influence is noticed. For these drillings, it is necessary to make a follow-up of the parameters below:
    - Gas pressure control.
    - Methane concentration control.
    - Temperature and other gases control.
2. Development of a system to capture methane from drillings, with the pressure and flood characteristics suitable for their production. The following parameters had to be analysed:
    - Induced pressure control in the drilling.
    - Aspiration flood control.
    - Aspired gas concentration control.
3. Development of a caption methodology.
4. Methodology validation.

Stage 2: SA HVL - Control and monitoring stations

a. General sump

The control stations of both caption drills will joint at the general sump where continuous monitoring will made. Different parameters will be measured in the general sump; Methane and CO concentration, Intake airflow, Intake air pressure and Temperature. The general sump is made of metallic pipe of a diameter of 152 mm (6 in).

b. Equipment

The pump was manufactured by ARZENER MASCHINENFABRIK. They make impeller type caption machines which may produce suction up to 0,5 bar. The equipment has an electrical engine with fireproof protection.

c. Dilution module

In the first stage of the project it was not considered to be profitable to extract methane for power generation purposes. The considered method was to diffuse the methane to the general ventilation return in the massif, with all aspects of safety being considered. A dilution module was designed to diffuse the methane into air whilst maintaining safe concentrations. This installation will connect directly to the

pump impulse by means of metallic pipe of 6" and a cleaned air stream will be induced by means of an electro-fan of 7,5 kW. The dilution module will be able to assure concentration of methane in exhaust air is into security margins, that's means less than 1,5 %. As a first consideration, a length of 6 m was established for the dilution module.

Stage 3: SA HVL - Tests on Control Modules

In order to prove the capability of the different sensors that constitute the degasification control and monitoring module, two laboratory tests were carried out. In this phase, all the measurements obtained by the instrumentation were checked and calibrated. For the purpose of monitoring gases, air speed and temperature, a TROLEX ENVIRO TX 6529 multisensor was acquired. The sensors in the final set-up were tested and calibrated prior to being used in actual conditions of the mine site.

Stage 4: HUNOSA - Definitions

In order to avoid coal/gas outburst it was considered necessary to drain the methane through boreholes before advancing into what is considered the danger zone. According to HUNOSA studies the dangerous content is $9m^3/t$, so it must have less than 6.75 m^3/t (75%) in the area where the advance workings are taking place. Also, the permeability value of the coal is very important for the efficiency of this method. It was necessary to study the permeability changes with the changes in rock pressures during the exploitation, to establish the best moment to drain the methane and the time required for it. The main objective for degasification is to capture methane before the advance working, for this reason it was intended to adapt the work done previously to HUNOSA's exploitation system. The following paragraphs describe the changes needed to achieve this adaptation.

Stage 5: HUNOSA - Methane caption

The caption methodology is similar to SA HVL's but with the following differences:

A. - In the SA HVL case, the methane is drained while the exploitation affects the drilled area, that is, the wall pressures generated by the exploitation cracks within the coal and the surrounded rock, then the methane can escape through these cracks and finally be drained by the boreholes. In the HUNOSA case, it was necessary to drain the methane before the exploitation and before the advance; consequently a natural permeability of the coal to let the drainage is needed.

B. - The methane must be drained in a short period of time in case of SA HVL exploitation. In the HUNOSA case, there is adequate time before the advance workings reach the protected area. Accordingly with the method implemented previously in the SA HVL case, the methodology has been divided in the following steps for HUNOSA:

1. Drilling of boreholes in two areas to protect and measure firedamp concentration in the coal, firedamp pressure, firedamp composition and natural permeability.
2. After several days measuring the pressure, the pipe ("canula") of the borehole will be open to let the firedamp drain to the ventilation circuit by its own pressure. This emission of methane must be controlled to avoid dangerous concentrations in the atmosphere. At the same time the amount of methane drained must be measured.
3. Drilling of intermediate boreholes to control the reduction of concentration in the coal.
4. In areas where the reduction of firedamp concentration is not enough, the vacuum equipment is applied to reduce this concentration to a secure level.

2.5.4 Task 5.4 - Methane capture boreholes

2.5.4.1 Project Task Objectives:

The objective of this task was to develop a methane capture borehole scheme to drain the seam working area prior to exploitation. These drillings had to be adequately sealed up to the point when the exploita-

tion influence is noticed. With respect to these drillings, it was necessary to make a follow-up study of the parameters below:

- Control of the gas pressure.
- Methane concentration control.
- Control of the temperature and other gases.

2.5.4.2 Comparison of initially planned activities and work accomplished

In the Hullera Vasco-Leonesa facilities a drilling campaign has been designed within the Santa Lucía Group, especially in the foot wall drift in the 6th level and in the main crosscut in the same plant. The drilling campaign has been divided into three consecutive phases, each corresponding to one of the drilling batteries below:

- B1: in the foot wall drift, at about 45 m from the crosscut.
- B2: in the foot wall drift, at about 90 m from the crosscut.
- B3: in the crosscut, in the intermediate section between the foot and the hanging wall drift.

In HUNOSA there are two areas to be used to develop the investigation. During the research period, several drilling tasks have been achieved in the two exploitations to measure firedamp concentration in the coal, firedamp pressure, firedamp composition and natural permeability and protect the workings. The drilling design is different in each working, as were the obtained results. In "Julia Seam" the permeability appeared to be high enough to allow the firedamp to drain without the use of vacuum equipment. An assessment of the two different drilling designs was made to evaluate the optimum design for the protection of the workings. The first design was proved in the old exploitation with excellent results and the second design was proved in the current exploitation but more time is required to evaluate this system, though initial data indicates that this system is not as good as the first one.

2.5.4.3 Description of activities and discussion

Stage 1: SA HVL - Test emplacement

With the mentioned plan, and according to the general situation and the working possibilities of the different mining groups, the Santa Lucía Group was chosen for the accomplishment of the project. Following the purposes and the working system, there must be a higher level in the working phase and a lower one with the initiated driving, where the caption drillings would be placed.

Massif 1 in Santa Lucía Group has an approximate thickness of 30 metres. The workings in the fifth level of this massif will begin shortly, with 1 meter/day maximal driving, depending on the thickness. For this reason, the footwall drift in the sixth level of massif 1 has been chosen as test development. The caption drillings would be positioned in two areas of that drift, separated one from the other, and at an adequate distance from the level crosscut. This emplacement will allow a detailed analysis of the massif degasification performance, depending on the coal drift driving, on the proximity to the higher stope and to the level stope itself, at the same time as the driving is taking place in the drilled section.

Stage 2: SA HVL - Operation Instructions

a. Drilling campaign description

A drilling campaign has been designed in M1 in Santa Lucía Group, especially in the foot wall drift in the 6th level and in the main crosscut in the same plant. The drilling campaign has been divided into three correlative time phases, each one corresponding to one of the drilling batteries below:

- B1: in the foot wall drift, at about 45 m from the crosscut.
- B2: in the foot wall drift, at about 90 m from the crosscut.
- B3: in the crosscut, in the intermediate section between the foot and the hanging wall drift.

Each drilling battery has a measuring section, independent for the desorbed gas study, before going into the caption collector that is joined to the vacuum pump.
This campaign's main purpose is to capture the gas in its highest concentration and reduce the air general body methane concentrations to safe percentages.

b. Drilling type choice

At a first stage, this task is a capture drilling operation in order to clear the layer of gas before exploiting it. A recovered gas "quality" plan has been established by using an aspiration pump and extracting methane from underground drillings. Different drilling methods which can be undertaken are:

- Long drillings.
- Foot wall or hanging wall drillings.
- Descendant drillings.
- Parallel to coal layer drillings.
- Perpendicular to coal layer drillings.

After the detailed study of the mine stratigraphy and the exploitation method in the area, as well as the study of the geotechnical conditions, it has been decided to install two drilling batteries (B1 and B2) from the foot wall drift and perpendicularly directed to the coal layer. The third battery was designed in the same way and so as to complete with the caption in the nearest area to the crosscut: B3 from that crosscut, with parallel drillings to the coal layer direction. This planning is reflected in chart 1.

This design's main purpose was to study the behaviour of the coal to be coaled between plants 5 and 6 in massif 1, as well as the methane's sequential dynamism in the coaling works. This system is positively backed by the knowledge of the methane concentration in the coal layer, obtained by the periodical test drillings undertaken in the exploitations.

Stage 3: HUNOSA - Degasification in the previous exploitation

In the old exploitation, there has been achieved a vertical drilling design. From each gallery (sublevel) vertical and downward drillings were made, as long as possible, to protect the next gallery 30m below the previous one with the objective to allow the methane drain through them.

Stage 4: HUNOSA - Degasification in the current exploitation. Level 4.1N

In this case, drilling boreholes to degas the coal face were done. This system is also effective from the point of view of safety but is difficult to introduce in the normal advance workings due to causing a reduction in the velocity of advance that is now of two meters per day and every two days it was necessary to drill six boreholes.

Stage 5: HUNOSA - Degasification in the current exploitation. Drillings in levels 4.4 and 4.5

In this exploitation, the design is different, where the degasification boreholes are drilled horizontally from a parallel gallery excavated in the surrounded rock (Esteril Planta sub 10ª), that is a part of the current infrastructure of access to the exploitable coal seam. It is excavated in parallel to the coal seam at an average distance of approximately 30 m. The horizontal drillings intersect the coal seam at the level where the gallery will become dangerous as illustrated in Figure 2. The length of the boreholes to the 4.5N sublevel changed from 43 m near the second cut incline to 10m near the fourth cut incline.

There has been established a diverse range of degasification drillings. These pass through the rock and bisect the whole coal seam at the back of the drilling. A total of six drillings to characterise the coal seam were made in this area. The drillings are named D-1, D-2, D-3, D-4, D-5 and D-6.

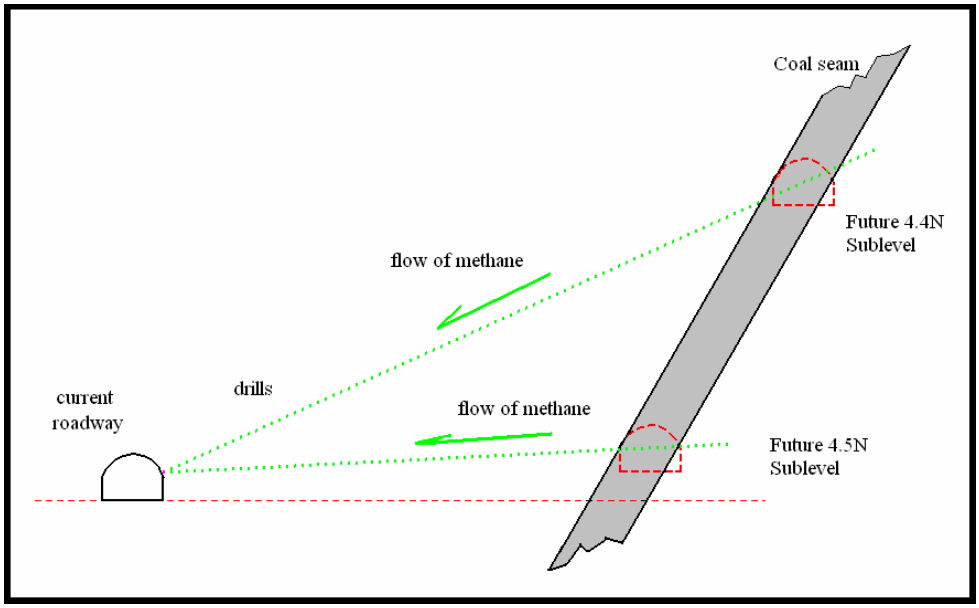


Figure 2: Boreholes drainage Sublevels 4.4 and 4.5

2.5.5 Task 5.5 - Environmental parameters

2.5.5.1 Project Task Objectives:

This task objective consisted of the control of environmental variables in the zone of capture and validation of the methodology. The following parameters were analysed; induced pressure control in the drillings, aspiration flood control and aspired gas concentration control.

2.5.5.2 Comparison of initially planned activities and work accomplished

Owing to some emplacement difficulty, which made it impossible in the past to set a position that allowed the project aims work simultaneously with mining operation, it was necessary to develop a "real" test in the open air. The test allowed to:

- Validate, in real aspiration conditions, the correct operation and registration of the control sensors, as well as the starting and shutdown phases and cut off control system relay.
- Observe a difference in the measurement of the two methanometers (maximums values of 15 % of concentration for high concentration cases). This difference may be due to the serial installation of the sensors.
- The feed cut-off to the circuit worked perfectly when the sensors were below the 20 % of methane, but it was observed that, probably due the sensors response, it is displayed with 4 or 5 seconds delay. That means that instead of cutting to 20 %, the cut is made at smaller concentration, which approaches too close to the highest limit of explosive concentration air/methane.

2.5.5.3 Description of activities and discussion

Stage 1: SA HVL - Test carried out outside the installation

The test of aspiration with methane was made in the same place where it was installed the circuit. The pertinent safety measures, which were taken during the test, can be observed in the pictures. Special attention was taken to the positioning of the methane bottles outside, with limited and restricted access, and also special attention with the canalization of gas to outside. For the different tests three Methane bottles N 25 (CH4 100%) with a pressure of 175 bar were used. That gives a volume of 11.2 N m^3, at atmospheric pressure condition.

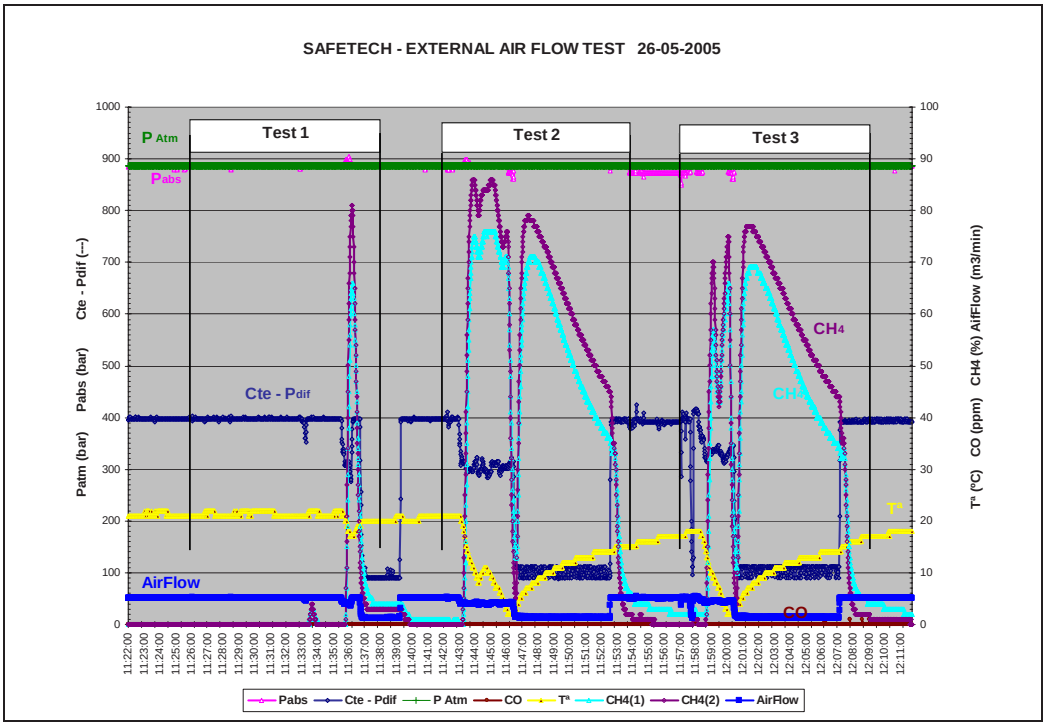


Figure 3: aspiration with methane

The pressure-reduction valve allows a volume around 30 m^3/hour with low-pressure conditions. The automatic stop of pump was programmed to work when methane concentration was equal to 20% in the circuit. This limit prevents the gas mixture reaching the higher limit of explosion. Figure 3 shows the parameters and the performances in the three tests. Test 1 starts with the methane supply from the bottles. The increase of the methane concentration is registered by two methanometres of the system. The methane supply stays during a minute, later it is closed. This causes a reduction of the percentage of methane until the automatic stop pump at 20 %. In Test 2, the methane supply stays during 3 minutes, and then it is closed. The increase and the decrease are also registered. In Test 3, the methane contribution continued during 3 minutes with a decrease of the percentage in the system because the exhaustion of a bottle.

Stage 2: HUNOSA - Degasification in the previous exploitation

After seven months, the flow of methane dropped of from 40 m^3/hour to almost zero and only 2670 m^3 of methane were drained through the new boreholes. This means that the initial boreholes had enough efficiency to protect the new sublevel. To have a better assessment of this positive effect, the methane concentration in these drills were checked and it was found that the initial concentration of 10 m^3/ton was reduced to 5 m^3/ton approximately.

The system appears to adequately protect the next sublevel, with the majority of firedamp being drained illustrated this point. Unfortunately in the production programme of the mine, the exploitation of the protected sublevel will be done at the end of 2007 so it is necessary to wait to get a real validation of this system. For this reason, a programme was initiated, to measure the quantity of methane that is exhausted by the auxiliary ventilation in the current advancing sublevel, which that time was in the 4.1N sublevel.

Stage 3: HUNOSA - Degasification in the current exploitation. Level 4.1N

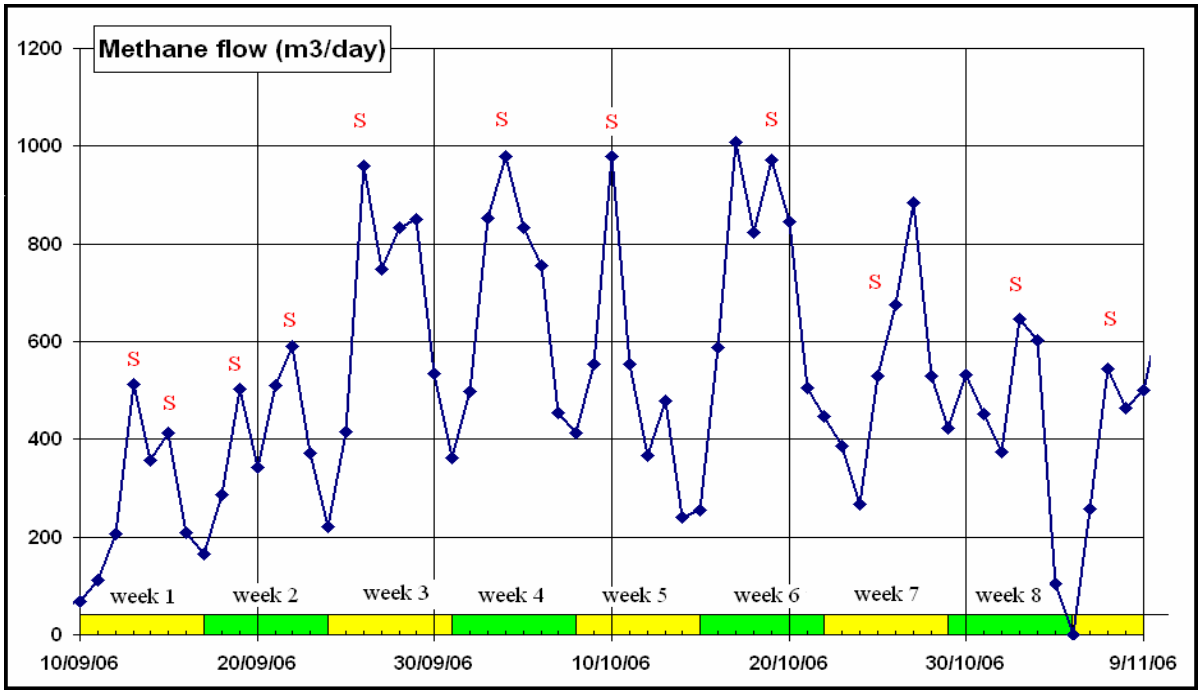


Figure 4: Methane flow per week

Figure 4 shows the methane drainage during 8 weeks, where in each week there is a cycle that shows Sundays are the days with the lowest emission of methane. It can be observed that there are peaks for those days on which the boreholes (S) were done. Work was continued by logging the amount of methane drainage so than when the protected sublevels are advanced, a check can be made on the real efficiency of this system to reduce the risk of coalburst.

Stage 4: HUNOSA - Degasification in the current exploitation. Drillings in levels 4.4 and 4.5

Figure 5 includes the average values of the concentration and of the desorption speeds obtained in the six boreholes named D-1, D-2, D-3, D-4, D-5 and D-6. In this case near the fourth cut incline it was considered that the coal seam was not affected by other exploitation because of the excessive distance, so an increased methane flow was expected with the other drilling design, but the results show some differences.

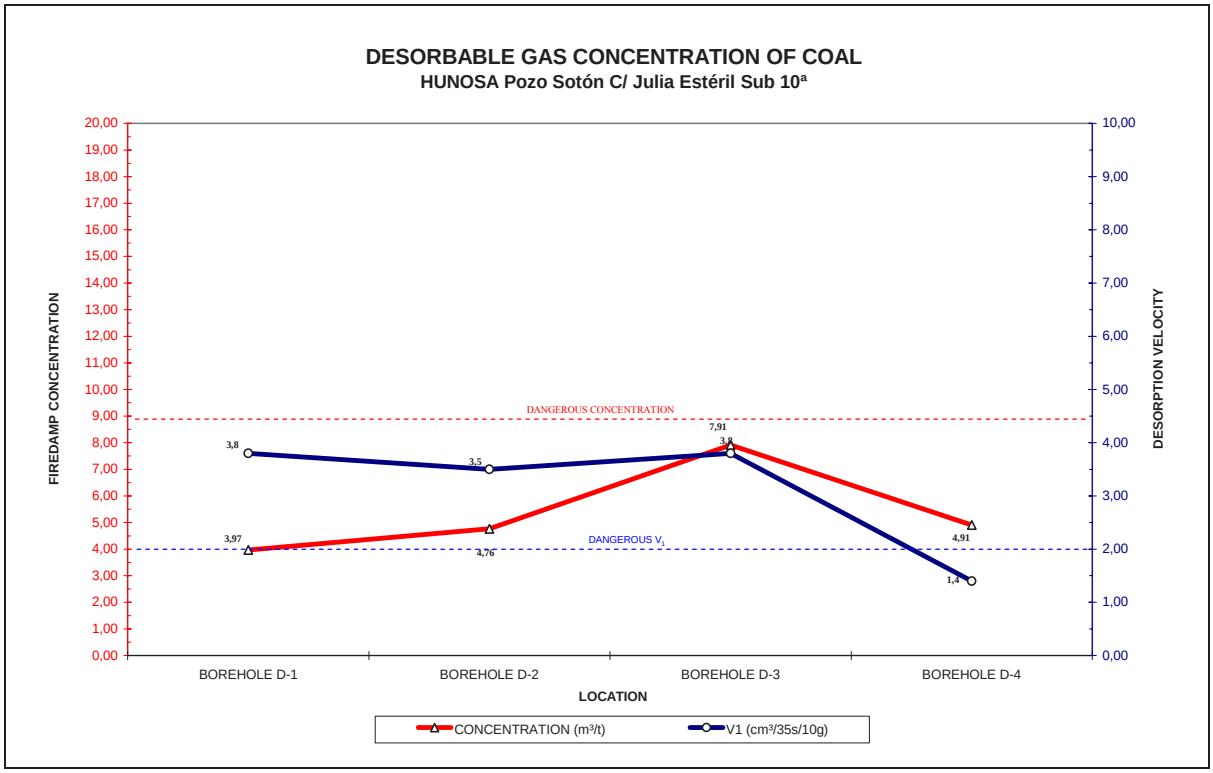


Figure 5: Firedamp concentration and V1

The methane pressure is very high, being measured to values between 11 and 21 bars. In the other drilling design the maximum value measured was 3 bars. At the same time the amount of methane drained by these boreholes is lower. An explanation to this situation was given by the modelling calculations. This modelling has been done under subcontract. The point 3.5.6.3"Wall Stress" shows the results of this part of the project.

Stage 5: HUNOSA - Analyses of samples of methane from drillings

Samples of the gas from the drillings have been collected in order to establish the permeability and pressure of the coal massif. These samples from the drillings were taken in glass bottles and analyzed in the laboratory. It was observed that samples of some drillings were contaminated with the air from the local mine environment, due to a fault in the sealing of the drillings. The average values of the gases concentrations for a firedamp caption obtained from the coal seam appear in table 2.

	N2 (%)	O2 (%)	CO2 (%)	CH4 (%)	Ethane (%)	Propane (%)	i-butane (%)	n-butane (%)
Average Values	**7,05**	**1,75**	**0,60**	**86,49**	**3,66**	**0,37**	**0,04**	**0,04**

Table 2: Average values on maximum firedamp concentrations

Stage 6: HUNOSA - Tests of gas pressure in the coal massif

Tests were carried out to determine the pressure of the methane inside the coal massif, in particular, inside the Massif 4 Sublevel 3 of the Julia seam and its evolution throughout the time. The drillings have been established from the sublevel caving working face to cutting the coal seam. The evolution of the pressure was controlled on the drillings made in the "Esteril sub 10 ª Plant". Diverse controls and

variations in their pressure were made in order to study their behaviour facing the degasification and methane caption. The results obtained are represented in the Figure 6.

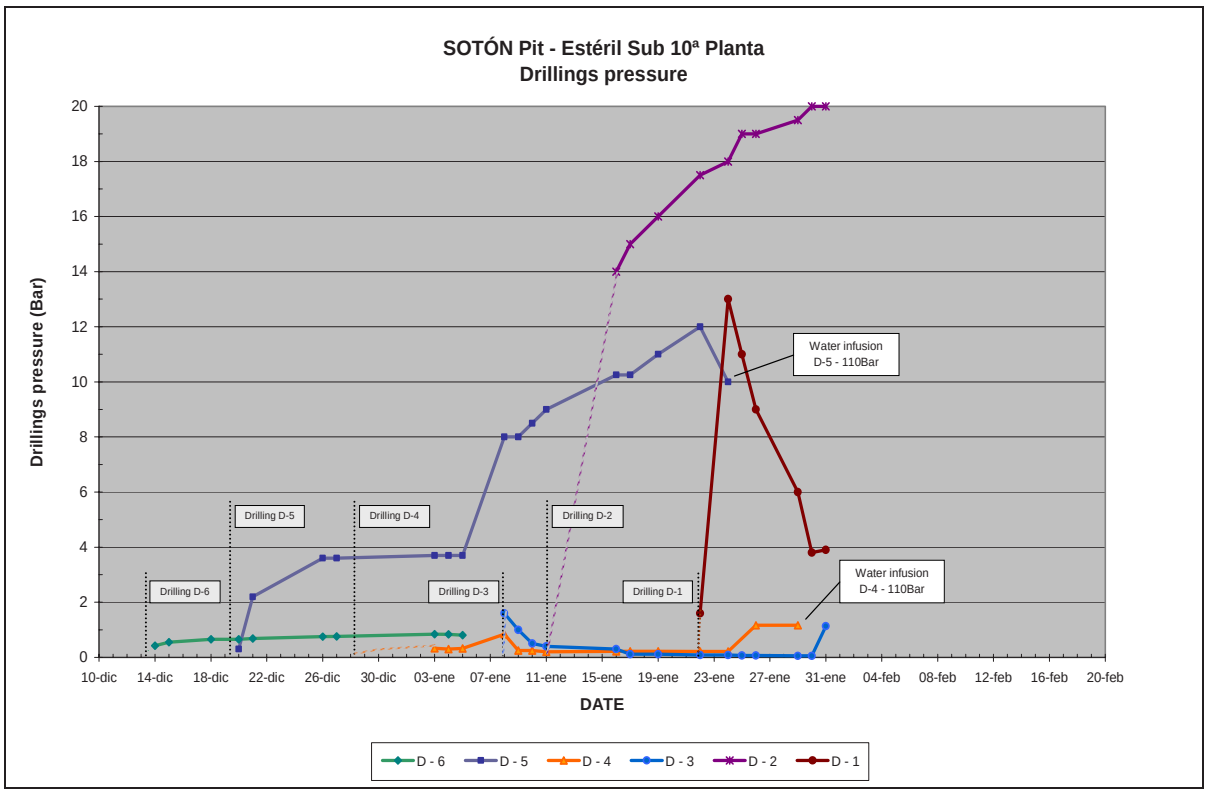


Figure 6: boreholes pressure evolution in "Estéril Sub 10ª Planta"area

2.5.6 Task 5.6 - Development capture

2.5.6.1 Project Task Objectives:

In close interrelation with the previous task, the main objective was to develop a system to capture methane from drillings, with the pressure and flood characteristics adapted to the production. The available technologies in the industry were used. Different and successive modifications were made in the design as a consequence of the results achieved and the characteristics of each exploitation as are wall stress or the massif pressure.

2.5.6.2 Comparison of initially planned activities and work accomplished

The conjunction of three factors; knowing the values that characterize the coal seams, planning adequately the exploitations of the mine, and applying forced aspiration in the drillings, will produce a high degasification in the coal seams before being won. This will increase the security conditions underground, because after having drained part of the methane, the sudden emission potential of gas will be decreased.

The stress levels play a more significant role than was first considered at the initiation of the research, and despite the difficulty in measuring the stress values, now it is possible to predict where overstress areas will occur. The exploitation sequence has been changed by making the advance working far from the winning workings. The investigation in the vertical drillings will be continued in HUNOSA, in order to find out the real usefulness of this system, to prevent coal/gas outburst and also be able to extract and utilise the methane and make a profit by using a simple extraction system.

2.5.6.3 Description of activities and discussion

Stage 1: SA HVL - Caption installation design

In figure 7, the main unities of the caption installation are shown, including caption drillings, driving pipes and measurement modules, suction pump and captured gas dilution area. In the figure 8, a borehole overview of firedamp boreholes is shown.

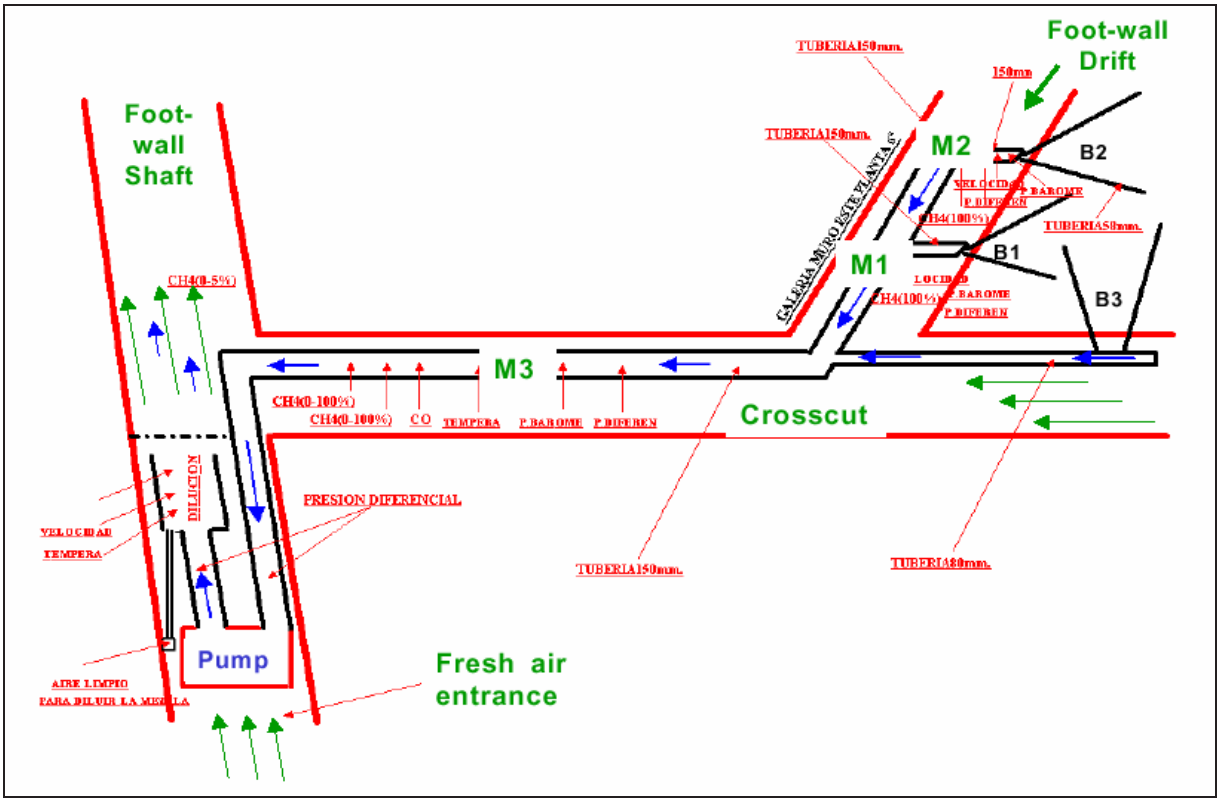


Figure 7. Caption installation design

Caption drillings

The drilling diameter was 76 mm into which was inserted a 58-mm diameter PVC inner pipe. To avoid the building of a methane air plosive mixture, a drilling " type seal" was designed. The drilling piping will consist of two pipe types: blind and filter pipes. The filter pipe is placed at the bottom of the drilling and the blind pipe is situated in the 10 next meters to the drilling mouth. The area between the pipe and the drilling wall (in the section of the 3 first meters from the drilling mouth) was sealed with expansive foam that seals and prevents the air entrance.

Up to date two tests have been made with the sealing foam, with similar conditions to the underground caption project. The results have been successful.

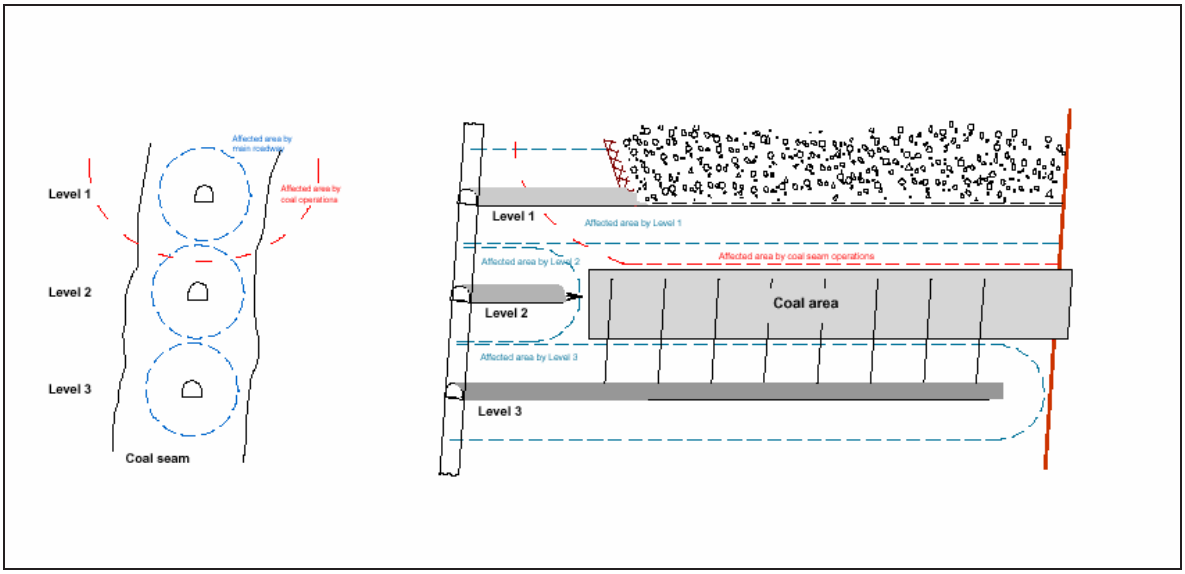


Figure 8: Investigation area. Boreholes overview

Driving pipes and measurement modules

The drillings will be connected to 50-mm diameter flexible pipes that will transport the gas up to a collector section (with 80-mm diameter), placed at the way out of each of the drilling batteries. An automatic water trap will be set in each collector. Next to each collector a straight measurement section will be located in order to manually take the flood measurements. Finally, the gas is delivered to a DN 150 collector pipe. In this pipe will be set the different devices and equipment to take the measures, continuous as well as automatic, of the various parameters.

Aspiration pump

The pump set-up consists of a compressor with rotator pistons, Aerzener make, model GM 15 L with an 11 kW engine. A gas filter is positioned at the entrance to the pump, including a trap with an automatic system to extract water. At the entrance of the pump, behind the filter and the trap and at the exit point, two explosion proof housings will be set to avoid the wave propagation in case of ignition/explosion (see photographs 1 and 2). The pump is designed to operate optimally for the foreseen gas flood of 6 m^3/min or even 11 m^3/min. For flood entrance inferior to 6 m^3/min, a by pass system will be placed in order to maintain the mentioned flood.

Dilution area

The aspired gas through the pump will be sent to the underground ventilation drift. It will be necessary to create a methane concentration dilution area to decrease this concentration to less than 1,5 %. This dilution will be made in a 1000-mm diameter 9-m pipe, where fresh air will be incorporated, blown by electro-ventilator.

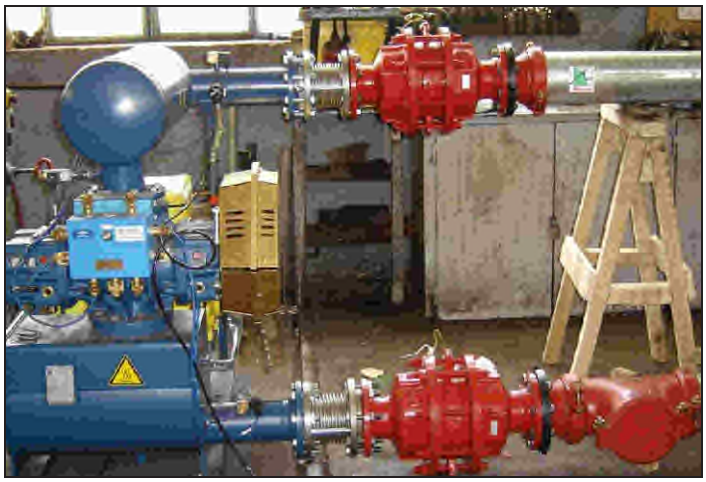


Figure 9: Caption pump

Figure 10: Test circuit detail. Pump and firebreaks

Stage 2: SA HVL - Underground emplacement mining problems

The progress of the referred massif preparation works where the investigation was foreseen has drawn negative results for the aims of the analysis, as the coal drift is now in a leaving area. The possibility of placing the test in a different mining group has been discussed after taking these circumstances into consideration. The drilling, design and operation expectances of the caption installation would equally be kept. Consequently, a new emplacement was chosen for the Flanco Sur Group, within Matallana synclinal, on 740 level. This selection means minimal design variations. Unfortunately, a spontaneous combustion fire was declared in a nearby area which affected access to the new site. The complete equipment management has been undertaken. Nevertheless, owing to some emplacement difficulty, which made it impossible to set a position that allowed the project aims to be carried out simultaneously with mining operation, it was necessary to develop a “real” test outside the mine.

Stage 3: SA HVL -Change of mine to continue the investigation

The most relevant aspects of the present difficulties in order to place the test in any of the mining groups in the HVL Company will be summarised below. The underground works cause these problems.

- Firstly, the initially foreseen test emplacement was unproductive and the mining planning advancements and drillings were no optimistic related to quality improvements and gas contents.
- After having studied the possibility of situating the investigation into other underground areas, when the test emplacement and the small design changes were already defined, a spontaneous combustion was declared in a nearby area, which affected the foreseen emplacement.

Nevertheless, and in order to advance in the methane caption methodology, work was continued on the surface to establish the settings of the components, which define the methane caption circuit. The necessary tests have been practised to verify the correct working of the installation, idle condition as well as with extracting methane. That has allowed contrasting, in real aspiration conditions, the correct working and the control sensor registration, and also the initiation and stoppage phases owing to the cut of the system control relay. The next phase required was the underground equipment installation and to proceed to develop the test during real operation. Unfortunately, the situation of the different mining groups is not compatible with the dedication needed to undertake the proceedings in the right way and consequently it was decided to finish the HVL participation in the project on 31st December 2005.

At this point of the project it was decided to continue the research with the collaboration of HUNOSA as a partner.

Stage 4: HUNOSA - Control of degasification drillings

On the drillings made from the parallel gallery in the current exploitation, from "Esteril Sub 10ª Plant" to bisect the coal seam there were achieved a series of controls and tests to determine; the pressure of the methane in the coal massif, the permeability and the degree of degasification that the drillings on the coal seam generated. The controls were as follows:

- Closing of the drillings in rock by means of cannula of rubber.
- Control of the evolution of the pressure of the drillings with fixed pressure gauge and monitoring outside in the environmental control room of the Pit.
- Opening and closing of drillings for control the evolution of the pressure and the gas captured.
- Water injection to increase the zone of influence of the drillings.

It is necessary take into account those drills that were made directly to the coal seam which is so far from the wining works that means they are not influenced for any kind of stress or pressure from surrounded area. The unique influence is from the borehole itself. It is estimated that, in the normal evolution of the mine exploitation, future works in upper sublevels would produce influences that are going to generate contributions of methane evacuation. The estimation of the degasification in one borehole obtained is of 4,7 m^3 of methane, approximately. This data applied to the characteristics of the borehole means a radius of influence about the borehole of 0,21 m. This calculus is made for the initial conditions of non mechanical influence over the zone of study.

Stage 5: HUNOSA - Wall Stress

Modelling calculations were made to study the wall stress behaviour in the current exploitation, CUT 4. As a summary, these results indicate that this area is under light overstress that causes a total reduction in the coal permeability so that the flow of methane is stopped. This is not the expected result because this area that is some distance from the exploitations (200m) and the hypothesis was that in this area the methane would drain by itself without needing vacuum equipment like in other exploitation. In this case, before the use of vacuum equipment, the mine staff decided to attempt breaking the coal using water at high pressure.

Currently, water is being injected in each drilling at a rate of 4000 litters per borehole, but it is necessary to wait till the end of 2007 to know the results. The results from the stress modelling have given an insight into the understanding of conditions that create the high risk of coal/gas outburst that resulted in four accidents in 2.4 N sublevel. It was also found that the area of exploitation can be increased, and realized how important the orders of the exploitation and left areas without caving in are. The following paragraphs show some results of this modelling with a brief explanation.

a. The stress before starting to extract the coal

Figure 9 shows the calculated stress in depth before any winning work in the exploitation. The depth is indicated from the upper level of the exploitation, the pink line indicates the natural pressure or the pressure that it would exist in case of no exploitation, but there are old upper exploitations that affect the current exploitation and this influence is indicated by the blue line. It can be observed that this influence goes more than 150 m in depth and that increases the natural stress in 3 MPa.
In this Figure it can also be observed that the stress of each sublevel, for example, the first sublevel (30 m deep) is under natural stress meaning that the workings in this sublevel are more safe that in the rest of them.

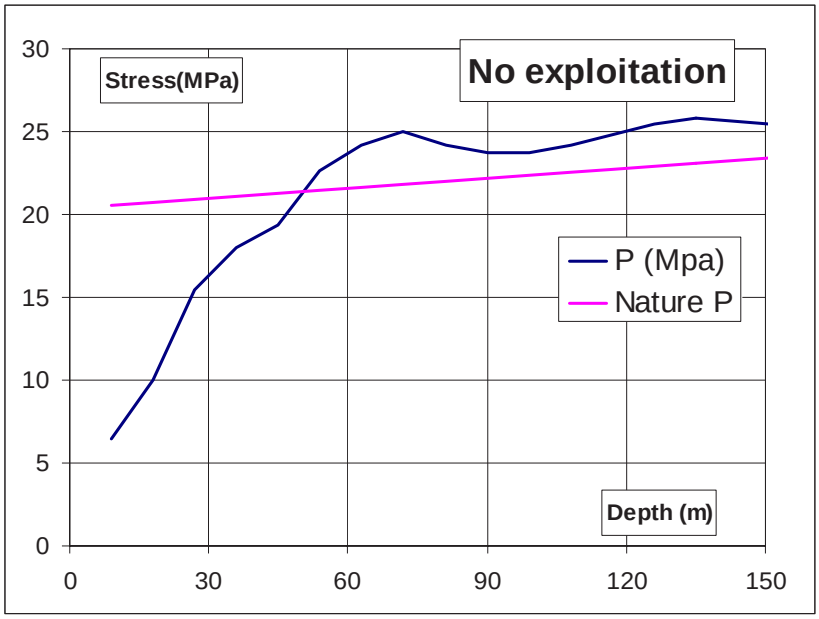


Figure 11: Stress before exploitation

b. The stress after the second sublevel

Figure 7 shows the new stresses after exploiting the first and second sublevels of the current exploitation. In this case, the origin of zero for depth is the second sublevel.
It shows that at 60m exist the most over stress area (almost 10 MPa). The advance of the fourth sublevel was done under these conditions and this explains the reason for the measured high concentrations of methane during the advance of this sublevel (with an average of 14 m^3/ton), and thus the occurrence of the four coal/gas outbursts in it.

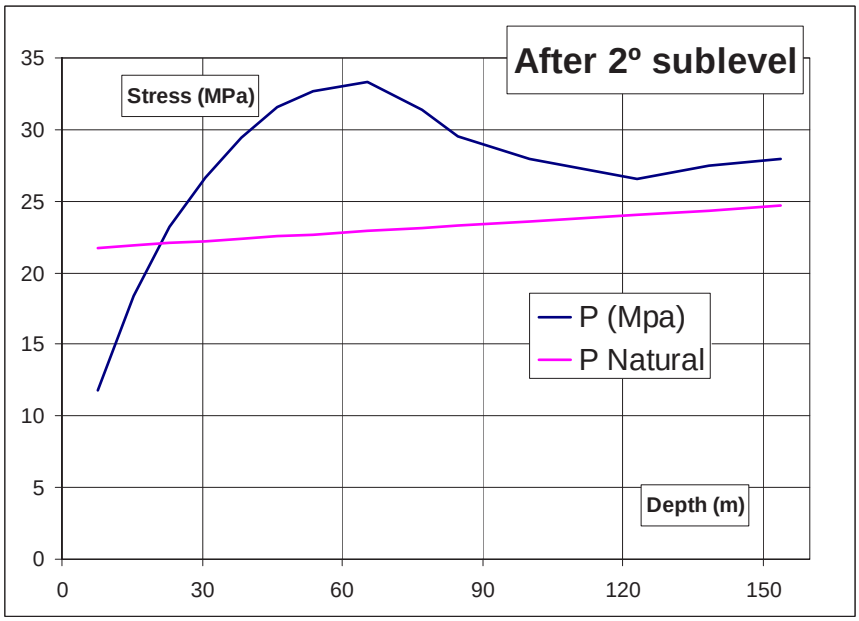


Figure 12: Stress after exploitation

From the stress modelling it can be concluded the following:

- Below no caved in spaces can have overstress up to 60 MPa, this value is very dangerous, so we must avoid it or in case of no possible, for example under faults, we must do the advance workings of the two below sublevel before creating cavity. Once this hollow caves in the stress allay to safer values.
- It is imperative to avoid crossing advance workings with caving in workings.
- Each exploitation has one area of influence by high pressures of 90 m broad, any advance working in this area must be accompanied by some degas system to protect more than 9 m in front of the face.
- Further than this 90 m broad area of strong influence there is another with light influence but enough to avoid the natural methane drainage. This area can be more than 150 m broad. In this area the coal is overstress enough to lose its permeability but no enough to be broken and let the methane out, so if we want to protect this area we must use some system that break the coal, like water, explosives, etc.

2.5.7 Task 5.7 - Industrial application

2.5.7.1 Project Task Objectives:

As a consequence of studied during the previous phases, depending on the results obtained in the research evolution and using the available means in the industry, is planned to develop a practical application in order to assess the economic utilization of the methane evacuated.

2.5.7.2 Comparison of initially planned activities and work accomplished

Due to the evolution of the underground works in both mines, it has not been possible to develop the task as initially planned. There has been estimated the energetic potential of coal seams, and is observed that exists a real possibility of an energetic utilization, improving even the mine safety conditions of the exploitations.

2.5.7.3 Description of activities and discussion

The analyses of the concentration of the gas evacuated from the drillings show that it is practically pure methane, around 90 % in volume, with small quantities of other hydrocarbons (Propane and Butane). For these conditions of concentration of combustible gases, the methane caught in the drillings presents a few characteristics similarly to a commercial Natural Gas.
From the analysis of the concentrations of the combustible gases, it is possible to estimate the energetic parameters of the methane collected in the drillings of degasification. Considering the average concentrations of the combustible gases indicated in paragraph 13.3, the energetic parameters are determined and showed in the Chart 3.

Higher heating value (HHV) (kcal/m3)	Lower heating value (LHV) (kcal/m3)	Density (kg/m3)	Relative Density	Index Woobe (kcal/m3)
8974,1	**8079,3**	**0,80**	**0,62**	**11339,9**

Table 3: Energetic characteristics of firedamp

2.5.8 Task 5.8 – Conclusions

A basic methodology has been defined, as well as the general principles of the methane caption installation: caption drillings, driving pipes, measurement modules, aspiration pump and captured gas dilution area.

The automatic control of all the process has also been designed through an advanced measurement station, with enough treatment capacity and using the flood, methane, pressure sensors and others referred to the parameters to be measured.

The research continued in HUNOSA, but the time suitable to develop all tasks initially planned for the project was not enough. Nevertheless several very important conclusions were found and ideas to be follow in the future. In this phase of the project main conclusions are as follows:

- There has been estimated the energetic potential of coal seams, and it is observed that there exists a real possibility of an energetic utilization, improving the safety conditions of the mine exploitations.
- The conjunction of three factors; knowing the values that characterize the coal seams, planning adequately the exploitations of the mine, and applying forced aspiration in the drillings, will produce a high degasification in the coal seams before being won. This will increase the security conditions underground, because after having drained part of the methane, the sudden emission potential of gas will be decreased.
- The effect of stress is very important and has more emphasis than was first thought at the initiation of the research and despite the difficulty in measurement of stress values, now it is possible to predict where overstress areas are going to happen.
- The exploitation sequence has been changed doing the advance working far from the winning workings.
- The investigation in vertical drillings is going to continue in HUNOSA, in order to find out the real usefulness of this system to prevent coal/gas outbursts and also be able to pick up the methane and make a profit of it without very complicated systems.

Once this new methodology has been implanted, important improvements are expected in underground works ventilation and decreasing flammable gas accumulations within the exploitations, which directly means increased safety for underground workers.

At this point it is necessary to certify that the experience must be considered as positive, although unfinished, owing to the absence of contrast between the methodology created in a theoretic manner and the results of a practical operation underground. Nevertheless, the underground works occasionally impose compulsory restrictions, which are contrary to the investigation development and which may be qualified as forced major reasons.

HUNOSA continues with works in order to apply the methodology started in this project. The possibility to harness the results of this research to any other mine belonging to Hullera Vasco-Leonesa or other mining company, using the sublevel caving method has good potential.

2.5.9 Exploitation and impact of the research results

Actual Applications

State of use of the equipments:

- It exists an alone pump and three units of measurement and control.
- The caption pump, which did not manage to work in real conditions (inside the mine), was acquired to the company DEILMANN-HANIEL MINING SYSTEMS, and it is deposited in SA HVL store.
- Accessories of gas caption are in SA HVL store. The equipments for gas measurement and environmental control are being used by SA HVL in the mine.
- To make boreholes in the HUNOSA mine used equipment, instruments and accessories belonging to the company.

Further measurements and developments

The research continued in HUNOSA, but the time suitable to develop all tasks initially planned for the project was not enough (practically along 2006).
The technology developed in the advance works is using actually in the advance of 4.1 sublevel North of the "Julia Seam" of HUNOSA. In case of appearing operations of coal seams with similar conditions this system would be applied.
Relating the method based on boreholes drilled from other roadway, excavated on rock, until the coal seam, the time have not allowed proving their effectiveness because the sublevel has not been concluded until now. The boreholes made with this method have allowed demonstrating that gas has been extracted but not in important amounts. Nowadays more boreholes are drilling to study the optimal distance among these boreholes and trying to define the reach that has the water injection within the coal seam.

Technical and economical potential for the use of the results

The mentioned methodology is a necessary proceeding for any other mine belonging to SA HVL, HUNOSA or other mining companies, using the sublevel caving method.

2.6 WP 6 - Improved Gas Capture, Control and Management within High Performance Workings (UNIV NOTTINGHAM)

Introduction

As the production rates obtained from modern high-performance retreat longwall faces increase, there is a need to maintain and improve the capture performance of methane drainage ranges. The boreholes often drilled above and below the caving zone are connected to a drainage range located along the return gate. To maintain the capture efficiency of the boreholes and ranges requires and understanding of both the caving mechanics and the movement of gas within, the waste and return gateroad. Thus in order to improve the safe operation and performance of the drainage range requires the enactment of a joint geotechnical and environmental study to identify the possible face end support methods that may be employed to effect good control over the caving of the waste and return gate.

The stress and fracture zones created by the extraction and caving process in the vicinity of high production retreat longwall faces strongly influence the emission of gas into these mine workings. A combination of two and three-dimensional geotechnical deformations creates the stress distributions around a working longwall coalface. The stress distribution has a major influence on the bulk volume permeability of the zone. There is a need to characterize the potential role of predominant fracture flow paths to the emission of gases through a fractured/permeable region. It is recognized that improved numerical modelling techniques are required to more accurately simulate the large-scale deformation around underground excavations, which define the potential flow paths of strata gases. To effectively model the deformation requires a determination of rock mass properties of the surrounding strata.

Recent research work carried out at the University of Nottingham has resulted in the development of an accurate rock mass classification method for UK coal measures. This rock mass classification is used to generate a range of strength and stiffness parameters necessary to create a representative geotechnical numerical model (FLAC™, Itasca 1995). Given a defined extraction sequence and geometry the FLAC™ model is able to comprehensively predict the deformations, failure zones and stresses around the excavations. These fundamental mechanical deformation properties strongly influence the permeability of the strata around a working longwall. These changes in permeability promote the release and enhanced flow of gases into the waste and mine workings. The stress-permeability behaviour of coal or Coal Measure strata is the key to the effective simulation of methane flow. It is essential that the bulk stress-permeability relationship (pre- and post-failure) be well understood before a reliable predictive model can be constructed for a longwall coalface. The presence of fractures in coal seams and the surrounding strata impacts on the rate of ingress of methane into mine workings. The flow of methane through the goaf and through the walls of the mines has been modelled using volume averages of permeability. Such averages are based on the number density, orientation and geometry of the fractures within the rock. It is proposed to conduct a local- or micro-scale study of the effects of fractures on the global permeability of the rock mass. Flow modelling will be used to model the interaction of these effects. The information obtained from these studies can be used to identify the optimum positioning of gas drainage boreholes and ranges to maximise capture. The modelling studies were validated against field data.

2.6.1 Task 1 - Geotechnical Modelling

2.6.1.1 Project Task Objectives:

- To establish geotechnical relationships between the caving characteristics of retreat longwall workings and the release and control of methane gas

2.6.1.2 Comparison of initially planned activities and work accomplished

The methodology adopted to achieve the set objectives produced the following deliverables:

- The development and application of a Coal Measure Classification (CMC) system applicable to UK coal measures, required for the empirical prediction of in-situ rock mass parameters used to produce geomechanical models of the strata surrounding UK longwall coal workings.
- The development of 2D and 3D geomechanical models to represent the geomechanical deformation of strata due to the presence of active UK retreat longwall workings. The predictions of these geomechanical models were validate against the geological, geotechnical and mining operational data obtained from five representative UK longwall panels at three collieries. Given access to the relevant geological, geotechnical and mining data bases the modelling strategy and methodology developed is equally applicable to similar retreat longwall workings practised within the EU.

2.6.1.3 Description of Activities and discussion

Stage 1:

With the assistance of the Environmental and Safety Engineers of UK Coal Ltd the University team of research scientists and engineers were able to identify the four case study mines at which longwall modelling studies could be conducted. The final selection of colliery longwall chosen for investigation was:

- Longwall Panel 43, Thoresby Colliery
- Longwall Panel DS30, Harworth Colliery
- Longwall Panel 44, Thoresby Colliery
- Longwall Panels 403 and 408, Kellingley Colliery

During the course of the execution of the research project the University Research team had to curtail a number of studies at other collieries when they were unfortunately closed by the company before the completion of the planned research programmes. The closure of these collieries during the execution of the project necessitated an application being made by the University to the EU Research Directorate to seek an extension to the duration of its RFCS research programme, to allow other mine sites to be identified at which to complete the proposed research programme. An extension was granted by the commission which permitted the successful completion of the research project documented in this report. The colliery based and head quarters UK Coal geologists, geotechnical and mining engineering teams provided to the University research team full and detailed access to the necessary geological and geotechnical logging data and detailed mining layout, support and methane drainage system specifications for each of the colliery longwall panels studied.

Stage 2: Laboratory Testing and Strata Classification

The primary aim of this research project was to develop an advanced computational modelling methodology to simulate the gas flow paths around an underground long wall panel. However, to effectively model the deformation requires a determination of rock mass properties of the surrounding strata. Recent research work carried out at the University of Nottingham has resulted in the development of an accurate rock mass classification method for UK Coal Measures [1]. This rock mass classification can be used to generate a range of strength and stiffness parameters necessary to create a representative geomechanical numerical model Given a defined extraction sequence and geometry, geomechanical modelling may be employed to predict the deformations, failure zones and stresses around the excavations.

Characteristically the UK Coal Measures comprise a distinctive sequence of strata layers known as a cyclothem [2]. The cyclothem represents the cycle of deposition on a subsiding delta and is typified by a coarsening upwards sequence above the coal seam. This forms distinctive strata units varying from

fine grained mudstones and claystones to coarse conglomerate sandstones. The sub-horizontal sedimentary layering represented by the bedding, lamination planes and shalyness imparts an engineering anisotropy upon the rock mass. The mechanical properties of the in-situ rock mass are often significantly lower than the equivalent properties determined by laboratory testing on intact samples due to the presence of structural features such as joints and bedding which are not present within the laboratory sample. However the direct measurement of the mechanical properties of the rock mass in-situ is also rarely possible due to cost and the difficulty of gaining access to the different strata horizons [3]. Rock mass classifications are now commonly used to predict the engineering performance/response of rock mass for a wide range of civil and mining engineering environments. Typically engineering rock mass classification systems quantify five or six parameters of the rock mass which are considered to have the most significant affect on the response to engineering operations. For the numerical modelling studies undertaken for this project a rock mass classification system, named the *Coal Measure Classification* (CMC) was adopted. The CMC system was developed under a previous ECSC (European Coal and Steel Community) funded project specifically for the empirical prediction of the in-situ rock mass parameters that are required in the geomechanical models of coal measure rock strata. The outputs from this classification system have been validated for use in establish empirical relationships to predict the in-situ anisotropic properties of the coal measure rocks strata around underground coal mine excavations. Furthermore a database of classification values for a wide range of UK coal measure lithologies existed which allowed the characterisation of rock strata based on lithological description alone when the required geotechnical information required for characterisation was not available. This classification system has been included within the Appendix. The outputs of the classification system were three numerical ratings reflecting; (1) the engineering competence of the rock mass as a whole, (2) the engineering competence of the rock matrix and (3) the engineering competence of the rock mass in the plane of stratification.

Stage 3: Data Analysis and Model Development.

To model the longwall panels studied at the three selected case study collieries, a series of geomechanical models were constructed using the numerical modelling software codes $FLAC^{2D}$ and $FLAC^{3D}$ [4],[5]. The models constructed were used to simulate the development of fracture planes and the stress redistributions that develop above and below the long wall panels during longwall extraction. The results of these simulations were used to interpret the major potential gas sources and pathway into the workings. The models also provided information to the mine ventilation engineers on the likely gas sources and gas flow paths into the face line areas and gate roads. This information allowed for the correct design of the orientation, length, spacing and support of the boreholes to maximise gas capture. This section describes the numerical methodology developed to simulate the fracture flow paths and gas flow around an underground long wall retreat panel during extraction. To provide information to the environmental engineers responsible for the gas drainage both two dimensional and three dimensional numerical modelling using the FLAC [4] [5] software code was undertaken. The modelling provided information on the presence of rock fractures that act as gas pathways, the horizons within the roof which were potential gas sources and on the changing stress condition around the face that affects the strata permeabilities.

Geological and Rock Strata Characterisation

To construct a numerical geotechnical model of the longwall panel, a detailed knowledge of the geological properties of the surrounding rock strata, the measured in-situ stress conditions and the longwall geometry was required. For each case study mine, the colliery geologist and geotechnical engineer supplied a range of technical data, which included detail of relevant geological borehole logs, the unconfined compressive strength and Young's modulus of the rock strata obtained from cored boreholes, the layout of the longwall panels and the orientation, spacing and length of the methane drainage boreholes. The geological borehole information obtained from the colliery engineering staff, was used to construct a simplified geological model of the strata surrounding the longwall panel under study. The mechanical properties required for the modelling such as the intact rock Young's Modulus, unconfined compressive strength and rock mass classification ratings were derived from laboratory tests synthesised with data obtained from a database of properties and rock mass ratings for similar strata types held by the Univer-

sity of Nottingham. The in-situ strength and stiffness parameters required for the constitutive models of the rock strata used in the FLAC modelling were determined using an in-house Visual Basic software application. The programme utilised a Hoek-Brown rock mass failure criterion [6] to determine the average friction and cohesion strength parameters over the range of confining stresses that were estimated to develop around the long wall panel.

Height of the cave and goaf characteristics

The height of the cave of the roof stratum above the extracted coal seam and the compaction characteristics of the broken rock within the goaf, have a significant influence on the strains, fracture patterns and redistributed stress field that occur within the rock strata as a result of long wall mining. Yavuz formulae [7] were used in this study to model the stress-strain behaviour of the goaf of the longwall panels studied and with which to calculate the degree of goaf compaction experienced.

Stage 4: Numerical Modelling

A geomechanical modelling strategy and programme was constructed using the numerically modelling software code $FLAC^{2D}$ and $FLAC^{3D}$ [4],[5]. The objective of the modelling strategy adopted was to simulate the development of fracture planes and the stress redistributions that develop around a long wall panels during longwall extraction at the case study mines. The results of these simulations were used to interpret the major potential gas sources and the resultant fracture pathways into the workings created due to the mining processes. The models also provided information to the mine ventilation engineers on the likely gas sources and gas flow paths into the face line areas and gate roads. This information allows for the correct design of the orientation, length and support of the boreholes to maximise potential gas capture.

Geotechnical Modelling of the Longwall Panel

Two-dimensional and three-dimensional geomechanical models of the strata surrounding the longwall panels being studied were constructed using the FLAC software codes. The three dimensional modelling was undertaken first, to validate the material models of the rock strata and goaf to compare the modelled stress redistributions to those predicted by established empirical methods. The three-dimensional model was also used to determine the degree of goaf compaction predicted behind the face line. This information was subsequently used to identify the relevant positions of the subsequently modelled cross plane two dimensional model sections.

$FLAC^{3D}$ three-dimensional geotechnical modelling

For each case study panel studied an initial three-dimensional computational model was constructed to represent the longwall panel using the geotechnical software code $FLAC^{3D}$ [5]. This model was developed to study the variation of the volumetric compaction across the goaf area and to identify the three-dimensional stress redistribution around the panel. The representative model geometries constructed were typically 1,000 metres long, 1,000 metres high and 600 metres wide. The model was subdivided into a mesh of sub-blocks of varying size dependant on the variation of the expected stress field. Within areas of high stress, such as within the vicinity of the longwall face line and stress abutment zone, smaller blocks of dimension 2m x 5m x 5m were employed. Whereas, in the periphery areas of the model away from the panel, larger 10m x 5m x 5m blocks were employed. The final models were formed from over 500,000 sub-blocks. The dimensions of the geometry of a representative geotechnical model are shown in Figure 4 below. The natural central symmetry of the geometry, mining layout and vertical stress distributions around each of the longwall panels studied were exploited to reduce both the model size and the computational time. The side boundaries of the $FLAC^{3D}$ model were prevented from moving in the horizontal direction. The affect of this condition was such that these boundaries acted as planes of reflective symmetry (Figure 1).

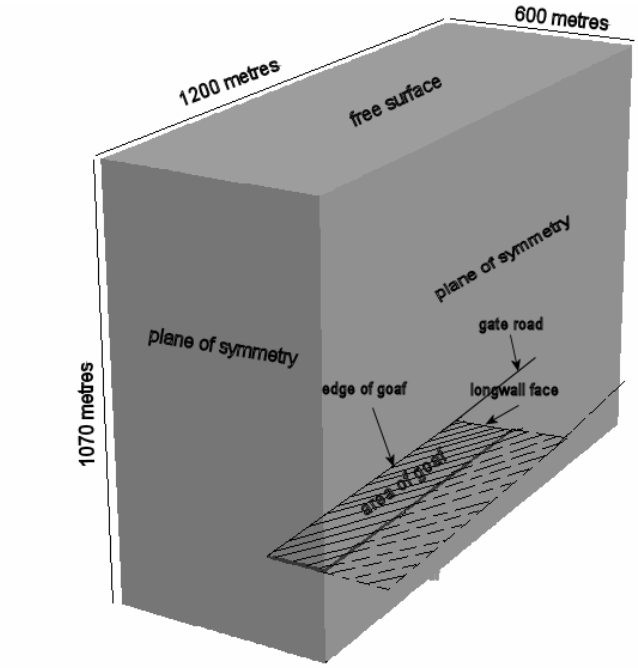


Figure 1: The geometry of a representative initial 3D geotechnical model showing position of boundaries representing planes of reflective symmetry.

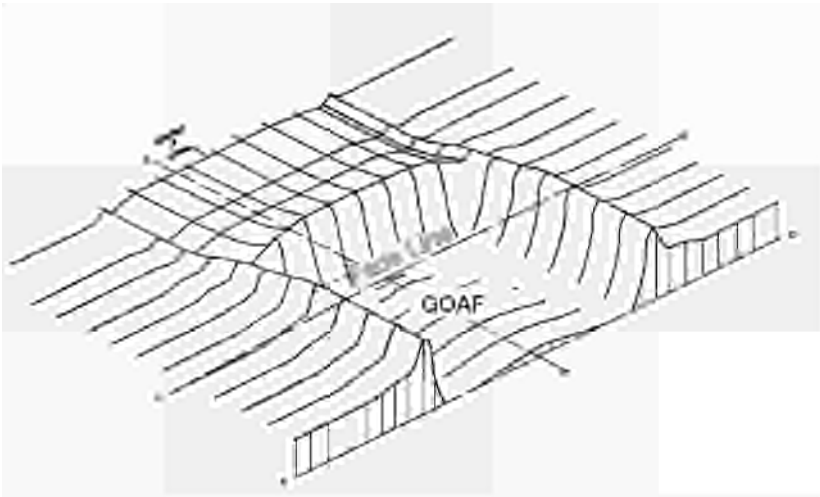


Figure 2: The vertical stress distribution (vertical axis) around a retreat longwall panel (after Peng and Chiang [8])

Initial Stress Conditions

Within the interior of the model the vertical stresses were assumed to increase linearly with depth due to the weight of the overlying rock strata. The density of the rock strata was assumed to be 2,500 kg/m$^{3.}$ The initial vertical stress varied from 26.75 MPa at the base of the model to 0 MPa at the surface, whilst at the depth of the Panel the vertical stress was predicted to be 19.25 MPa.

Within the UK coal fields the horizontal stresses are affected by tectonic strain and therefore cannot be directly determined from the lateral restraint provided by the vertical loading [9], [10]. Stress measurements taken within the Parkgate Seam at Thoresby colliery indicated a maximum horizontal stress of 17 MPa [9]. This horizontal stress measurement was a factor 0.88 lower than the predicted vertical stress. Thus in the solution of the geotechnical model the initial values of the horizontal stresses were set equal to the measured maximum vertical stress multiplied by this factor. The extraction of coal in a longwall panel leads to the redistribution of the in-situ vertical stress into the strata on the periphery of the panel.

Enhanced zones of stress relative to the in-situ stress are created close to the periphery of the extracted area and reduced stress zones are created within the worked out area (Figure 2), [8]. Immediately adjacent to the panel, the increase in vertical load leads to rock failure and yield zone development.

The validation of the FLAC^{3D} geotechnical model

A series of validation exercises of the model were undertaken to compare the modelled vertical stress redistribution around each longwall coal panel to that predicted by established empirical formulae. In the absence of the actual stress and displacement measurements taken in the rock strata around Panel 43, the predicted numerical stress distributions were compared to those determined from an established empirical model developed to estimate the vertical stress pattern around the panel proposed by Wilson [11]. To investigate the behaviour of the caving experienced behind the longwall face following extraction, the downward displacements of the roof stratum predicted by the FLAC^{3D} model along a line drawn along the central axis of the panel, were examined. The displacement of the roof stratum at a level corresponding to the top of the goaf in front of the face, across the face line and then along the central symmetry axis of the goaf was plotted as a percentage of the maximum vertical roof displacement. A comparison with actual goaf convergence confirmed the applicability of the models employed.

2.6.2 Task 2 - Stress Permeability of Coal Measure Rocks

2.6.2.1 Project Task Objektives:

- To determine the stress-permeability relationship for representative UK Coal Measure Rocks.

2.6.2.2 Comparison of initially planned activities and work accomplished

The methodology adopted to achieve the set objectives produced the following deliverables:

- A comprehensive critical review of the recent technical literature identified a suitable experimental method with which to determine the permeability of coal measure strata, by measuring the gas flow permeating intact and failed rock core samples confined within Hoek cell permeameter subjected to an increasing compression load.
- A series of compressive laboratory experiments were conducted on both intact and fractured coal measure samples obtained from a representative UK coal mine longwall operation. From an analysis of the experimental data it was concluded that :
    - The permeability of the intact siltstones was lower than the detection limit of the flow meters, giving an intact permeability of less than 1×10^{-19} m^2.
    - An empirical relationship was obtained to represent the variation of the strata permeability with stress.

2.6.2.3 Description of activities and discussion

The stress-permeability behaviour of coal or coal measure strata is the key to the effective simulation of methane flow. It is essential that the bulk stress-permeability relationship (pre- and post-failure) be well understood before a reliable predictive model can be constructed for a longwall coalface. Research into the permeability characteristics of UK Coal Measure rock strata has concluded that as coal measure rock fails and fracture occurs, the permeability rises rapidly. These results suggest that fracture permeability may be a dominant factor in determining the gas flow paths into working areas from adjacent coal seams or gas bearing horizons [12],[13]. For this study, to develop a relationship between stress and stratum permeability, a series of laboratory experiments using a test method devised by Durucan [14] were undertaken on both intact and fractured coal measure siltstone samples. The experiments were undertaken on five samples of a fine grained coal measure siltstone that can be considered typical of the major proportion of the coal measure sequence around 43s longwall panel of Thoresby Colliery. The siltstone test samples were prepared into cylindrical cores 37.5mm in diameter and 75 mm long. In turn the prepared core samples were placed into a Hoek-Cell which also acted as a permeameter. The sample was supported in a stiff press between steel perforated end platens. Nitrogen gas was injected into the top of the sample at a pressure of 0.276 MPa. The volumetric flow rate of gas through the sample was

monitored during the test using gas flow meters connected via a plastic tube to the bottom platen. To determine the intact permeability of the siltstone an initial confining stress of 0.5 MPa was applied to the sample and then the sample was axially loaded such that then axial stress was equal to three times the confining stress until a final confining stress of 8 MPa was obtained. As each increment of stress ratio was increased, the loading on the stiff press was put on hold to allow a flow reading to be taken.

To investigate the permeability of fractured siltstone the final increment of confining stress on the intact sample was reduced to 0.5 MPa and the axially stress increased until failure of the sample occurred. The process of reapplication of the stress ratio starting from 0.5 MPa to 8 MPa confining stress at 0.5 MPa increments whilst increasing the axial stress in a ratio of 3 times the confining stress was undertaken with flow readings being taken at each increment in confining stress.

The intrinsic permeability at each confining pressure was then calculated using Equation 1 below.

$$k_i = \frac{Q.\mu.L}{A.\Delta p} \qquad [1]$$

where k_i = intrinsic permeability (m^2)
Q=volumetric flow rate (m^3/s)
μ = viscosity ($N.sec/m^2$)
A=cross sectional area (m^2)
ΔP=pressure drop over length of sample (N/m^2)
L=length of sample (m)

It was found in all cases that the permeability of the intact siltstones was lower than the detection limit of the flow meters, giving an intact permeability of less than 1×10^{-19} m^2 From a preliminary analysis of the experimental data it was concluded that the gas permeability of the sheared samples decreases rapidly as the confining pressure increases over the range 0.5 to 4 MPa.
A further analysis of the data was undertaken to develop a functional material parameter that can be coded into the numerical models to predict the distribution of permeability of the fracture planes that develop around the panel. The functional relationship that was found to provide the best fit to the test data is represented in Equation 2 below:

$$K(\sigma_1, \sigma_2) = K_1 . \left(\frac{\sigma_1 + \sigma_3}{2} \right)^m \qquad [2]$$

where,
K = the intrinsic permeability (m^2)
σ_1 and σ_3 = the minimum and maximum confining stresses (MPa)
k_1 = the intrinsic permeability (m^2) when ($\sigma 1 + \sigma 3$)/2 =1 MPa
m = a material parameter found by regression analysis

2.6.3 Task 3 - Fracture Flow Modelling

2.6.3.1 Project Task Objectives:

- To identify the bulk volume permeability of the rock mass surrounding a longwall working panel created by the location and geometry of the stress patterns and fractures created within the rock mass due to mining. The presence of fractures in coal seams and the surrounding strata impacts on the rate of ingress of methane into mine workings.
- To identify improved tailgate support methods with which to maintain the life and capture efficiency of the drainage boreholes driven above and below the seam being extracted behind the advancing faceline.

2.6.3.2 Comparison of initially planned activities and work accomplished

The methodology adopted to achieve the set objectives produced the following deliverables:
A generic modelling strategy was developed that consists of the following four sequential stages:

- A series of 2D geotechnical models were constructed across parallel planes set at regular distances behind the face line. These models were able to show the re-distribution of the in-situ stress field, the extent of broken ground and the development of major fractures around a long wall panel occur progressively with the extraction of the coal. The permeability of a rock field to gas flow is strongly dependant on the state of compression of the rock strata. Consequently, the strata permeability and gas flow regime varies behind the face line as the goaf compacts under the force of the bridging beds lowering on the goaf. Three stages of goaf compaction were identified at specific distances behind the faceline:
    - Model 1: uncompacted goaf
    - Model 2: partially compacted goaf
    - Model 3: fully compacted goaf
- Using the range of coal measure strata permeability results obtained from the experiments performed as part of Task 2, combined with the geological section data, and the 2D geotechnical compaction data obtained from above, a rock permeability map was created across each of the three 2D geotechnical sections. Each of these planes in turn defines three zones of permeability extending from these planes towards a vertical plane through the face line.
- Each of the coal seams in geological section was modelled to be potential gas reservoirs and was charged with a pressure source of gas. A Darcian gas flow model, with the coal seams as the gas sources, was solved across each of the 2D permeability planes. The solutions predicted the path of gas flow from these reservoirs along both preferential interconnected fracture paths and through permeable broken rock mass to the low pressure mine workings. The location of the coal bearing seams and angle and length of the connected fracture paths assisted in the design of the borehole length and inclination to intersect the predicted gas flow above and below the goaf.
- The project developed a novel mechanical model to predict the stability of a borehole subjected to the varying shear force applied by the collapsing strata to the borehole as the face retreats. The predicted neck off of boreholes behind the face line was confirmed by data collected from the case study mines.
- From both field measurements and geotechnical model analysis it was shown that the maintenance of the strength of the roof support in the tailgate road immediately behind the face line greatly influenced the longevity and efficiency of the boreholes in this area. It was demonstrated that for a weak siltstone and mudstone roofs the installation of both rib side props and central gate cribs could increase the distance of effective drainage of the boreholes by 50%.

2.6.3.3 Description of activities and discussion

FLAC^{2D} two dimensional geotechnical modelling

As an extension to the three-dimensional numerical models, a series of two-dimensional numerical models were constructed to investigate the plane strain problems associated with each of the long wall extraction systems studied. The two-dimensional models represent vertical sections, parallel to the face line, passing through the goaf and gate roads. As with the solution of any numerical technique the accuracy of the results obtained depend upon the detail and refinement of the grid used to represent the physical system. In general, a finer grid leads to more accurate results. Typical complete model dimensions extended for a horizontal distance of 1,000 metres and a vertical height of 900 metres and consisted of 280,000 zones.

The modelling methodology adopted

The modelling methodology adopted in the FLAC^{2D} modelling is illustrated by the flow chart in Figure 9. The dark grey shaded boxes in the Figure represent the acquisition of the model input data. The pale

grey shaded boxes represent the modelling processes whilst the un-shaded boxes represent the iterative model optimisation procedures and model solution output. The re-distribution of the in-situ stress field, the extent of broken ground and the development of major fractures around a long wall panel occur progressively with the extraction of the coal [8]. The permeability of a rock field to gas flow is strongly dependant on the state of compression of the rock strata. Consequently, the strata permeability and gas flow regime varies behind the face line as the goaf compacts under the force of the bridging beds lowering on the goaf. Thus, the ability to obtain effective gas capture from drainage boreholes located above and below the goaf changes progressively along the gate road with increasing distance behind the face line. To investigate these changes, numerical models were constructed to represent the three stages of goaf compaction represented by the following regions:

- Model 1: uncompacted goaf
- Model 2: partially compacted goaf
- Model 3: fully compacted goaf

Analysis of the solutions to the 2D FLAC^{2D} geotechnical models

Model 1: Immediately behind the face line.
After an initialisation of the in-situ stress field, the coal and immediate roof of the panel were removed and replaced by a goaf extending to a height of seven metres above the top of the coal seam. This was to simulate the removal of the coal seam due to longwall extraction. Time stepping mechanical computations were undertaken until a small amount of vertical displacement of the bridging beds downwards onto the goaf had occurred. The vertical displacement of the bridging beds was monitored during the modelling by a history log of the vertical displacement of the stratum directly overlying the goaf in the centre of the panel. The regions around each working panel where the stress confinement was predicted to be less than 3 MPa were identified. This limit value was chosen, since the laboratory test work on the fractured coal measure rocks indicated that the permeability was substantially reduced in areas where the confining stress was greater than 3 MPa [45], [46]. The models predicted that fully developed shear planes would be limited to the edges of the goaf and to small localised zones around the roadways. Due to the dilation of the planes that would occur during such shearing, these planes provide potential pathways for gas flow from surrounding gas bearing strata into the mine roadways. From an analysis of the laboratory shear box test data obtained by Denby [46] a shear strain increment of greater than 10mm per metre was chosen to represent a fully developed shear plane.

Model 2: Partially Compacted Goaf
Model 1 was then restored and the time stepping computations were continued until the monitored vertical roof displacement represented 80% of the vertical roof displacement. An analysis of the predicted regions across this plane where stress confinement is less than 3 MPa was performed. It was concluded in all of the four colliery longwall panels studied that the regions of low confining stress, and therefore enhanced strata permeabilities, had increased significantly extending in a broad region from within the goaf into the overlying roof beds. An analysis of these models also concluded that steeply dipping shear planes would originate from the roadways and extend over the goaf to a height of over 30 metres. Horizontal shear planes are also predicted to occur at several distinct horizons in the regions around the edges of the panels. Several of these horizons were present at a height greater than 100 metres. However, it was concluded that many of these shear horizons would be unconnected to any potential continuous flow path way for gas into the active mine workings.

Model 3: Fully Compacted Goaf
Finally, the computational models were solved until a state of static equilibrium was obtained. The monitored vertical roof displacement until it achieved 100% of vertical displacement of the bridging beds and full goaf compaction. In the FLAC^{3D} model 90% goaf compaction occurred at a distance behind the face line of greater than 120 metres (Figure 7) and so Model 3 was considered to represent a distance behind the face line of greater than 120 metres. Once again the regions around panel across this 2D section behind the face where the stress confinement was predicted to be less than 3 MPa

were investigated. It was concluded that there was no longer a stress of lower than 3 MPa present immediately above the goaf. This is due to stress build up in the roof stratum as the goaf resists the downward movement of the bridging beds. The modelling also indicated that no further development of fracture planes compared to that for Model 2 had occurred.

To produce an effective gas drainage system for Panel 43, the drainage boreholes drilled into the roof and floor of the tailgate have to intercept the main gas flow pathways from the coal seams and strata that act as potential gas sources. To predict the position of these pathways and sources flow modelling was undertaken using an adaptation of the fluid flow facility present in the FLAC code. The FLAC^{2D} geomechanical modelling indicated that the conditions most conducive to gas flow were present in Model 2, representing an approximate distance behind the face line of 65 metres. This model was therefore used for the gas flow modelling.

Overview of fluid flow modelling using the FLAC computational code

FLAC models the flow of fluid through a permeable solid, such as soil. The flow modelling may be done by itself, independent of the usual mechanical calculation of *FLAC*, or it may be done in parallel with the mechanical modelling, so as to capture the effects of fluid/solid interaction. The FLAC flow code adopts a finite difference computational method of solving a Darcian flow problem. The numerical methodology computes flow through a semi-porous medium with the rate of flow directly proportional to the strata permeability and the fluid pressure gradient.

Strata Permeability

Zones of different strata permeabilities develop around a long wall panel as a result of the fracturing of the rock strata and the redistribution of the in-situ stress field (Figure 13). From an analysis of the results of the Geomechanical modelling described in Section 5 it was concluded that there were four characteristic regions of different permeability, namely; zone of intact rock strata, zone of broken rock strata, zone of fully developed rock fractures, zone of collapsed goaf. The permeability, k, required by FLAC is the mobility coefficient (coefficient of the pore pressure term in Darcy's Law). The relation between hydraulic conductivity, k_h (m/s), commonly used when Darcy's Law is expressed in terms of head, and permeability, k, (m^2/pa.s) is given in Equation 3:

$$k = \frac{k_h}{g.\rho_w} \qquad [3]$$

where g = gravity (m/s^2)
ρ_w = fluid mass density kg/m^3
The property of "intrinsic permeability" k_i (m^2) is related to k and k_h as follows:

$$k_i = \mu.k$$

where k_i = intrinsic permeability (m^2)
μ = viscosity (N.s/m^2)
k= mobility coefficient

Permeability of intact coal measure rock strata

In the laboratory test work described in Task 2 the intrinsic permeability calculated for the intact samples was below the detection limit of the flow manometers, which gives a value for the intrinsic permeability of the intact rock of less than 1 x 10^{-19} m^2. This value was adopted for all regions in the model where no yield had occurred. The mobility coefficient was then calculated using Equation 3 with a gas viscosity of 1.75 x 10^{-5} N.s/m^2

Permeability of the Goaf

Laboratory testing of broken coal measure rocks in a permeameter under different degrees of compaction has been undertaken by Jozefowicz [13]. He tested eight samples consisting of four sandstones, three shales and one gritstone. For purpose of determining the variation in strata permeability around the panel a general relationship between state of compaction and permeability was developed using this data by averaging the data and the fitting a third degree polynomial curve. The intrinsic permeability of the goaf was related to the volumetric strain within the goaf using the following relationship:

$$k_{i\,goaf} = -4\times10^{-16}.\varepsilon_{vol}{}^{3} - 6\times10^{-15}.\varepsilon_{vol}{}^{2} - 7\times10^{-14}.\varepsilon_{vol} + 1\times10^{-11} \qquad [4]$$

where $k_{i\,goaf}$ = intrinsic permeability of the goaf m^2
ε_{vol} = volumetric strain in the goaf

The mobility coefficient required in the FLAC modelling was then calculated using Equation 3 with a gas viscosity of 1.75 x 10^{-5} N.s/m^2

Permeability of the fracture planes

The permeability of the fracture zones was determined using Equation 8. From consideration of the permeability testing described in Task 2 an *m* value of -0.8616 and k_1 permeability of 2.613 x 10^{-13} m^2 was used within the Equation. Again the mobility coefficient, *k*, was calculated using Equation 3 with a gas viscosity of 1.75 x 10^{-5} N.s/m^2. This viscosity was assumed for both air and the methane gas within the rock strata.

Permeability of the yielded rock

This is defined in the models as the regions where the rock has undergone yield but without a fully developed shear plane defined by a total shear strain of 10mm/m. It was considered that the permeability in these regions would have an upper bound represented by the permeability of a fracture plane. To estimate the permeability of the yielded strata the stress dependant permeability for fractured rock (Equation 4) was reduced by a factor determined by dividing the shear strain by a shear strain value of 10mm/m. The functions that predicted the stress dependant permeability were developed for each of these four regions and coded into the $FLAC^{2D}$ model to predict the distribution of permeability around the Panel. The predicted permeabilities for the goaf and rock strata for Model 2 are shown in Figure 3 below.

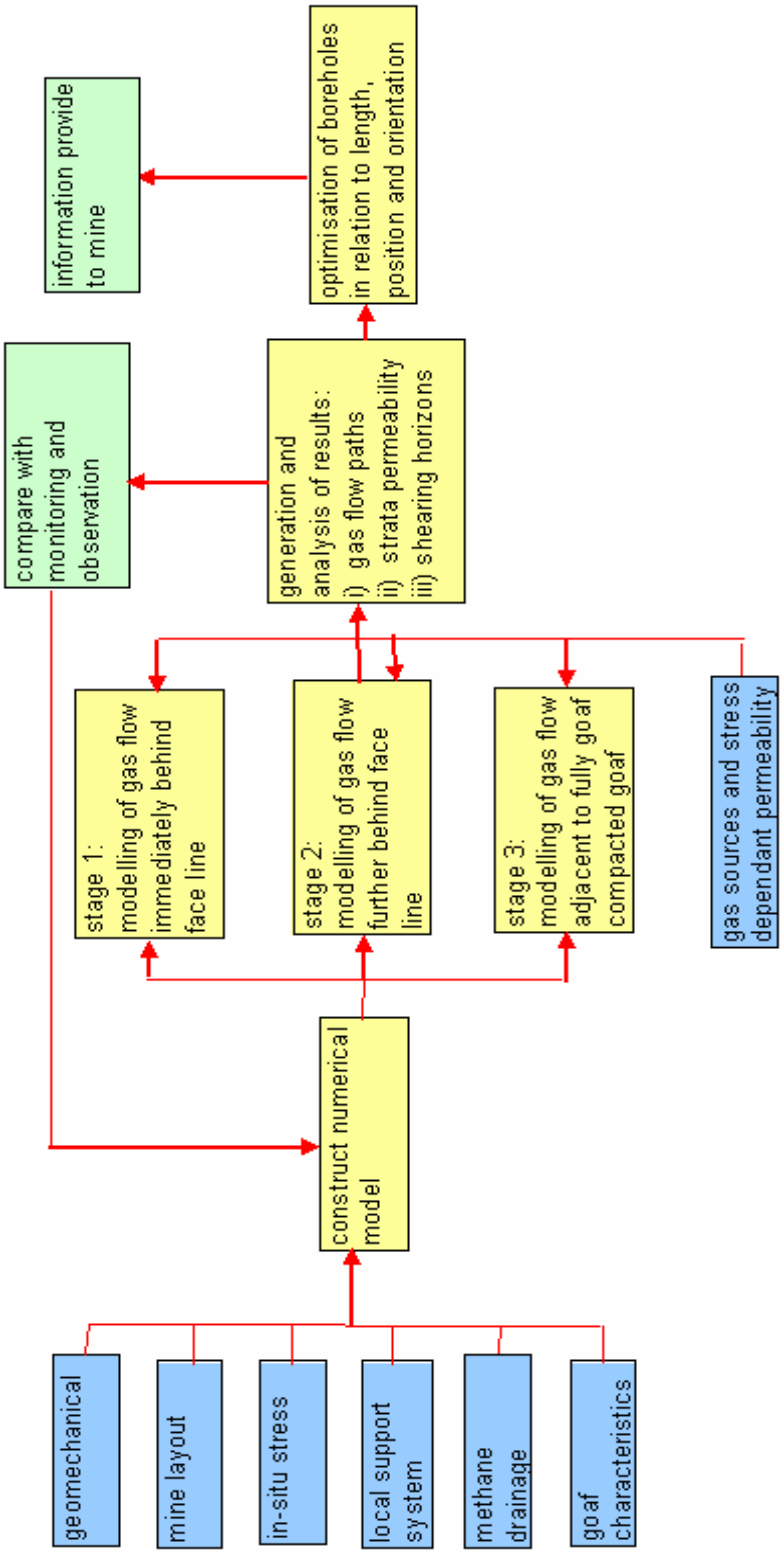


Figure 3: The numerical modelling methodology adopted for the construction of the 2D FLAC models

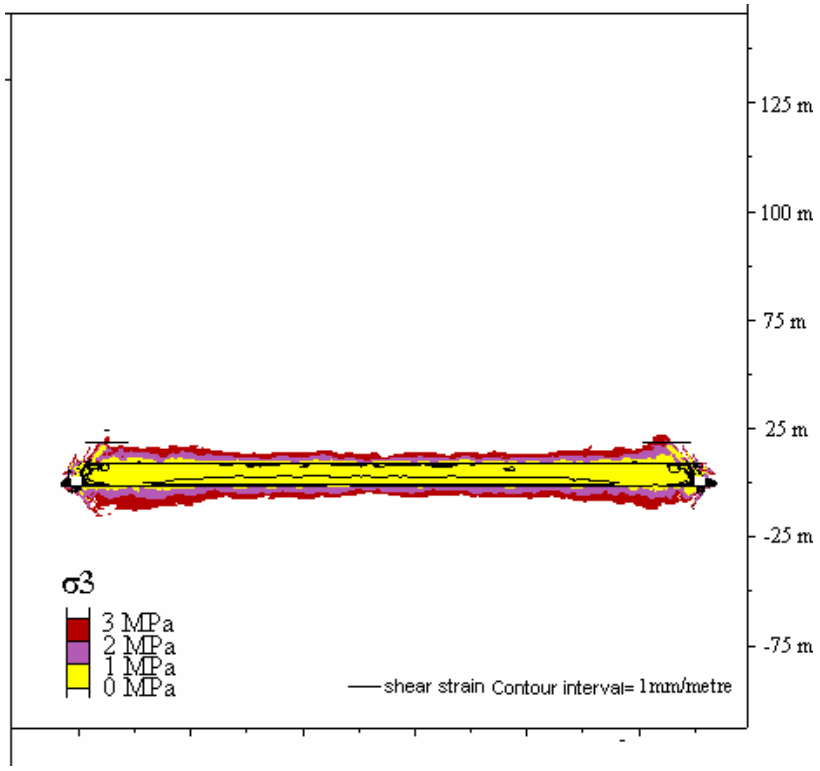


Figure 4: Model 1: Confining stress and shear strain developed behind the faceline of panel 43 Thoresby Colliery

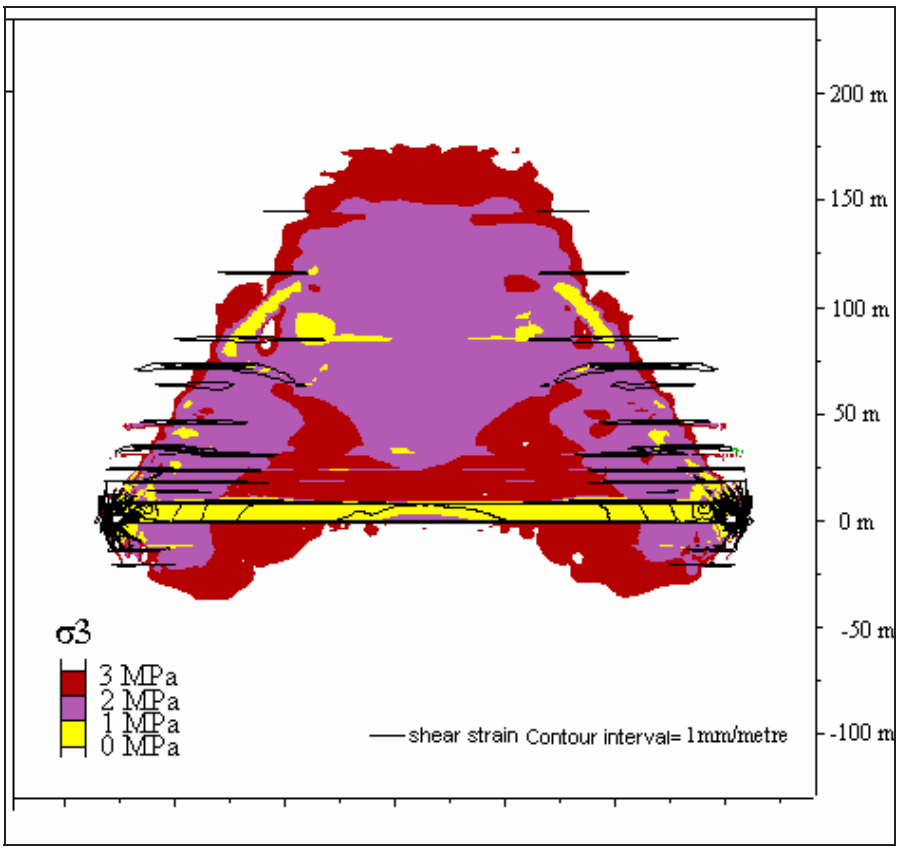


Figure 5: Model 2: Confining stress and shear planes around Panel 43, Thoresby Colliery

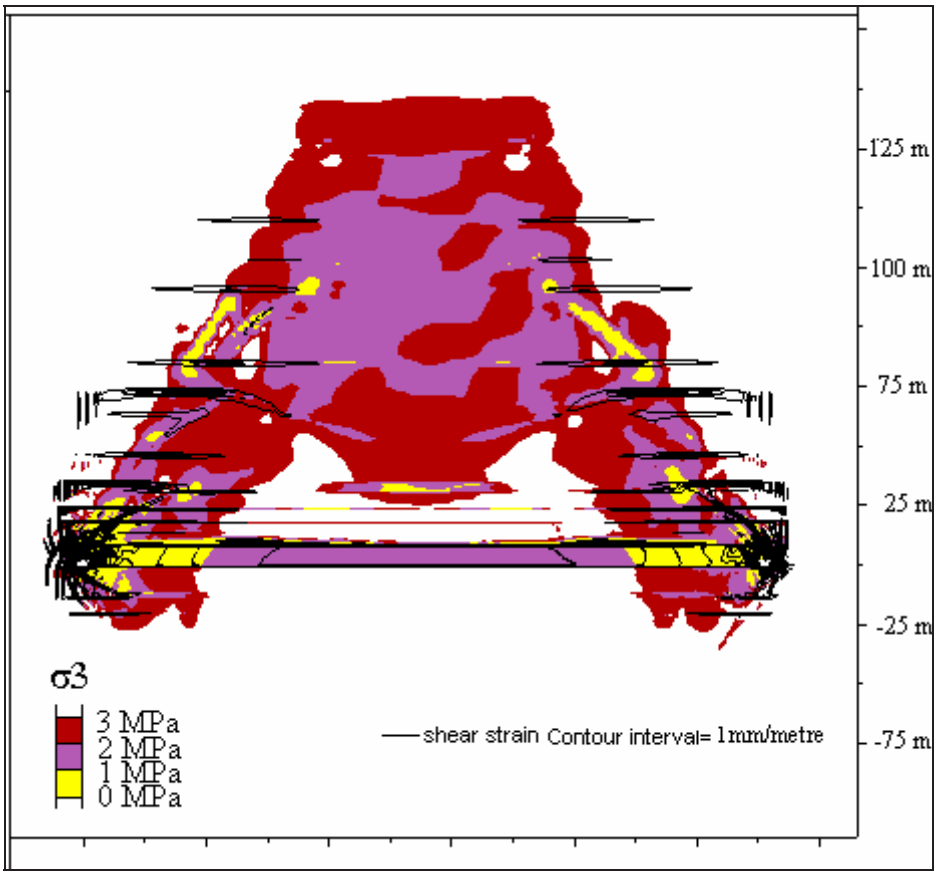


Figure 6: Model 3: Confining stress and shear strain developed around panel 43 Thoresby Colliery

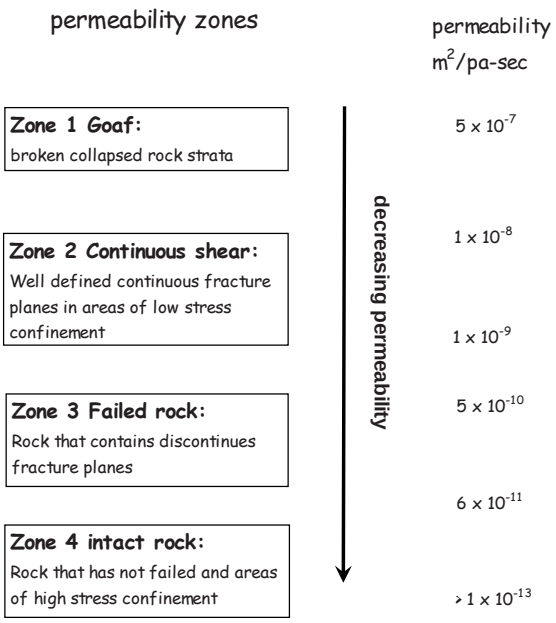


Figure 7: Zones of rock permeability

Methane Gas Sources

All the coal seams within the models were considered potential sources of methane gas. The aim of the numerical flow models was to predict the potential gas flow paths that would arise from the presence of methane gas within the interconnected adjacent coal seams. To simulate the presence of methane gas under pressure with the coal seams the fluid pressure was fixed at 1 x 10^6 N/m^2 within all the seams. All UK deep coal mines traditionally employ an exhaust main fan ventilation system whereby the pressure within the entire mine roadway network underground is maintained at pressures below the surface atmospheric pressure. This ventilation regime ensures that whilst the main fan is running a positive pressure gradient exists between the strata and collapsed waste areas, and the mine roadways. Thus, any gas within the strata or waste is under a suction pressure. Should the power to the main fan fail the mine roadways re-pressurise, and push the potential gas emission back into the relaxed waste areas. During normal operating conditions any methane gas entering the ventilation air from the strata or waste is diluted and removed by the ventilation system. To simulate the pressure differential, the gas pressure on the sides of the roadway was kept fixed at 0.95 x 10^5 N/m^2 i.e. 0.06325 x 10^5 N/m^2 below the air pressure in the surrounding rock strata. This lower pressure was also initialised at the start within the goaf but was allowed to vary as gas flow into the goaf occurred.

The gas saturation was assumed to be 1, i.e. fully saturated with respect to gas in the rock strata. The saturation in the edges of the goaf and roadway roof and sides was fixed at zero i.e. acting as a fluid sink. The porosity was set at 50% for all the rock strata. In a perfect gas the bulk modulus is equal to the gas pressure. Thus the bulk modulus of the gas in the Deep Soft seam was set at 1MN/m^2. A lower bulk modulus of 0.1 MN/m^2 was set for the gas outside the seam to account for the lower gas pressures in these regions. However the bulk modulus of the gas is not critical for determining steady state conditions as it only influences the prediction in the initial transient flow stage, but has no affect on the flow characteristics once steady state flow occurs.

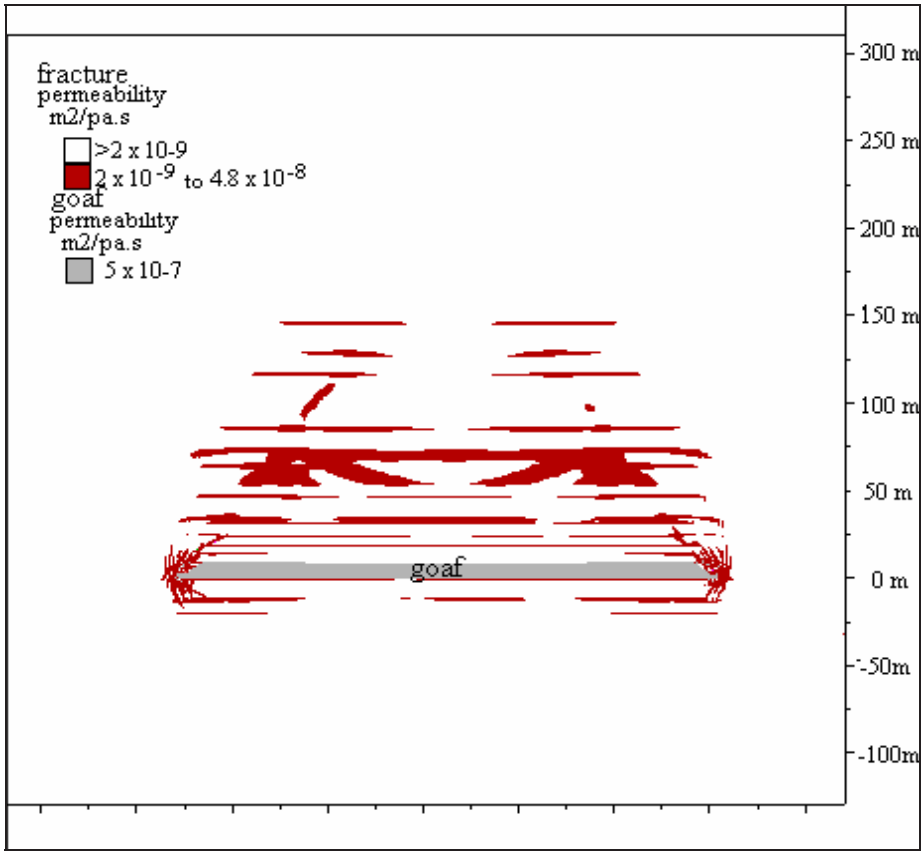


Figure 8: Predicted goaf permeability map across a section of Panel 43, Thoresby Colliery

Results of the Gas Flow Modelling

Gas flow modelling was undertaken using the FLAC finite difference time stepping method for solving the Darcian flow problem. The models were run until a steady state flow condition was obtained. Steady state flow was identified by monitoring pore pressures within the rock strata. The flow pattern predicted around the end of a representative longwall coal Panel is shown in Figure 16 below. The flow modelling predicted that major source of the gas flowing into the gate roadways was the Deep Hard seam, which lay at a distance of 19 metres above the Parkgate seam. Although other coal seams may act as potential gas sources the lack of connectivity of gas flow paths from these seams into the workings prevents the gas flow. An implication of this observation is that coal seams above longwall workings can be disturbed by mining but are unable to release their gas into the mine if the intermediate strata layers are not fractured. Therefore if the gas drainage boreholes are extended to these coal seams, the drainage boreholes may be venting substantially more gas than would have been released into the mine workings by the disturbance created by the mining. However, in practice, it would probably be wise for the mine operator to extend the boreholes into higher seams to reduce the likelihood of emissions occurring into the mine workings caused by variations in the geological conditions.

Analysis of the bending moment and axial strain on the borehole casing

The shearing of a drainage borehole occurs when horizontal shear planes within the rock strata intercept the borehole walls. The affect of the shear forces are to generate a buckling of the borehole which may lead to axial rupture of the casing and to borehole closure. From laboratory shear tests performed on borehole stand pipe, the elongation (axial strain percentage) of the steel standpipe at failure was determined as 20% [50]. The function (Equation 18) that was derived in Section 4 was used to determine the axial strain within the casing as a function of rock shear.

$$\varepsilon(x) = \frac{\frac{\pi . A^2}{a} . \sin(\frac{4 . \pi . x}{a}) + x}{\sqrt{A^2 . 0.5 . (1 - \cos(\frac{4 . \pi . x}{a}) + x^2}} - 1 \qquad [5]$$

where:

ε(x) is axial strain at distance x along the pipe (metres/metre)
A is is amplitude of displacement (u_{max}/2) (metres)
a is wavelength (2*width of shear zone) (metres)

For the prediction it was assumed that the width of the shear zone was 2 metres and that the casing displacement was 64% of the total shear displacement of the rock to account for the plastic flow of the rock around the casing. Failure at 20% elongation of the steel within the standpipe was assumed. Using these parameters within Equation 4 it was calculated that a horizontal displacement with a shear strain of 0.02 would generate failure of the casing.

Influence of roadway support systems on stability of drainage boreholes

At Thoresby colliery the closure of boreholes within the immediate roof of the tailgate had been observed to be associated with the magnitude of the downwards movement of the roof of the roadway behind the face line. An analysis of the borehole probing data collected behind the face concluded that the boreholes became closed at a distance of approximately 10 metres behind the face line. In order to reduce the amount of movement in the roof of the roadway and hence to extend the life of the boreholes during panel retreat, geomechanical models of Rockprop hydraulic supports were added along the rib side edge and the waste side edge, in addition to the rock bolts and cribs. The affect of this additional support increased the distance along which the boreholes were predicted to be operational to a distance of 17 metres behind the face line.

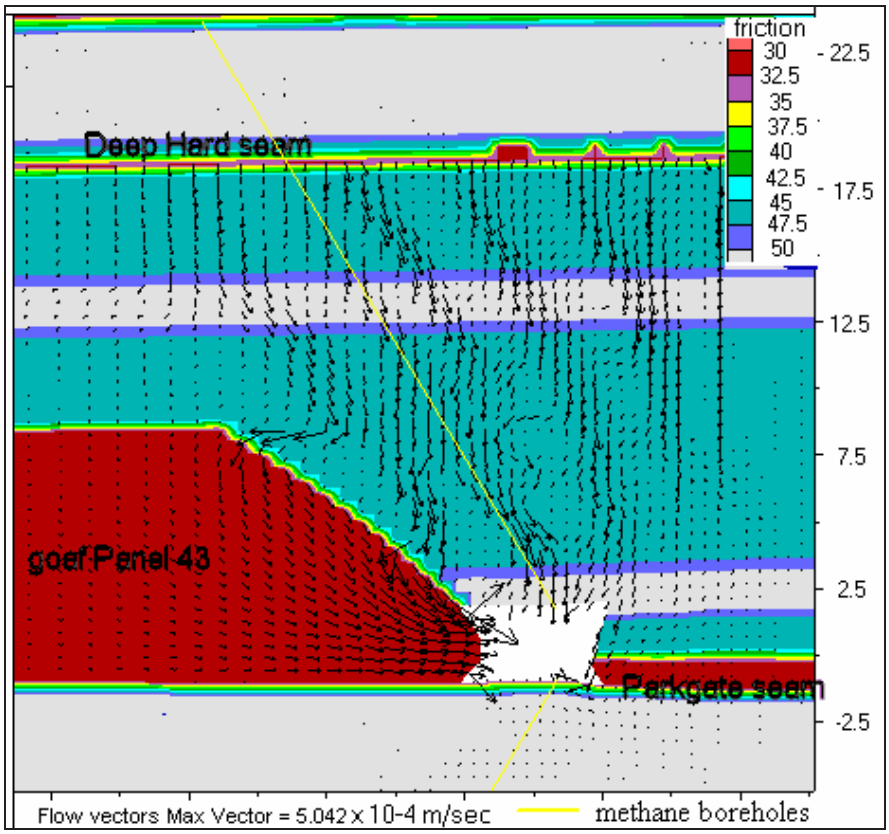


Figure 9: Flow pattern around the tailgate showing strata gas velocity vectors on Panel 43, Thoresby Colliery

These models were subsequently repeated by employing in turn three roof geologies, namely: weak siltstone, competent mudstone and strong sandstone. An analysis of the predicted roof displacement data concludes that the roof geology had a significant affect on the vertical displacement within the immediate roof, and consequently on the stability of any borehole passing through this roof geology. These conclusions were subsequently confirmed by borehole probing data performed on supported and unsupported roof geologies.

2.6.4 Task 4 - Gas Drainage Trials

2.6.4.1 Project Task Objectives:

- To identify improved drainage stand pipe length and orientation and spacing to maintain/improve the life and capture efficiency
- To develop combined geotechnical and environmental planning and operational guidelines to improve the life and the capture efficiency of methane drainage systems on rapid retreat longwall panels

This section of the research work involve the execution of field trials to confirm the optimum length, orientation and spacing of boreholes necessary to maintain the integrity and drainage efficiency on the range of colliery longwalls included in the study. The continuity and magnitude of the flow and purity of gases captured by the range will be a measure of the success of the various drainage configurations.

2.6.4.2 Comparison of initially planned activities and work accomplished

The methodology adopted to achieve the set objectives produced the following deliverables.

A series of field surveys were performed of the methane drainage ranges at each of the case study mines included in the study. The main conclusions from these studies were:

- The length and inclination of the boreholes employed at each of the mines studied were confirmed to intersect the predicted predominant interconnected fracture flow paths formed above and below the workings as the face retreats. Thus, the designs adopted by each mine by using the trial and error experience gained from previous longwall panels was shown to be directly related to the predicted geotechnical stress distribution and fracture propagation. This commends the use of the geotechnical modelling strategy detailed above to be used in the planning and design of the methane drainage systems of future longwall panels.
- The probing surveys conducted on boreholes to the rear of the face line confirmed that the use of an immediate support in the form of props and cribs greatly increased the longevity of the effective drainage of the boreholes. There is obviously a cost benefit analysis required to evaluate the cost of additional support against the benefit of maintaining coal production and gas utilisation. Unfortunately, it was not possible to complete this task during this project.
- For the selection of case study mines studied, it was concluded that the optimum spacing distance between adjacent boreholes was dependent of the strength of the roof in the tailgate. For an unsupported competent sandstone roof an optimum spacing of 12.5 m was recommended, and for a weak mudstone the spacing was recommended to be less than 8.9 m. It was realised that the use of additional roof support could therefore allow for both the increased spacing of boreholes as well as increasing the effective drainage life of each borehole.

2.6.4.3 Description of activities and discussion

Optimisation of the Spacing and length of the Boreholes

The position along the length of the standpipe and the distances where rupture of the casing was predicted to occur along behind the face line of Panel 44 of Thoresby Colliery is detailed in Table 1. An analysis of the data presented on the Table allows a prediction of the critical distance behind the face line when rupture is predicted to occur. It can be observed that the critical distance for rupture varies from 11.3 metres for the roof strata in Geology 2 where rupture was predicted at a height of 17.3 metres into the roof, to 8.9 metres for Geology 3 where rupture was predicted at a height into the roof of 6.9 metres.

Height into roof (metres)	Distance along standpipe (metres)	Geology one		Geology two		Geology three	
		bolted	Bolted and cribs	bolted	Bolted and cribs	bolted	Bolted and cribs
5.2	6	No fail	No fail	21	21	No fail	No fail
6.9	8	20.4	20.4	21.2	No fail	8.9	8.9
17.3	20	10.7	9.6	11.3	11.3	9.8	10.7
23.4	27	20.4	20.4	21.1	20.6	20.9	21.3
35.5	41	16.3	16.3	16.2	16.2	16.7	16.2

Table 1: Distance behind the face line where bore hole casing failure is predicted. [* Grey shade represents critical distance]

From an analysis of the results predicted by the models it may be concluded that to maintain the integrity of at least one gas drainage borehole in the area of potential gas flow immediately behind the face

line to a distance of approximately 30 metres then the spacing should be less than the predicted distance for the rupture of the casing. Thus for the weak mudstone roof the distance should be less than 8.9 metre whilst for the case of the stronger sandstone roof the distance should be less than approximately 11.3 metres. From the experience developed on the previously extracted adjacent panel 42, the optimum average operational spacing was determined to be 12.7 meters, which corresponds very closely to the optimum distance predicted for the strong sandstone roof.

Analysis of Gas Monitoring Data

Gas flow and gas purity levels were recorded from two parallel methane drainage ranges (Range A and Range B) installed within the tailgate of Panel 44 on a daily basis by the mine ventilation engineers at Thoresby Colliery. The two ranges were used to collect gas drained via the drainage boreholes installed in the tailgate of Panel 44. This research work, conducted to fulfil of Task 6.4, involved an analysis of the daily monitored flow and purity data, to identify the main factors that may influence the efficiency of the methane capture and drainage during the retreat of the longwall. The gas flow monitoring data was obtained for the period 21st July 2005 to 5th March 2006. During this period the face had retreated from the 2105 metre mark to the 1004 metre mark, a total distance of 1101 metres.

The affect of suction (differential pressure) on the total volume of flow (mixed flow) and the total volume of methane (pure) flow.

The effect of suction (differential) pressure in the drainage ranges on both the total (mixed) gas flow and methane (pure) flow within the ranges was analysed by plotting the differential pressure within each range against mixed and pure flow (Figure 10). Figure 10 shows that the total flow had a strong non-linear dependency on the suction pressure within the ranges, i.e. the data indicated that increased total volumetric gas drainage of the rock strata can be obtained by an increase in the suction pressure. However, the non-linear relationship shows that the increase in volume flow rate per unit of pressure is less significant at high differential pressures.

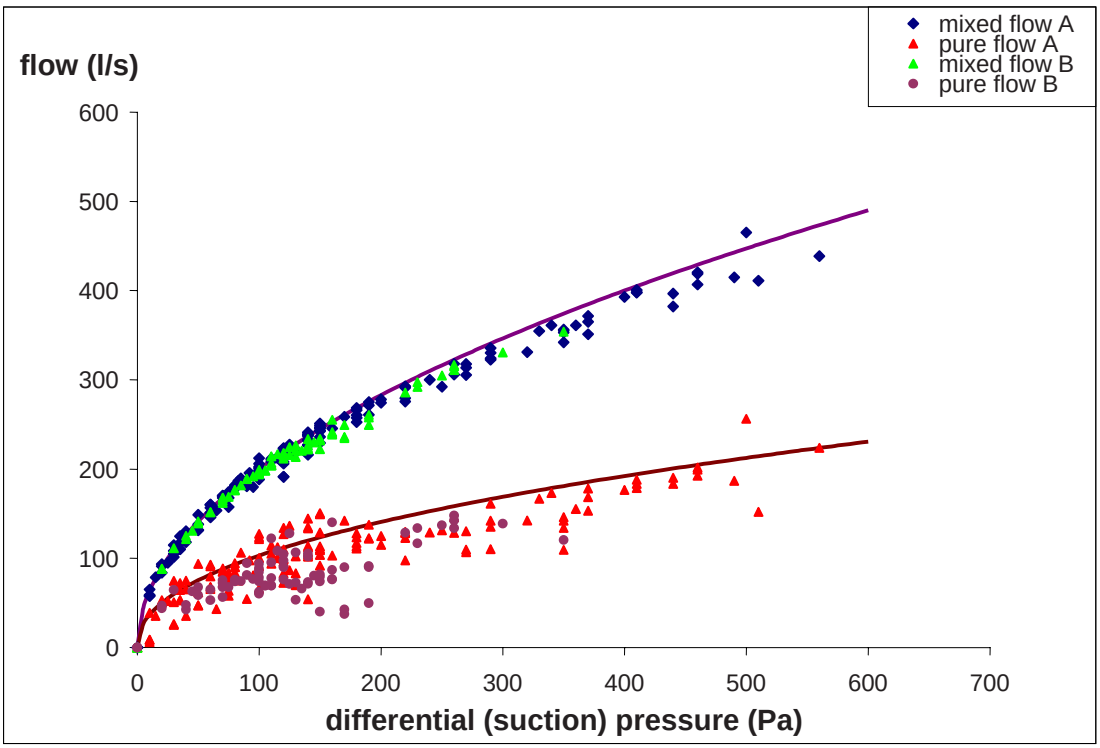


Figure 10: Operational relationships determined between the suction pressures within the drainage ranges to total gas capture rate (l/sec)

An investigation was also conducted to determine if there was a relationship between the differential pressure within the ranges and the percentage pure flow measured. An analysis of the data concluded that on average, for Range A, 46% of total flow was methane and in Range B 40% of total flow was

methane. \it was further concluded that above approximately 180 Pascals differential pressure, the % pure flow drained was fairly constant. However where the differential pressure was less than this value, the purity of the captured methane flow is highly scattered, varying between 10% pure flow up to approximately 65% pure flow. Thus, it was concluded that in order to maintain a constant volumetric flow of methane within the ranges, that a differential pressure of 180 Pascal or greater had to be maintained. The effectiveness of the methane drainage system to capture strata gas was determined as the percentage of the total methane flowing out of the district that had been drained via the boreholes. To analyse the effect that the magnitude of the differential pressure within the drainage ranges had on the % purity of the gas drained, the % drained from a district via the borehole drainage system was plotted against the average differential pressure within the two ranges. An analysis of this data concluded that the % purity of strata gas drained decreased rapidly for a differential pressure below approximately 160 Pascal. However, above a differential pressure of approximately 160 Pascal the % purity of the drained gas remained fairly constant at approximately 30%. The above analysis indicated that to maintain a constant volumetric flow rate of methane within the ranges, and also to maximise the amount of gas drained from the district, the differential pressures in the drainage ranges should be approximately 180 Pascal. Increasing the suction pressure to greater than this would have only a small effect on the increase in methane captured, whilst at pressures below this the effectiveness of the methane drainage system falls rapidly.

Relationship between borehole position in relation to the coal face and % methane gas capture

To investigate the effect that the location of a borehole in relation to the face position, may have on the percentage of pure methane flow obtained, the percent pure flow was plotted against the distance relative to the face of the most recent drilled borehole to be connected to that range. An analysis of the data concluded that their was significantly more scatter in the percentage of pure flow when the last borehole was less than approximately 30 metres from the face line. It is proposed that this may be attributed to two factors which are namely: (a) Propagation of fractures and rapid changes in stress conditions occur around the face line that may affect gas release and flow into this region and (b) Damage and blockage of the standpipes and boreholes due to the strata movements that occur as the face retreats. This may effectively limit the drainage of the boreholes to a specific horizon, at distances greater than 30 metres.

2.6.5 Conclusions

From the research work undertaken on the field case studies conducted at Thoresby, Harworth and Kellingley Collieries the following scientific and technical achievements have been obtained:

- The development of an advanced computational modelling methodology to simulate the development of rock fracture and stress redistribution around active longwall faces.
- The development of mathematical functions based on the analysis of permeability test data obtained from laboratory permeability tests to predict the stress dependant permeability of coal measure rock strata.
- The construction of gas permeability maps within the rock strata surrounding longwall panels
- The development of a methodology of simulating gas flow and allowing the identification of major gas flow paths and gas source horizons that can be targeted by gas drainage boreholes.
- Development and application of a three dimensional computational methodology of simulating the zone of rock fracture and high permeability around a longwall face so to allow the depth of both the steel casing and boreholes of a surface to gob gas drainage well to be determined.
- The development of an analytical equation that can be used to predict axial strain and point of rupture within the steel casing of a well due to rock shear resulting from longwall mining.
- Identification and quantification of the major geomechanical and mining factors affecting the long term stability of surface to gob drainage wells

This modelling work was validated against the geotechnical and gas drainage data obtained from retreat longwall faces at the Thoresby, Harworth and Kellingley collieries of UK Coal Mining Ltd. The field validated simulation models developed were able to identify the major fracture flow patterns created around the retreat longwall panels studied, and which intersected the major gas bearing horizons above and below the longwall being worked. This work was able to confirm the optimal orientation of the standpipes to intersect the strata gas flow. The analysis of the strain induced on these standpipes was able to provide a predictive tool to indicate the potential failure of the boreholes behind the face line and thus indicate an optimum spacing of standpipes to maintain gas capture.
The research work was also able to confirm the applicability and efficiency of face end support systems to maintain the stability of boreholes behind the face line.

The major conclusion is that the modelling methodology developed may be used to give a greater understanding of the fracture flow patterns to be intercepted, and hence the orientation and length of the boreholes to be employed to maintain effective gas capture. A bespoke analytical method has been developed to predict the deformation of in seam drainage boreholes and their potential rate of failure due to movement and collapse of the strata due to face retreat. This method may thus be used to design optimal drainage borehole spacing to maintain effective strata gas drainage. Dependent on the competency of the immediate roof, it has also been shown from modelling and field measurement studies that the use of gate end support methods can also prolong the drainage life of installed boreholes. The analytical methodology developed above could be used to design the optimal orientation, length and spacing of conventional gas drainage boreholes at the planning stage, and to assess the suitability of employing additional gateside support on a case by case basis.

2.6.6 Exploitation and impact of the research results

Future Application and Development of the Coupled Geotechnical and Gas Modelling Method

During the execution of the project UK Coal gave the University of Nottingham research team access to the geotechnical and operational data of a number of longwall production units at several deep underground coal mines. The results of the fundamental modelling exercises performed at these operations were able to:

(a) give the mine geotechnical and environmental engineers an improved understanding of the major fracture patterns induced above and below working longwall panels, and where these fracture profiles connected gas bearing measures with the mine workings, and
(b) to be able to use this improved geological and geotechnical information to improve the optimal orientation and spacing of the boreholes to maximise the capture of the strata gas released from the identified major coal bearing horizons.

Geotechnical stress modelling is routinely conducted by the UK Coal to assess and design the underground support methods employed within the operational deep mining operations. This project has developed a coupled geotechnical and environmental systems design process that will, it is the intention, be integrated into the production planning process of future long wall operations of UK deep coal mines. This method has also been successfully applied to the assessment and design of the gas drainage systems at the independent deep mining company, Tower Colliery, in South Wales.

Dissemination of Results

The results of the research improve mine safety as follow:

The development of the coupled geotechnical and environmental assessment tool allow the mine ventilation engineers to improve the optimal orientation and spacing of the boreholes and hence maximise the capture of the strata gas released from the identified major coal bearing horizons above and below the seam being mined. The improved efficiency of capture achieved by such optimised gas drainage

systems will consequently lead to improved control of strata gas entering the mine airways. This will principally improve the safe operation of these workings, as well as maintain coal production, and ensure the maximum extraction and utilisation of strata gas for energy generation.

References

[1] Whittles D.N., Application of rock mass classification principles to coal mine design, PhD Thesis, University of Nottingham, 1999

[2] Anderton R., Bridges P.H., Leeder M.R. and Smallwood B.W.,1979, A dynamic stratigraphy of the British Isles, George Allen and Unwin, 1979

[3] Sonmez H. and Gokceoglu C., Discussion of the paper by E. Hoek and M.S. Diedrichs: Empirical extimation of rock mass modulus, Int J Rock Mech. Min. Sci., 2006; 43, pp 671-676

[4] Itasca Consulting Group Inc., FLAC Version 4 Users Manual, Itasca Consulting Group Inc., Minneapolis, Minnesota, 2000

[5] ITASCA, FLAC^{3D}, Version 2.1, Itasca Consulting Group Inc., Minneapolis, Minnesota, 2002

[6] Hoek E. and Brown E.T., Practical estimates of rack mass strength', Int. J. Rock Mech. Min. Sci. & Geomech. Abstr., 34, 8, pp 1165-1186, 1997

[7] Yavuz H., An estimation method for cover pressure re-establishment distance and pressure distribution in the goaf of longwall coal mines, Int. J. Rock Mech. Min. Sci & Geomech. Abstr., 41, pp 193-205

[8] Peng S.S. and Chiang H.S., Longwall Mining, John Wiley and Sons, 1984

[9] Bigby D.N., Casie J.W. and Ledger A.R., Absolute stress and stress change measurements in British Coal Measures, ISRM Symposium: Eurock 92, Ed Hudson J.A., Thomas Telford, 1992, pp 390-395

[10] Anon, Assessment of retreat options and maingate support requirements for Rossington with initial application to B100s unit, Interim Report – Stage 1, Rock Mechanics Technology, May 1997

[11] Wilson, A.H. The stability of underground workings in the soft rocks of the coal measures, Int. J. of Mining Eng,, 1983,Vol. 1, pp 91-187

[12] IEA Coal Research, Gas control in underground mining, Creedy D.P., Saghafi A. and Lama R., April 1997

[13] Jozefowicz R.R., The post failure stress permeability behaviour of coal measure rocks, PhD Thesis, University of Nottingham, 1997

[14] Durucan, S., An investigation into the stresss permeability relationship of coals and flow patterns around working longwall faces, PhD Thesis, University of Nottingham, 1981

Annex

Requirements for the design of an auditable fire detection system

1 ORIGINATION

Within the R&D project SAFETECH an innovative measuring system is to assess an actual fire risk by measuring various different gases that represent fire hazard indicators. The measurement is effected by means of semiconductor sensors.

Devices that are intended to perform such a safety function must be subjected to a type approval testing of their technical measuring function. However, standards and other test regulations that would be directly applicable to these new types of equipment do not exist at this time.

The object of this investigation was to identify how and by means of what modifications the existing test regulations can be applied to such new equipment concepts. The application of such modified test regulations is to verify that the equipment operates in compliance with the equipment concept described and that generally customary requirements such as e.g. interference immunity against electromagnetic influences are fully met.

2 DESCRIPTION OF THE NEW TYPE OF EQUIPMENT CONCEPT

Early fire detection underground in German deep coal mining is based on the measurement of the carbon monoxide concentration of the mine air. If limit values are exceeded that are still to be defined and that correspond to specific CO production values, or if trends are recognized that lead one to expect short-term or long-term increases, warning messages will be triggered by computer. Operational interference influences such as blast events are filtered out by suitable algorithms. The devices used for this purpose operate by means of electrochemical sensors that feature a high verification sensitivity (verification limit under 1 ppm) in connection with a relatively high selectivity. This selectivity can be increased further by the use of filter receivers so that these devices only measure carbon monoxide and hydrogen. The ratio of the verification sensitivities for these two substances varies depending on the equipment type used.

This system for early fire detection is limited in its usefulness if unfavourable operating conditions (e.g. in the event of high air currents) or operational influences such as accumulator charging rooms (release of higher hydrogen concentrations) or Diesel traffic have a detrimental effect on the system. Depending on the setting of the parameters in the program, a significant increase in the number of incorrect warning messages is to be expected.

Within the framework of this project the reliability of forecasts should be increased by including further indicators such as e.g. the measurement of ethane and by taking into account any interference factors even at the conception stage of the actual data evaluation; in this way the frequency of such incorrect messages should be reduced. A detection system for early fire detection underground was designed that is to realize this objective by using a virtual sensor array (1 sensor whose operating temperature is cyclically varied) as well as suitable methods for pattern recognition. A detailed description of the equipment concept is provided in [1] and [2].

According to [1] the gases CO, H_2, C_2H_4 and NO_x are to be verified against a variable background of CH_4 and humidity. To this end, the operating temperature of the semiconductor sensor is varied cyclically according to a specified profile (cycle time approx. 70 s) and the measured values will be captured every 25 ms. When the temperature changes, the verification sensitivities for the various different gases to be considered change in respective different measures so that by a suitable evaluation the occurrence of the individual substances can be detected and any relevant information on their concentration ratios gained. As an indicator the ratio between CO and C_2H_4 is used; the verification of NO_x is primarily used

for the detection and suppression of interferences. As initial values, the signals "fire", "non-fire" or "warning" are to be provided.

By means of the cyclical operation of the sensor at different temperatures a respective set of measured values will be provided so that the sensor can be considered as an array of different sensors (virtual sensor array). A set of defined features is selected from these measured values. A pattern recognition process will then be applied to these features; as a result, together with a subsequent classification of the initial values, this will supply the states "Fire", "Non-fire" or "Warning".

The selection of the features and the interpretation of the pattern recognition (here: linear discriminant analysis LDA) as well as the classification are effected by means of training data generated in the laboratory. These training data contain the measuring data of the virtual sensor array when feeding in test gas mixtures that are assigned to certain specific situations underground. This comprises "normal" atmospheres, gas mixtures for starting self-ignition fires and open fires as well as typical interference situations (blast fumes, Diesel exhaust fumes, hydrogen releases). As the sensor has a considerable measuring effect for methane and the humidity of the measuring gas, the training data must also contain different methane and water vapour concentrations. The composition of these gas mixtures, that are to characterize cases of fire or non-fire, depend directly on the model upon which the equipment concept is based. The pattern recognition has been designed such that, starting from the training data, it is also possible to classify similar gas mixtures (e.g. the same concentration ratios for different absolute concentrations) (generalization).

3 TEST REGULATIONS FOR TYPE APPROVAL TESTING OF EQUIPMENT FOR EARLY FIRE DETECTION UNDERGROUND

3.1 Existing test regulations

The standard equipment types Polytron M CO, CO-Trans or BTS 48 CO, used in German coal mining today, have been subjected to type approval testing of the technical measuring function in accordance with the "Richtlinien des Landesoberbergamtes NW über Anforderungen an ortsfeste Kohlenmonoxid (CO)-Meßeinrichtungen und Durchführung der Eignungsuntersuchung" [3] ("Directives of the State Superior Board of Mines NW on the requirements for stationary carbon monoxide (CO) measuring facilities and the implementation of an approval test"). In the meantime these directives have been superseded; today a type approval test according to the European standards in the series EN 45544 (Part 1 in combination with Parts 2 or 3, depending on the application) [4-6] is required instead. In addition, it is now the state of the art to submit gas warning devices (and the equipment under consideration here is such a device), which use digital components and/or software, to a supplementary test according to EN 50271 [7].

Both test regulations are based on the concept that the devices measure the concentration of the target gas, show a quantitative measured value on a display and/or provide the same at a measured value output in the form of an impressed current (4 - 20 mA), a digital telegram or, as is customary in German coal mining, in the form of an impressed frequency (6 - 15 Hz). The requirements to be met by the device are formulated in the form of "general requirements" for the design (e.g. flameproof design, requirements for control elements), requirements for appropriate marking and the contents of the operating instructions as well as by maximum measurement errors which may occur during implementation of the partial tests described in the test regulation. In addition to the basic technical measurement properties such as linearity or response time, the individual partial tests here also deal with the normally occurring interferences such as changes in the climatic conditions (temperature, air pressure, humidity, blower stream direction and velocity) and the electrical environment (variation in the supply voltage, electromagnetic disturbances).

3.2 Transfer to devices with new type signal processing

The test regulations described in 3.1 do not take into account new type signal processing technologies such as are to be used in the equipment concept considered here. As the signal processing differs significantly from the type of processing used in conventional equipment, a different type of reaction to changes in the external influence factors is at least possible. In addition, influence factors occur that

need not be considered at all for conventional devices. An example for this is provided by changes in the methane concentration which in the case of the design usual for CO measuring instruments do not have any influence on the measured value.

A modified or supplemented test regulation must ensure that for the devices considered here a functional reliability can be verified that is equivalent to the same in conventional devices. The application must allow the verification that the devices operate in agreement with the equipment concept described and that the generally customary requirements for the technical measurement function are complied with. It is pointed out that the assessment of the suitability of the equipment concept for operational use is not the subject of such test regulations.

In all cases the starting point must be a detailed description of the equipment concept including a description of the methods selected for signal processing. In a first step it must be checked whether the input data used for the interpretation of signal processing are complete; that is, whether all relevant operating states occurring at the site of operation have been taken into account. This includes the gases and vapours occurring at the site of operation (if necessary also in the form of traces!) as well as the entire range of any occurring climatic conditions. With regard to the gases occurring, only those need to be considered which either cause a measuring effect at the sensor (in the case of a semiconductor sensor this would include almost all gases) or influence the sensor in its measuring behaviour either reversibly or irreversibly (sensor toxicants). Changes in the climatic conditions, in dependence on the sensor properties and the resulting signal evaluation, may possibly need to be considered also in combination; thus, for example, with a change in temperature there will always automatically occur a change in the range of water vapour concentrations which may occur (absolute humidity).

The standard EN 50271 mentioned at 3.1 will not be considered any further here. With the exception of a few points, a transfer to the equipment concept considered here should be possible without any significant problems.

The directives of the State Superior Board of Mines NW as well as the EN 45544 can be considered together here, as the basic structure is identical. The following text refers to the structure of the EN 45544 and, if required, is supplemented by individual partial tests from the directives of the State Superior Board of Mines NW (marked as "LOBA").

The partial tests can be roughly subdivided as follows:

- generally applicable tests, that is, independent of the measurement and evaluation method; with a suitable selection of test gases (see below), these can be applied without change or with slight modifications only
- specific tests for the measurement and evaluation method; therefore, these need to be transferred in an analogous fashion to this equipment concept.

For the individual partial tests described in the standard this means:

- Generally applicable tests:
  - General requirements (4.1 - 4.6)
  - Storage of the deactivated device (6.1)
  - Oscillations (6.3.1)
  - Drop test (6.3.2)
  - Temperature (6.4.1); if necessary, a modification is necessary due to the coupling with the humidity influence (see above)
  - Pressure (6.4.2)
  - Air humidity (6.4.3); if necessary, combine with temperature test (see above)
  - Flow speed (6.4.4)
  - Acoustic alarm (6.5.1)
  - Interference signal for a gas flow under-run (6.5.4)
  - Heating time (6.5.5)
  - Rise time (6.5.6)
  - Relaxation time (6.5.7)
  - Removal probe (6.5.8): not applicable as only stationary devices are considered here
  - Calibration device for field use (6.5.9) plus test gas adapter (LOBA 4.3.2)

- Usability position (6.5.12)
- Electrical tests
  - Mains energy supply (6.6.2)
  - Reporting electrical faults (6.6.3)
  - Electromagnetic compatibility (6.6.4)
  - Mean value weighted in time (6.6.5): not applicable
- Drift tests (6.7)
- Dust (LOBA 4.3.9)

- Specific tests:
  - Operating instructions (4.7)
  - Measurement of the deviations (6.2)
  - Alarm threshold (6.5.2)
  - Alarm rise time (6.5.3)
  - Gas concentrations above the end value of the measuring range (6.5.10)
  - Extended operation with standard test gas (6.5.11)
  - Influence of interference gases and vapours (LOBA 4.3.8)

The test conditions should be selected as follows:

- Ambient conditions as required by EN 45544, that is, laboratory conditions (with the exception of climatic tests); as otherwise unusually high costs would be generated;
- Sequence: Any stipulation beyond the measure selected in EN 45544-1 would not appear to be necessary.
- Otherwise as in 5.X EN 45544-1 with the exception of 5.6 (test gases, see below)
- In view of the measurement principle the test gases must be moistened and the absolute humidity must be kept constant (for example, corresponding to 50 % relative humidity at 20 °C).

For both types of partial tests, the requirements for measuring devices are expressed in measurement quantity units; naturally, due to a lack of direct measurement values, this concept cannot be used here. A meaningful transfer is necessary; within the meaning of EN 45544 such a device could be considered as a „pure warning device“. However, the standard covers this case only very inconsistently (it requires the specification of suitable measuring points for the purpose of type approval testing). As a rule, this concept will therefore not be applicable in the case of the equipment concepts considered here.

EN 50194 [8] offers a clue on how this problem can be dealt with: There, "Alarm" and "Non-alarm" test gases are defined that are fed in during the partial tests. During the transfer, the three states "Fire", "Non-fire", "Warning" of the equipment concept considered here must be taken into account:

- Zero gas (also as an ambient medium e.g. during the drift test):
  Typical underground atmosphere in the non-fire case, for example a mixture of 0.5 % CH_4, 3 ppm CO, 20.8 % O_2, remainder N_2
- Standard test gas 1 (Warning gas):
  Gas mixture(s) to be defined in accordance with the equipment concept
- Standard test gas 2 (Alarm gas):
  Gas mixture(s) to be defined in accordance with the equipment concept

Here, depending on the model upon which the equipment concept is based, several mixtures may need to be provided for each gas type, if necessary. The definition requires an individual consideration of the model upon which the equipment concept is based.

As already mentioned above, the specific tests must be adapted to this equipment concept. In addition to the stipulations of the standard, the operating instructions must describe in particular the equipment concept, the maintenance measures to be taken as well as any possibly necessary special operating notes (e.g. with regard to the permissible concentration ranges of the individual gases).

The partial tests ‘measurement of deviations’ (calibration curve) and ‘influence of interference gases and vapours’ must be modified considerably. The concept of the standard provides that, for the calibra-

tion curve test, the measurement gas as such, and in the case of examining the influence of interference gases, only interference gases, if necessary as an addition to the measuring gas, are considered. As the equipment concept considered here is based on a multi-component detection, such a subdivision cannot be maintained. A comparable problem exists in the case of the partial test "Gas concentrations above the end value of the measuring range": The equipment concept described does not provide any measuring ranges for the gases under consideration here. Nevertheless it must be required and verified by experiment that even in the case of very high concentrations (for example for CO in the case of a starting open fire) there will be a clear response by the device.

A suitable test strategy for this question must therefore take the following criteria into account, with the term “gas” also comprising the interference gases CH_4 and water vapour:

- Which gases may occur within what concentration ranges? Have these ranges been selected appropriately? What happens, if these ranges are exceeded or even under-run by one gas or several gases (humidity)?
- Which concentration ratio ranges between which gases characterize the various different states (from the definition of the model upon which the equipment concept is based)?
- What is the smallest concentration of each gas, which leads to a response by the device (verification limit)?

In a type approval test the concentrations of the individual target gases must in any case be varied across the permissible value range. In addition, the behaviour when exceeding the end value must be determined. Here, for each of the states to be considered, own test gas mixtures must be used. The requirement must be that the device will produce the correct classification „Fire“, „Non-fire“, „Warning“ for each test gas mixture within the permissible value ranges. If there are individual components outside the value range, either a correct classification must be effected, or there must be a clear message that the device is operated outside its specification. The dynamic properties of the device must be taken into account appropriately.

These dynamic properties can either be traced back to the measuring process or the evaluation procedures. Thus, in analogy to the filtering of blast fumes in the existing early fire detection system, it is conceivable that, before a fire is reported, a specific development over time of a certain feature pattern – that is, the composition of the gas atmosphere (or, during a type approval test, a certain composition of the test gas mixture fed in) must be present. These special cases must be taken into account in particular in the case of the partial tests ‘Alarm threshold’ and ‘Alarm rise time’. In addition, these tests must include all scenarios realized in the equipment concept.

The partial test "Extended operation with standard test gas" must also be transferred in an analogous fashion. Here, two different motivations must be differentiated in terms of content:

- The sensor can change its measuring behaviour by the existence of the measuring gas (for example, electrochemical ammonia sensors may become damaged even in the event of an extended occurrence of ammonia in concentrations within the measuring range).
- The evaluation procedure of a device provides for a dynamic drift compensation. This is used above all in the case of sensors that have an unstable zero point. In this case the test is to ensure that such a compensation does not „compensate away“ the measured value over an extended period of time.

For the equipment concept considered here, it is primarily the second aspect that needs to be considered here. Damage to a semiconductor sensor by the gases considered here is considered to be very unlikely, with the exception of very low humidities; however, there are no experience figures available with regard to the behaviour of semiconductor sensors during extended operation in the measuring gases. Therefore, this aspect must also be taken into account appropriately.

4 LABORATORY EXAMINATIONS

The prototype made available for laboratory examinations contained the hardware required for sensor operation and provided a serial signal output. Via this signal output, the unprocessed measuring values

of the sensors, their temperatures as well as the nominal temperature were transmitted to a laptop. Using the programs and scripts installed on this laptop, it was possible to carry out offline a data evaluation up to and including the implementation of a linear discriminant analysis (LDA). Further evaluations and the application of the LDA to additional measuring data were effected manually by means of commercial programs.

A specified algorithm for evaluating the measuring data within the meaning of the description in Section 2 was not made available. In particular, no details were provided on how the plane created by the discriminant functions is to be subdivided into the areas "Fire", "Non-fire" and "Warning".

Therefore, although it was possible to gain some initial experience with this equipment principle, an application or even refinement of the modified test criteria described at 3. was not possible.

5 SUMMARY AND CONCLUSIONS

The content of this expert comment is the examination as to how the existing test regulations for CO measuring instruments for use in the early fire detection in coal mining underground can be transferred in an analogous fashion to equipment devices with a new type of signal processing (offsetting the signals from several possibly virtual individual sensors). Proposals were formulated for the extension or modification of the current European standards for the testing of the technical measurement function of devices for the measuring of toxic gases (see Section 3). However, during the examination period it was not possible to implement a test application of test regulations modified in this way.

First laboratory measurements were carried out on a prototype. In terms of hardware equipment and the mode of sensor operation, this prototype corresponded to the equipment concept presented. However, the signal evaluation and classification of these states had not yet been integrated. Although the software provided for an offline evaluation of the sensor signals, it did not offer a possibility for classifying the states within the meaning of the equipment concept (compare Section 2). Therefore, the task was restricted to an examination of the functional principle and the derivation of some potentially critical criteria.

An assessment whether the equipment concept, complete with the model upon which it is based, is suitable for the monitoring task, that is, for the early fire detection in coal mining underground, should in any case be effected before carrying out any testing activities. Thus, the composition of the gas mixtures to be used in the training phase, which are to characterize the fire or non-fire cases, depends directly on this model. In the event of an incorrect or incomplete selection an urgent risk of defective indications is to be expected in the case of fire as well as in the case of a non-fire.

Measuring instruments, in particular if they are used for a safety function, require regular calibration and adjustment, if necessary, (if at all possible, directly at the site of use) in order to ensure their correct functioning. This measure is also demanded by the relevant regulations [9]. A practical application concept for such measures must be provided for the user and, if at all possible, assessed at the type approval examination. Such a concept must ensure that a safe operation between two calibration intervals is possible. The following criteria must be observed for the special case of the use of such equipment in coal mining:

- It must be possible to carry out the work at the site of use.
- It must be possible to carry out the calibration with as few test gases as possible; any transportation of a whole number of different test gas mixtures on site would be very laborious and costly. In addition, it must be noted that multi-component mixtures are very complex and such costly in their manufacture and that some of the components (NO_x) considered would involve stability problems with the handling and infrastructure customary in coal mining.

As an alternative to these criteria it could be conceived that a secured operating time for a defined period between two calibrations is to be verified if, additionally, there is the option of a simple functional check on site (e.g. by feeding in a CO test gas). In such a case the device could be removed for calibration and transported above ground. However, the necessary criteria require a consideration on a case by case basis. At least it must be proven without any doubt that, in connection with the functional test, the

development over time of the relevant sensor parameters allows a stable operation over such a calibration period.

6 BIBLIOGRAPHY

(1) Lehrstuhl für Messtechnik der Universität des Saarlandes, Saarbrücken:
Systembeschreibung "Detektionssystem zur Brandfrüherkennung unter Tage", Stand 28.11.2006

(2) Reimann, Peter: "Entwicklung von Auswertealgorithmen zur effizienten Branderkennung unter Tage", Diplomarbeit, Universität des Saarlandes, Saarbrücken, 2006

(3) Richtlinien des Landesoberbergamtes NW über Anforderungen an ortsfeste Kohlenmonoxid (CO)-Meßeinrichtungen und Durchführung der Eignungsuntersuchung, Ausgabe Juli 1976, Bezugsquelle: Verlag Glückauf GmbH, Essen

(4) DIN EN 45544-1: Elektrische Geräte für die direkte Detektion und direkte Konzentrationsmessung toxischer Gase und Dämpfe - Teil 1: Allgemeine Anforderungen und Prüfverfahren
Deutsche Fassung EN 45544-1:1999, Ausgabe Oktober 2000,
Bezugsquelle: Beuth Verlag GmbH, Berlin

(5) DIN EN 45544-2: Elektrische Geräte für die direkte Detektion und direkte Konzentrationsmessung toxischer Gase und Dämpfe - Teil 2: Anforderungen an das Betriebsverhalten von Geräten für Konzentrationsmessungen im Bereich von Grenzwerten
Deutsche Fassung EN 45544-2:1999, Ausgabe Oktober 2000,
Bezugsquelle: Beuth Verlag GmbH, Berlin

(6) DIN EN 45544-3: Elektrische Geräte für die direkte Detektion und direkte Konzentrationsmessung toxischer Gase und Dämpfe - Teil 3: Anforderungen an das Betriebsverhalten von Geräten für Konzentrationsmessungen weit oberhalb von Grenzwerten
Deutsche Fassung EN 45544-3:1999, Ausgabe Oktober 2000,
Bezugsquelle: Beuth Verlag GmbH, Berlin

(7) DIN EN 50271: Elektrische Geräte für die Detektion und Messung von brennbaren Gasen, giftigen Gasen oder Sauerstoff. Anforderungen und Prüfungen für Warngeräte, die Software und/oder Digitaltechnik nutzen
Deutsche Fassung EN 50271:2001, Ausgabe Mai 2002,
Bezugsquelle: Beuth Verlag GmbH, Berlin

(8) DIN EN 50194: Elektrische Geräte für die Detektion von brennbaren Gasen in Wohnhäusern. Prüfverfahren und Anforderungen an das Betriebsverhalten
Deutsche Fassung EN 50194:2000, Ausgabe Dezember 2000,
Bezugsquelle: Beuth Verlag GmbH, Berlin

(9) Richtlinien für die messtechnische Überwachung wettertechnischer Messgeräte und Messeinrichtungen (Messgeräte-Überwachungs-Richtlinien) vom 15.11.2005

European Commission

EUR 23353 — Optimisation of surveillance, technical equipment and procedures to prevent workers from danger attributed to fire, hazardous or toxic gases, firedamp or climatic conditions

J. Lehmann, D. Tischler , P. J. Morillo , J. L. Staub, M. Meyer , J. Walasiak, E. Hinz ,
J. L. Fernández Eguíbar , A. Czyz, T. Houeix, R. Jozefowicz , I. S. Lowndes , J. L. García-Siñeriz,
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variations in their pressure were made in order to study their behaviour facing the degasification and methane caption. The results obtained are represented in the Figure 6.