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Low carbon mine site energy initiatives

(Lowcarb)

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1. FINAL SUMMARY

A task by task summary of objectives, results, conclusions and applications:

Work Package 1 Basic Research: Potential of Carbon Reduction Initiatives

Task 1.1 The Energy & Carbon Cost of Underground Coal Mining

To review the proportion of total energy supply for extraction of coal from underground mining; to appraise / define sector accepted methodologies / metrics for carbon usage and reduction.

Estimations suggested that between one and two per cent of global electricity consumption is used for underground coal mining. Estimations of methane (CH_4) emission from underground coal mining suggested just over 1,000 $MtCO_2eq$ per year globally (on a rising trend) and approximately 20 $MtCO_2eq$ for the EU (on a declining trend). The application of carbon and ecological footprint tools to different mines showed that the most significant contribution to both footprints was the CH_4 released, although the ecological footprint, in assessing the capacity of available land to assimilate pollutant impacts, revealed that the impact of consumed electricity may be approaching a similar order of magnitude upon the ecosystem as a whole due to acidic gases from power station flues. This work confirmed that the success of the project may be measured in terms of potential for oxidation of CH_4 to carbon dioxide and reduction in electricity use.

Task 1.2 Potential for implementing Modern Renewable Energy into Mine Sites

To appraise the potential for introducing modern renewable technologies into mines sites; to examine options for direct-compression wind turbines, heat storage and compressed air.

Wind turbines, solar photovoltaic and solar thermal technologies and to a lesser extent geothermal heat recovery, algae and bio-mass could be electricity generators at mine sites. Solar energy correlates best with the Spanish mines, where an advanced solar industry exists, while wind power correlates best with the UK. Due to variability, the output must either lie well within the overall load (e.g. <9MW) with synchronisation or use a storage mechanism. The anchoring of wind turbines at the top of spoil heaps was mooted. Study of the existing air compressor function at the Velenje mine gave a mean of $18m^3$/second, consistently sufficiently high to take input from more than 2.5MW of direct compression wind turbine provision that has been calculated to provide up to a maximum of $6.77m^3$/second. In order to exploit energy potential and to show 'green' credentials, mining companies are to be encouraged to contemplate useful investment, which could be direct, joint venture or by leasing land space to and establishing a provision agreement with a specialist provider.

Task 1.3 Economics of Low Carbon Mining

To report on the economics of 'low carbon' mining.

A study was made of the general global economic environment and financial enablers, from carbon credit markets to a range of national incentives. At EU level, a comparative review of the UK and Poland showed that significantly more grant-based financial support and carbon credit opportunity is available in an 'economy in transition' such as Poland than in the already 'industrialised' economies of the EU, where the major state incentive is in the form of quota targets backed with financial disincentives to failure. Parts of the world appear to be economically active once again and natural resources are in demand, especially in the Asia-Pacific region. However, much of this activity relates ultimately to western markets and significant politico-economic challenges have emerged in the USA and even more so in the Eurozone, while the Chinese economy may be entering a transition phase. Thus the macro-economic background is unpredictable as seen in the current malfunction of carbon credit markets. Proven technologies are needed, with acceptable investment risk rates and sound cost/benefit projections for investment decisions, reinforced by ethical investment pressures.

Task 1.4 Define an Inventory of Methane Emissions from Coal Mines in the EU

To produce an inventory of CH_4 emissions from underground coal mines in the EU, with more detailed study in Poland and Spain.

The inventory of CH_4 emissions from active and closed mines for years 2001-2010 was produced using best available sources (UNFCCC, EIA, partner contacts), including Belgium, Czech Republic, France, Germany, Hungary, Poland, Romania, Slovenia, Spain and the UK. A more detailed study was made for Poland, Spain and the UK, using domestic methodologies for emissions. Hard coal exploitation was terminated in Belgium (2007) and France (2005) and had reduced year by year in the remaining eight countries, both from methane mines and in total. In 2010, total CH_4 emissions

from coal mines in the major EU producing countries, i.e. Poland, Czech Republic, UK and Spain, was found to comprise 2.4% of the global total. Detailed information is available for reference.

Work Package 2 Minimisation of Fugitive Gas Emissions

Task 2.1 Quantification of Methane Emissions

To revise and update the IPCC 2006 methodologies for estimating CH_4 emissions vented from coal mines (active and abandoned) and to detail the characteristics of ventilation air CH_4 emissions from underground coal mines; to construct a CH_4 emissions inventory at European level by applying the estimating methodologies, with a general data review and report on CH_4 content in EU coal seams; to evaluate the CH_4 reserves within selected EU coal seams to be mined in ten years; to evaluate the capture possibilities for minimising CH_4 emissions and future utilisation; to develop co-ordinated strategies for minimising emissions and utilisation of the gas; to provide an inventory of CH_4 emissions to the atmosphere from EU coal mines.

An extensive review compiled EU emissions data covering the period 2000-2009. Regarding the IPCC (2006) Guidelines, many countries were believed to be using incorrect basin emission factors when the Tier 2 approach is used, and the Tier 1 approach is recommended, aligning measurement methodology in each country. Technologies for utilising CH_4 include gas engines and fuel cells utilisation strategies where drainage is efficient or at abandonment, as well as direct extraction of coal bed CH_4 and other concepts. National CH_4 reporting standards are variable in quality and some major non-EU coal-mining countries do not appear to provide data at all. Using a methodology proposed for Polish coal mines, some variation was noted between those values and the values calculated according to IPCC 2006 that do not require such precise data. Emissions history has been described in detail for EU countries, with a declining trend within the major producers that may be extrapolated forward, noting the anticipated closure of many underground mines. Detailed estimation of methane reserves has been performed for hard coal seams planned for extraction within the next ten years at a KWSA colliery.

Task 2.2 Laboratory Testing

To provide a catalogue of methane content, porosity and strength for various types of coal; to define a method and to provide testing results for the geophysical location of highly gas-saturated coal seam zones.

Coals were tested from various lithostratigraphic groups and depths in the KWSA mines area, as well as lignite from the Velenje Mine. Hard coals were sampled from 93 beds in 17 mines at a range of depth of 288–1200m. Coals displayed effective porosity (open) 0.68-12.48%, porosity (closed) 1.24-14.92% and porosity (fractured) 3.29-18.17%. Uniaxial compressive strength (UCS) ranged 7.6-51.5 MPa, most frequently 10-30MPa, with an inverse trend of UCS against porosity in particular stratigraphic groups. CH_4 content ranged from 0.0 to 21.1m^3/Mg. At Velenje: porosity 16.42-17.26%, CH_4 content 0.002-0.003 m^3/Mg, UCS 13.4 MPa. Higher values were observed for the younger stratigraphic groups (Cracow Sandstone Series; Mudstone Series). Increase in depth showed a decrease in porosity. Despite the inverse relationship of UCS against porosity in particular stratigraphic groups, no consistent changes in mean UCS were identified with the increase in the age of consecutive stratigraphic groups. This has provided a sound understanding and classification of coal properties for future exploitation in this important region. In order to exploit common outcomes, the geophysical location objective was addressed in Task 2.3 below.

Task 2.3 Numerical Modelling

To provide a computer programme and modelling results of methane drainage from coal seams of various permeability; to provide a computer programme and modelling results of methane and coal outburst with warning values for physical indices measured.

A model in Petrel software (ver. 2010.1) of a longwall measuring approximately 750m x 250m x 2m was developed from coal seam maps and the results of coal properties tests. Demethanation was simulated using Eclipse software, enabling a range of predicted outcomes with pressure drop over time for different borehole configurations and leading to a borehole layout design for the field trials in Task 2.4. A computer model in Particle Flow Code of coal outbursts was built, showing high CH_4 content, high porosity and low compressive strength of coal to create the conditions for coal and CH_4 outburst generation and long horizontal transportation of dispersed coal masses, leading to recommendations for early risk evaluation methods. Ground penetrating radar 'in situ' test measurements were performed in a longwall and analysed in order to detect in-seam high CH_4 pressure zones, which led to a specific drainage borehole test in Task 2.4 below. The drainage modelling was shown to be fairly close to actual operation once parameters had been adjusted to match real conditions and offers a useful tool for drainage efficiency optimisation.

Task 2.4 Field Testing & Optimisation

To provide the results of coal mine trials and the optimisation of methane drainage technology; to provide an economic analysis of pre-mining methane extraction and guidelines for various coal mining conditions.

Following a fire on the longwall panel 193 seam 510 initially chosen for this work, exploratory borehole analysis was repeated for longwall 121 in seam 364 at Brzeszcze Colliery to enable modelling (see Task 2.3), leading to a design for drainage boreholes. Six drainage boreholes 76mm diameter, 100m length (except two which were shorter due to technical problems) were drilled, with one being positioned out of sequence to investigate the suspected high methane pressure zone detected by GPR (see Task 2.3). Boreholes were set and a set of daily measures taken (%CH_4, out-flow m^3min, in-borehole pressure, negative pressure, distance from longwall). Overall outflows were much lower than for the classic drainage technique over the gob and hence cost per m^3 much higher, showing that the technique as it stands is not suitable for the Polish mines although this has led to a subsequent project to test different approaches. However, the borehole used to validate the use of GPR to detect high concentration zones gave significantly higher output, thus supporting the concept of GPR use in this regard.

Work Package 3 Mine Site Energy Generation

Task 3.1 Examine Various Ventilation Air Methane Utilisation Technologies

To review current combustion technologies; to undertake a survey of heat loads suitable for tri-generation applications; to investigate and to characterise catalytic foam flow reversal reactors.

A review was produced of potentially useful technologies for the very low concentration ventilation air CH_4 (VAM) oxidation, including experimental work carried out by one of the partners (UNIOVI) on the catalytic flow reversal reactor as well as literature review and direct enquiries with specialists in the fields. Other than the thermal flow reversal reactor (TFRR), which is operating at a limited number of locations where CH_4 is within 0.18-1.2% and subsidy from credits is available, no technology has been successfully advanced to adoption. The heat load survey concluded that some Polish mines provide the only promising sites for tri-generation, considering ambient temperatures and mine conditions. Computer models have been developed in Matlab™ for both the flow reversal reactor and the gas turbine. For the catalytic flow reversal reactor (CFRR), laboratory testing of catalyst supports and reaction kinetics has resulted in deeper understanding of optimum reaction states and design and control systems. Water was shown to be the only reaction inhibitor in the mine gas stream, for which a potential solution was designed. The CFRR offers several advantages over TFRR such as negligible NOx emissions, lower operating temperature and compactness, but catalyst cost is currently inhibitive and there is lower grade heat output with less potential for secondary use. While previous experiments with gas turbines had little success, a potential configuration for a cogeneration array using clean imported gas and creating heat for VAM oxidation was designed and modelled for a total energy system.

Task 3.2 Appraisal of current Sequestration and Gas Cleaning Technologies

To investigate pressure-swing and temperature-swing adsorption systems; to investigate gas-cleaning systems; to develop enhanced procedures for the operation of the flow reversal reactor with coal venting gases.

Several separation technologies, such as absorption, adsorption, membrane and cryogenic separation were studied, with separation by pressure-swing adsorption emerging as the most promising. Review and laboratory tests suggested the adsorbent with the best capacity and selectivity for CH_4 adsorption to be isoreticular metal–organic frameworks (IRMOF), being more stable than existing materials such as carbon or zeolites. It was shown that capacity for adsorption increases with cavity size of the structures, but selectivity for the higher volatile organic compounds also decreases with the size of the cage. The organic linkers of the IRMOFs were demonstrated to be the most active sites for adsorption, offering the possibility of varying the functionality of these molecules to favour CH_4 adsorption, while hindering the retention of other molecules. There is, however, a cost factor to consider along with challenging engineering considerations. Gas cleaning using plasma reactors was investigated and might work at a scientific level in creating usable by-products but requires intensive research and energy input analysis.

Task 3.3 Evaluation of Co-generation and Tri-generation Potential

To evaluate the co-generation and tri-generation potential for the main competing methane utilisation technologies.

Tri-generation exists using drained CH_4 to internal combustion engines at very gassy mines. Where VAM is in the range 0.45-1.2% (less than 10% of mines worldwide), sufficient energy is available to go beyond simply sustaining the TFRR reaction and to capture heat energy in a heat recovery steam generator (HRSG) for high pressure steam to drive steam turbines. This directly produces electricity at a scale to be significant to the mine while lower pressure steam remains usable to match heat loads or to feed an absorption chiller for tri-generation. The gas turbine configuration considered in Task 3.1 would also be a powerful tri-generation system but with a cost in imported gas. The need for tri-generation does not appear to be great, unless co-located industries can provide suitable heat or cooling loads. Only very gassy seams produce the concentrations for the technologies to be viable without supplementation and hence a cost/benefit gain against conventional approaches, but, where such conditions do exist, a system design can currently be commercially offered.

Task 3.4 Techno-economic Criteria for all Candidate Schemes

To quantify the cost / benefit and the environmental performance of competing technology routes and options.

Due to catalyst cost, CFRR was found to be excessively expensive to install, although providing a clean environmental output, and estimation of usable power output demonstrated a lower output than from the TFRR. Gas turbine experiments have been largely unsuccessful, although a novel configuration was proposed and some research carried out (see Task 6.1). From testing of upcast air at Hunosa shafts, only one had VAM emissions in the concentration range to enable CH_4 oxidation by TFRR. Using this for a study for TFRR design at different efficiency outcomes and financial assessment, a subsidy of between €1.87 - €2.10 per tonne CO_2eq would be required. For co- or tri-generation with TFRR, at a mine with an unusually high concentration of 0.5% VAM and a flow of 200m^3/second of air, it is estimated that 4MW of electrical output could be achieved from the high pressure steam, which at 24 hour working and depending on the country and supplier could be in the order of €2m per year in savings, with a further 2MW or more of cooling potential from the remaining energy in lower pressure steam. However, the TFRR does have some environmental impact with trace NOx and CO emissions. A return to a much higher price per tCO_2eq is needed to generate more activity, when a very large global, although not European, market could be brought into play.

Work Package 4 Improving Energy Efficiency

Task 4.1 Gathering Data, Simulation & Forecasting Energy Demand of Mining Operations

To develop a comprehensive load profile database in a mine operation; to model and to forecast the regional demand profile to compare with the load curve of an underground mine to understand the effects of load-shifting and load reduction.

Electronic power meters (replacing manual recording) and associated computing were installed at a large UK mine and integrated with the mine SCADA system. This provided twelve months of intensive data, enabling ‘snapshots’ of different working phases. Higher-level comparison data was taken from other partner mines for validation and scaling. A load profile database was built and a Matlab Simulink simulation of basic loads created. This provides a valuable research resource that goes well beyond anything the mines have available.

Task 4.2 Intelligent Demand Reduction & Responsive Technology for Underground Coal Operations

To examine smart grid initiatives across the EU; to appraise demand reduction & management strategies; to assess uncertainty in the energy industries and potential impact; to evaluate relevant technology systems; to evaluate the potential for carbon footprint reduction and to examine the cost / benefit.

A unique demand reduction system has been developed. The system structure is defined by the Demand Reduction Pyramid (DRP). The levels of the DRP are; Data Acquisition, Load Profiling, Load Characterisation and Load Scheduling. The load profiling uses the concept of load skeletonisation, reducing the detail of the load profile to a minimum. This allows fast computation of the load profiles. Simulation data was generated and used to develop and prove the demand reduction system. The system was then applied to the real-time data collected in Task 4.1. The results proved successful and an operational demand reduction system has been produced that is ready for operational refinement.

Task 4.3 Ventilation Improvement

To design the ventilation network and to develop required algorithms; to automate the system; to introduce robotic control devices; to provide measures to reduce ventilation circuit leakage; to

provide warning systems and requirements; to define escape ventilation characteristics; to consider cost effectiveness.

A comprehensive programme of upgrading air speed and gas sensors in the Velenje mine was achieved, with verification using portable devices and more rigorous cleaning and calibration regimes to cope with the very dusty conditions. Algorithms were developed and the Central Energy Surveillance System (CESS) upgraded for consumption monitoring. The Zračenje ventilation software has been upgraded, including comparison with other software including that of another partner AITEMIN. This included analysis of 'object types' and other characteristics in the roadways to improve airflow analysis and auto-interaction with the New Linear Airway Network Map (NLANM). A major exercise to analyse the efficiency of fan blade angle adjustment in response to gas occurrence at faces resulted in understanding of the need for a total mine system approach using simulation prior to significant changes. Cost analysis was undertaken, with a difference of €1,220 in 24hrs between blade angles -9° and at +6° on the main fan showing the importance of optimal adjustment.

Task 4.4 Pumping Efficiency Improvement using Intelligent Control

To identify / develop and appraise technology to measure efficiency and emissions of turbo-machinery in real time; to integrate temperature, pressure and electrical power technology into the mine network demand reduction strategy; to conduct trials on a test rig; to integrate real time acquisition and control of turbo-machinery into the demand reduction strategy.

A 'mine simulator' was constructed at the CSM test mine, including pipework with tappings for sensors, a pump, a throttling calorimeter, a three-phase power analyser and a computer. Sensor technology for very fine pressure and temperature measurement was assessed, tested and specified. The accuracy of this approach to pump efficiency was verified against a controlled traditional method in the simulator before applying the sensors to working pumps at two locations, showing a potential 25% cost saving in one case and indicating great potential for the many large, and often very basically maintained, pumps in deep mines. This technology is now available for commercial application.

Task 4.5 Surface Facilities Energy Efficiency

To assess the potential for carbon dioxide emission reduction at a modern colliery site by consideration of construction standards, energy systems, fuel switching, renewable energy and decreasing demand through behaviour or technology.

The importance of starting with a baseline survey, including regulatory review, was highlighted. Potential improvements to the fabric of typical mine site buildings to improve energy efficiency, either minor or major refurbishment, were identified. Condensing boilers, combined heat and power, micro-processor controlled burner modulators with optimisers, de-stratification fans, photo-electric lighting controls and light-emitting diodes are among the options, along with modern insulation standards. Use of intelligent systems such as smart energy metering, sensors and timers was described and, importantly, methods of changing to human behaviour with regard to energy saving were put forward. Trends in the costs of different fuel sources for potential switching were studied and the various renewable energy options for industrial sites again reviewed. Where a mine has a known long-term future life, investment in some of these upgrades is strongly recommended.

Work Package 5 Energy Storage

Task 5.1 Comparative Review of Energy Storage Options

To provide a comparative review of energy storage options to assess the potential application of Compressed Air Energy Storage (CAES).

A review has been carried out of energy storage systems, either available or under development. In addition to compressed air, this included pumped hydro, flywheels, electro-chemical batteries, organic molecular (photosynthesis), biological, magnetic and thermal methods. It was concluded that, with the right conditions and scale, compressed air has the potential to compete well, especially in the adiabatic system, where thermal energy from the system is concurrently stored and re-used at expansion.

Task 5.2 Compressed Air Scheme Options & System Optimisation

To evaluate CAES options and technology requirements for system optimisation, taking into account a specified range of options, limitations and other factors.

CAES has been identified as probably the lowest-cost utility-scale bulk storage option for dealing with the intermittency of non-hydro renewable energy sources, such as wind turbines that might be anchored on nearby spoil heaps. Adiabatic and diabatic CAES systems have been considered, favouring adiabatic that uses a thermal energy store to reheat the expanding air, although a dual option capable of adding additional heat (perhaps from a mine methane energy technology) has been identified to provide greater capability in periods of sustained demand. For this study of a site on typical UK flat surface topography where the cost of constructing a surface vessel would be prohibitive to the investment, an isochoric underground air storage cavity (constant volume) is preferred to isobaric, where mine water might be used as a 'piston', although isobaric would have higher energy storage per unit volume of air. A novel underground 'thermocline' sensible heat store using rock as the storage medium is proposed and heat exchanger design has been modelled. Heavy equipment such as compressors, expanders, motor/generator, recuperator has been reviewed and some useful estimation of profit /kWh carried out using actual data from a facility in the USA and UK energy price fluctuations. Potential for overall profitability once the considerable initial investment is included depends entirely upon predictions of future energy price movements.

Task 5.3 Site Selection, Sealing & Preparation of Storage Volume

To investigate a specified range of factors to provide a critical review and to identify the key factors that need to be considered / evaluated for the development of a CAES facility.

A simple axisymmetric FLAC^{2D} model was employed to investigate the effect of geotechnical and construction factors on the application of a circular cross-section mine shaft as a CAES facility. This showed the key aspects to be the integrity of the shaft caps, with backfill on the top cap significant, deformation ranges significantly reduced by concrete thickness and especially high-strength concrete in the caps, cap width having a very limited influence on the deformations, longer shaft length giving smaller deformations when there was backfill material above the upper cap, distance between upper cap and surface must at least exceed 5m. Following on from this initial work, further numerical models were developed that additionally showed an increase in vertical displacement when a liner-rock interface is introduced, emphasising the importance of the rock-liner bond. A geological fault was shown to create higher loading on the liner. Epoxy-based sealant or silica fume admix were identified options for concrete treatment. A concrete testing rig was built with the intention of raising the pressure to 80bar, but in the event air leakage occurred at the joint between the concrete and the seal to the vessel, showing the need to make extra provision where concrete sections abut in the shaft lining or any other such inconsistency in the structure.

Task 5.4 Evaluation / Assessment of Site Structural Risks

To apply empirical and numerical modelling to case examples to demonstrate the impacts of identified factors as appropriate.

Preventative measures against static include using conductive materials for the pipework and providing a low resistance path to earth, and either humidifying or de-humidifying the air. Gas vents to mitigate against unwanted gas migration must be widely separated from the CAES air inlet and in the opposite direction to the prevailing wind. Solenoid-operated valves in the upper cap may be triggered to release air if the rate of loss of pressure in the reservoir exceeds a threshold. Methane sensors in and around the facility would be installed.

The Maltby No 3 shaft was identified as the best case study from a review of UK shafts. 2D modelling was conducted based on strata and lining support information from Maltby. Top cap depths were analysed against pressures and lateral displacements measured. Two critical areas were identified as close to the top plug and close to the depth of 470 m underground where the shaft liner's thickness changed suddenly from 1.2 m to 0.4 m. The overall conclusion was that the concept would be viable in theory. Much of the necessary shaft work could be carried out as part of the normal closure work, thus minimising the additional cost of the civil / mining component of the total project costs.

Task 5.5 Exploitation of Heat Storage for Energy Generation System Efficiency

To review the current span of established heat-store methodologies with emphasis on liquid-solid phase-change approaches; to consider cavern-lining methodologies for enhanced thermal insulation; to examine pressurised thin-walled tank possibilities; to consider 'nested' heat storage configurations; to undertake basic modelling of cost-performance; to integrate simple heat-store and heat-exchange models into system dynamic models.

Following a review of sensible, latent and chemical heat storage options, a sensible heat storage mechanism has been chosen and designed for the flow reversal reactor (FRR) 'energy from VAM' system, capable of storing excess heat at high methane concentration periods and returning it to the reactor during low concentration periods. This improves the efficiency and stability of the reactor. The heat storage system has been modelled and integrated into the FRR model to simulate the dynamics of the combined system. The simulation of a TFRR equipped with a heat storage system under real coal mine ventilation air conditions indicates that the proposed heat storage system is capable of achieving stable reactor operation when CH_4 concentration varies in the 0.5-0.25% range.

Work Package 6 Potential Impact of Low Carbon Mining

Task 6.1 Testing & Evaluation of Low Carbon Mining Technologies including Operational Trials & Simulations

To test and to evaluate: mine site energy generation (WP3); energy efficiency systems (WP4); compressed air energy storage / heat storage (WP5)

For the simulation of flow reversal reactor performance, the code developed (Task 3.1) was applied to performance under specific reactor properties. The influence of methane feed concentration on the stability of the reactor was studied, showing that below 0.25% methane the reactor cannot be operated at autothermal conditions and extinction takes place progressively unless it rises again. When the concentration at the feed increases above 0.30%, the temperature in the combustion chamber increases above 1000ºC, for which heat extraction strategies were proposed.

A concept for using an array of small standard cogeneration gas turbines running on clean imported gas providing electricity and oxidising the VAM with the exhaust heat was studied. The oxidation depends upon the process of heat capture in a combustion zone recycling heat from a heat exchanger, aided by supplementary gas-fired igniters. The key modelling related to the combustion process and the heat exchanger, indicating that the system will work at a theoretical level, while a modest field trial was also undertaken with limited success.

Thermodynamic pump efficiency monitoring using the Poirson method refined with modern sensor technology in the CSM mine simulator (Task 4.4) has been subjected to two field trials, wherein certain pumps were clearly identified as in need of refurbishment and the method was validated.

For the ventilation improvement work at Velenje mine (Task 4.3), comparison of field data had revealed some mismatches and a trial exercise was set up for a non-production day. This validated the improvements made and highlighted the need for a total mine system approach, using the Zračenje software upgraded during this project to assess impacts before changing the fan blade angle. A concurrent trial of the CSM thermodynamic pump efficiency analysis to fans proved very sensitive to the moisture content of the air and a saturated condition equation had to be refined and applied before showing a generally good agreement between the thermodynamic and conventional methods.

N.B. Although referred to in the objectives, as the extensive modelling for compressed air energy storage is reported in Work Package 5 it has not been repeated here.

Task 6.2 Economic Analysis of Pre-mining Methane Drainage Technology for Underground Coal Mines

To conduct an economic analysis of the costs associated with pre-mining methane drainage technology (WP2) to produce a model and guidelines for mining operations.

A working model for pre-mining methane drainage economy assessment was produced according to the UNIDO methodology and Dynamic Generation Cost (DGC), Net Present Value (NPV) and Internal Rate of Return – (IRR) were proposed as an assessment criteria. Based on the investigations at Brzeszcze Colliery (data for years 2009-2012), the following main cost groups were identified relating to the methane drainage department: wages with contributions and

additions (71.3%), energy (21.1%) and other costs (7.6%). The cost of the underground trial of pre-mining methane drainage was twice as high as that of classic methane drainage technology. Calculated values of NPV (-1,833,900 PLN) and DGC (1.37 PLNpm^3 for pre-mining and 0.43 PLNpm^3 for classic) indicates that pre-mining drainage is not economically efficient, but the work will feed into a new RFCS project entitled Gas Drain to investigate additional measures.

Task 6.3 Evaluate the Techno-economic Model of Implementing a 'Low Carbon' Approach in Underground Mining Operations

To construct a techno-economic model on the feasibility of a 'low carbon' approach to underground mining operations against representative mining systems in the EU.

The currently available technologies of flow reversal reactor (FRR) and associated co- and tri-generation (running on ventilation air methane) and pump efficiency monitoring were considered. The technology features and limitations of the methane-related systems were analysed in some depth, followed with economic application to the potentially workable processes. Although having received considerable scientific investigation within the specification of the project, the catalytic version of FRR was rejected due to catalyst cost. A requirement for a methane concentration within a range of 0.18% - 1.0% CH_4 for the thermal FRR was supported with concentration measurements at the HUNOSA upcast shafts that showed only one in five within the range. A thermal reactor design was produced, based upon this shaft. Economic analysis showed that this would not be viable at current carbon credit prices. For the proposed co- and tri-generation system, a minimum of 0.45% CH_4 would be needed, limiting the number of shafts even more. The economics of this demonstrated the need for carbon credit subsidy to make this profitable and that a higher than current price is needed for this also. The pump monitoring sensors had been proven during the trials (Task 6.1) and attractive cost savings were demonstrated here, offering some of the best outcomes from the project.

2. SCIENTIFIC & TECHNICAL DESCRIPTION

A task by task description of the approach taken, work performed and main results achieved.

Work Package 1 Basic Research: Potential of Carbon Reduction Initiatives

Objectives of WP1: *The output of this Work Package (WP) is to provide a baseline of current information to underpin the work to be developed within the remaining WPs, including a review of the proportion of the total energy supply for the extraction of coal from underground mining in Europe and potentially around the globe; appraisal and definition of sector-accepted methodologies for quantifying key industry metrics for evaluating carbon usage and reduction; appraisal of the potential for implementing modern renewable technologies into mine sites; addressing the economics of 'low carbon mining'; and compilation of an inventory of methane emissions to the atmosphere from EU coal mines.*

Comparison of initially planned activities and work accomplished: The planned activities were all successfully accomplished, with some further useful published work in the area of 'ecological footprint'.

Task 1.1 The Energy & Carbon Cost of Underground Coal Mining

To review the proportion of total energy supply for extraction of coal from underground mining; to appraise / define sector accepted methodologies / metrics for carbon usage and reduction.

Activities and Discussion: A general review of the global and EU position of underground coal mining energy use and emissions has been carried out and further supported with detailed analysis of the carbon footprint and ecological footprint of certain mines in the partner countries, using carbon and ecological footprint tools. All emitted gases were considered, but methane, due to its high global warming potential (21 times that of CO_2 over 100 years) and high emission level, represents the most significant environmental impact. A less easily anticipated outcome of the use of the ecological footprint tool, in relating pollutants to the capacity of available land to assimilate impacts, revealed that the environmental impact of consumed electricity may be approaching a similar order of magnitude upon the ecosystem as a whole due to acidic gases, primarily sulphur dioxide from coal-fired generation.

Conclusions: The estimations of global and European electricity consumption suggested that between one and two per cent of global electricity consumption is used for underground coal mining (in the order of 300,000GWh per year, a significant overhead upon the coal energy process), while approximately 10,000GWh per year are consumed by European Union underground coal mines. Estimations of methane emission from underground coal mining suggested just over 1,000 $MtCO_2eq$ per year globally (on a rising trend) and approximately 20 $MtCO_2eq$ for the EU (on a declining trend).

Any action for reducing methane emissions in the ventilation gases would provide major benefit from the perspectives of the carbon (up to 85%) and ecological (up to 53%) footprints. The use of these detailed tools has provided support for the intuitive premise that the key metrics by which the underground coal mining industry might measure environmental performance at a global level are the emission of methane and the consumption of electricity. The latter has a natural appeal to the operators as it relates directly to the control of costs, while the former may also be seen to have benefit through the sale of carbon credits or reduction in carbon taxes, thus creating a virtuous circle of environmental damage mitigation and balance sheet improvement.

Exploitation and Impact: An academic paper has been published from the Spanish work on carbon and ecological footprints, while this task also provided much background material for a presentation at the 23rd World Mining Congress.

Díaz E, Fernández J, Ordóñez S, Canto N, González A (2012); Carbon and ecological footprints as tools for evaluating the environmental impact of coal mine ventilation air; Ecological Indicators; 18, 126

Bennett J G, Kennedy G A, Foster P J, Williams N C, Cluff D L, Clifford T P (2013); Low Carbon Coal Mining – A Contradiction or an Opportunity?; Proceedings of the 23rd World Mining Congress; ISBN: 978-1-926872-15-5

Task 1.2 Potential for implementing Modern Renewable Energy into Mine Sites

To appraise the potential for introducing modern renewable technologies into mines sites; to examine options for direct-compression wind turbines, heat storage and compressed air.

Activities and Discussion: A review of potential renewable energy technologies and, for the most promising, a description of the relevant engineering and energy factors, has been carried out. The potential for the application to actual mine sites has been considered, as have wider electrical transmission issues such as the potential for the microgrid. As direct compression wind turbines are one of the technologies, compressed air use has also been investigated to scale the opportunity, taking Velenje mine data for a case study. Solar energy clearly correlates best with the Spanish and other southern mines, supported by the advanced nature of the solar thermal energy sector in Spain. As solar power shuts down every night, a dynamic relationship with the grid is essential. However, greatest emphasis has been given to wind turbines. Site selection, wind turbine heights, wind turbine sizes and wind turbine spacing have been identified as factors for power generation planning, while computation of power, torque, forces, efficiency, scaling factors and wind shear effects make input to the technical design. Direct-generating or direct-compression wind turbines are potential candidates in the context of a coal mine. The wind source is, of course, variable and for direct generation the output must either lie well within the overall load as a supplement with appropriate synchronisation or must have a relationship with a storage mechanism or, more likely, the electricity grid for output and input.

Conclusions: Wind turbines, solar photovoltaic and solar thermal technologies, geothermal heat recovery, algae and bio-mass could be electricity generators at a mine site. The strongest correlation in the EU of available wind energy with underground coal mining activity is in the United Kingdom, although the effects of thermal gradients in mountainous terrain provide some additional potential elsewhere. The UK is a country where land space is at a premium and a detailed study of the surface characteristics of a UK mine site showed space and unfettered facing to prevailing wind for only one high wind turbine structure. Nevertheless, as the UK operates a feed-in tariff up to 5MW of wind energy, provided that the system is connected to the grid and some exchange of electrical power takes place, there should be some financial attraction in this single installation. The overall load of such a mine is up to around 16MW with wide variation, so a typical land-based turbine of, for example, 1.8MW maximum output would produce variable power well within the load and the variable generation output would not be an issue with appropriate transformers, synchronisation and control systems.

Another option mooted is conversion of wind energy directly into compressed-air with the help of moving mass inside the rotating turbine blades. This turbine is not yet at the stage of 'mature' technology, but was under consideration within this project for reasons of mine use of compressed air and post-mining energy storage potential. The pattern of compressed air usage at the Velenje mine site in Slovenia has been analysed (Figure 1, below), giving a mean of 18m^3/second, suggesting that it is consistently sufficiently high to take the input from more than 2.5MW of direct compression wind turbine provision that has been calculated to provide up to a maximum of 6.77m^3/second.

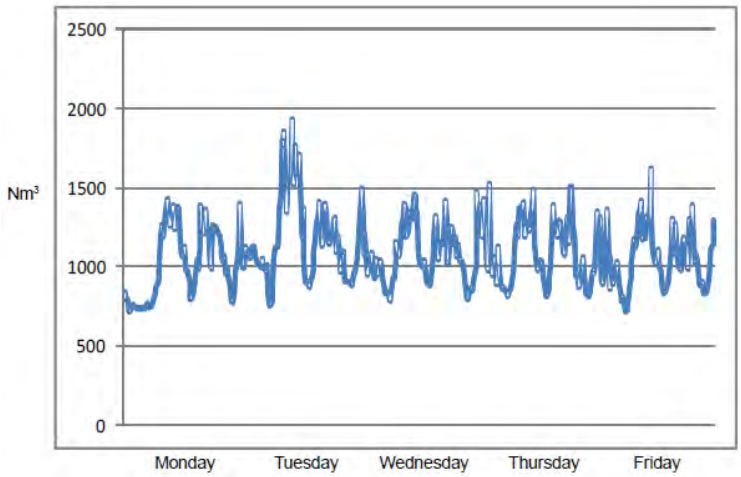


Figure 1: Total Use of Four Operating Compressors at an Underground Coal Mine
(Nm3 for air compressors refers to m3 air intake volume per minute at 0ºC, 760mm Hg in dry state. The charts are based upon measures at 15 minute intervals over five days)

If an effective and sufficiently cheap method could be found for anchoring wind turbines upon the potentially mobile material in the large waste tips adjacent to UK coal mines, these could provide ideal artificial 'hills' for a line of three or four turbines in otherwise open country. Elsewhere, the Slovenian site is situated close to high mountains where the down-draughts of air provide wind energy potential and plans may be encouraged by the grant support available in some of the newer EU states.

Water is always in use at mines and algae cultivation might be an option, especially where an adjacent coal-fired power station provides carbon dioxide. Ground source heat pump technology has been used in abandoned mine water and has potential for abandoned coal mines close to heat loads, such as an existing operating plant at a Hunosa site that heats the town and has been studied during the project.

Exploitation and Impact: Where a company has the incentive to invest, renewable energy penetration has positive payback for both the mine and the community. The concept of the smart grid on-site energy management system, with the necessary control, communications and inverters, lends itself to efficient future control and integration of these potential renewable energy resources in a mining operation and is currently being studied in a follow-on RFCS project entitled M-SMART GRID.

Task 1.3 Economics of Low Carbon Mining

To report on the economics of 'low carbon' mining.

Activities and Discussion: A study has been made of the global economic environment, relevant technologies and financial enablers, from the carbon credit markets that have developed following the Kyoto agreement to a range of national incentives. Detailed study was made of the UK and Poland as two EU countries with very different recent histories to contrast economic incentives and barriers. Coal will continue to provide a large part of the world's power requirements and parts of the world appear to be economically active once again with natural resources in demand. However, much of this activity relates ultimately to western markets and significant politico-economic challenges have emerged in the USA and especially in the Eurozone, while inflationary pressures are indicating that the Chinese economy may be over-heating. Thus the macro-economic background to general demand and political will and resources for subsidising environmental initiatives is unpredictable. To reduce greenhouse gas impact from coal mining, the primary areas for control are the emissions of methane and the high and uneven energy consumption, primarily electricity. This requires proven technologies with acceptable risk to address these areas and sound cost/benefit projections for investment decisions. These projections will be most acceptable if based upon clear energy cost savings without recourse to public money that may bring conditions, but a range of possible financial interventions also currently exists.

In a global context, most mining companies consider gas drainage to be a mining cost, although many companies have accepted the benefit from investing in the proven and viable technologies of gas engines. However, if, as appears likely, climate change mitigation and clean energy become an intrinsic part of the value chain, including ethical investment pressures at shareholder level, mine operators will need to take a more holistic view of these factors. An emergence of proven VAM mitigation and energy management technologies would start to create environmental 'champions' to emulate within the sector. Underground coal mines have been closing in EU countries, and the issue of abandoned mine methane (AMM) has become increasingly significant. This is an example where government incentive can be usefully deployed, but a country that may have the greatest number of abandoned mines, the UK, does not do so, unlike France and Germany. It is estimated that in the UK there is potential for at least another fifty recoverable energy plants with a total capacity of around 150MW.

Conclusions: Speculative investment scenarios were explored that indicate that, provided energy prices continue to rise, some of the concepts under investigation within this project should provide viable approaches to a cleaner underground coal mining industry if the technologies can be shown to work well. Appendix 1 provides a summary of the overall economic influences and enablers. The comparative review of the UK and Poland showed that a great deal more grant-based financial support and carbon credit opportunity is available in an 'economy in transition' such as Poland than in the already 'industrialised' economies of the EU, where the major state incentive is in the form of quota targets backed with financial disincentives to failure. A number of direct financial subsidy mechanisms exist in Poland (and probably in similar EU countries such as the Czech Republic), that

may clearly be related to themes in this project such as renewable energy, cogeneration and methane oxidation.

Exploitation and Impact: There has been no direct impact, but this work supports understanding for the overall project.

Task 1.4 Define an Inventory of Methane Emissions from Coal Mines in the EU

To produce an inventory of CH_4 emissions from underground coal mines in the EU, with more detailed study in Poland and Spain.

Activities and Discussion: The inventory of methane emissions from active and closed mines for years 2001-2010 has been produced for the EU. A more detailed inventory (i.e. number of mines, coal production and methane emission) for Poland, Spain and UK has been carried out. It should be noted that the values of methane emission have been assessed on the basis of domestic methodology. The methane emissions from Polish hard coal mines have been precisely assessed based on the Polish methodology with respect to the guidelines of 2006 IPCC Guidelines for National Greenhouse Gas Inventories - Volume 2: Energy (see Task 2.1).

Conclusions: Appendix 2 shows the data gathered, including direct enquiry with industry for countries represented in the project and open source data for others. Some further study was carried out to place this into context, such as the global figures shown in Table 1 and Figure 3, on page 18, which indicate the EU position, although uncertainty increases with more country returns and some significant countries such as South Africa do not appear to make returns. As shown in Figure 2 below, in 2010 the total methane emissions from coal mines in the major EU producing countries reported, i.e. Poland, Czech Republic, UK and Spain, comprised only 2.4% of the global total, although data for Germany was not yet available. This value corresponded to actual methane emissions of 667.99 Gg.

Exploitation and Impact: This data provides value in monitoring further trends in the large Polish industry, which remains largely coherent, while providing largely historic interest in a country such as the UK, where two large mines out of five have closed since 2010.

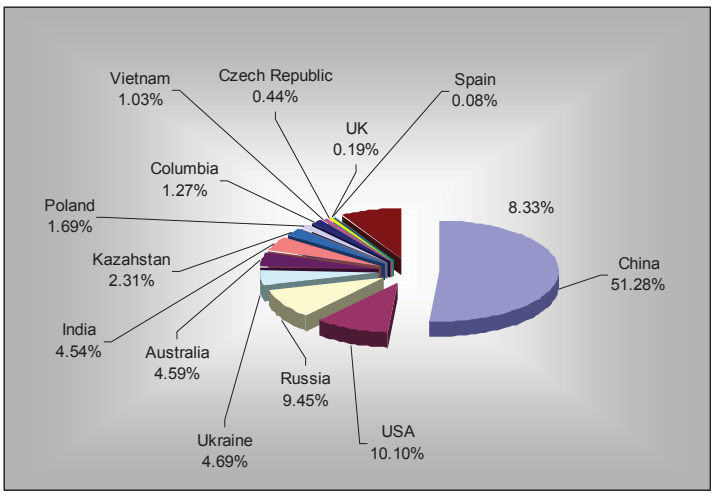


Figure 2: Comparison of methane emissions by country during 2010
(Sources of global data: www.eia.doe.gov; http://unfccc.int)

Work Package 2 Minimisation of Fugitive Gas Emissions

Objectives of WP2: *To review current technologies and evaluate methane reserves at active/inactive coal mines within the EU; to compile an inventory of coal properties relating to methane within various EU coal fields through laboratory testing; to undertake numerical modelling for methane drainage from coal seam data to determine optimum distribution of methane drainage systems for coal seams of various permeability; and to undertake trials at coal mines of methane drainage systems under various suction and time regimes.*

Comparison of initially planned activities and work accomplished: All the work has been completed as specified, despite an underground fire on a longwall planned for research that resulted in repeating much of the preparatory work for a different longwall and a very tight schedule for final reporting.

Task 2.1 Quantification of Methane Emissions

To revise and update the IPCC 2006 methodologies for estimating CH_4 emissions vented from coal mines (active and abandoned) and to detail the characteristics of ventilation air CH_4 emissions from underground coal mines; to construct a CH_4 emissions inventory at European level by applying the estimating methodologies, with a general data review and report on CH_4 content in EU coal seams; to evaluate the CH_4 reserves within selected EU coal seams to be mined in ten years; to evaluate the capture possibilities for minimising CH_4 emissions and future utilisation; to develop co-ordinated strategies for minimising emissions and utilisation of the gas; to provide an inventory of CH_4 emissions to the atmosphere from EU coal mines.

Activities and Discussion: The implementation of IPCC (2006) Guidelines has been considered and a Polish methodology used to provide a basis for assessing the Guidelines, explained in Appendix 3. Using this methodology, a final collation of obtained values was made and used in the Task 1.4 inventory. The values meet IPCC[1] 2006 requirements, but are based on actual data from methane coal mines. Regarding the exploitation of coal mine methane, drained methane piped to surface is often oxidised in gas boilers, internal combustion engines or, to a lesser extent, fuel cells (primarily solid oxide but some experimentation with catalytic phosphoric acid systems). There has been some experience in the UK of using large gas turbines with drained gas and an experiment with a micro-gas turbine using gob gas from a degas vent was undertaken in the USA, but none of these operations have proved competitive with the big internal combustion engines such as the Jenbacher™ 1.4 MWe, which uses a methane pure flow of 100 L/s at 30% purity or above to generate electricity. Similar engines are in operation in a number of places and are also used with abandoned mine methane (AMM), where a rapid decline in emission levels in the years after closure typically settles to a fairly steady rate, depending on rising rates of mine water. A UK AMM site was examined during the project. An extensive review has been undertaken of the methane impact of the total EU coal mining sector. EU emissions data covering the period 2000-2009 has been compiled, with consideration of individual European countries. Estimation of methane reserves has been performed for hard coal seams planned for extraction within the next ten years at the Knurów-Szczygłowice colliery.

Conclusions: In considering the IPCC (2006) Guidelines, AITEMIN concluded that these are fit for purpose in themselves but that many countries are believed to be using incorrect basin emission factors when the Tier 2 approach is used and use of the Tier 1 approach is recommended. To achieve a reliable inventory of CH_4 emissions in the EU, measurement methodology in each country should be aligned. Table 1 below shows the differences between the actual value of methane ventilation emission at the mines using the Polish methodology and the value calculated according to IPCC 2006. The value of the emission factor in both cases is identical. From this approach, the relative error of predictions of methane emission estimation from Polish mines 2001-2010 between the Polish methodology and that proposed by IPCC is 0.651. Regarding strategy for exploitation, it is noted that a high proportion of drained methane is still released to the atmosphere and strongly recommended that all sites where drainage from working or abandoned coal mines takes place be reviewed for suitability for gas engines. This may require some government policy intervention, such as changing the classification in the UK of mine methane to recoverable energy applicable for feed-in tariffs as with landfill gas.

[1] IPCC (2006). 2006 IPCC Guidelines for National Greenhouse Gas Inventories

Table 1: Comparison of IPCC (2006) with Polish methodology for CH4 emissions

Year	Hard coal output (Mt) in the CMM Coal Mines in Poland (1)	Emission Factor, m^3/tonne (2)	Emissions of CH_4 (Gg) according to the Polish methodology	CH_4 Emissions (Gg) by IPCC (1*2*0.67)	The difference values for emissions, Gg	Relative error of predictions by IPCC, %
2001	72.370	7.010	345.260	339.900	5.360	1.552
2002	72.130	7.284	360.902	352.015	8.887	2.463
2003	65.710	8.457	367.486	372.325	-4.839	-1.317
2004	69.170	7.640	372.534	354.067	18.467	4.957
2005	67.350	8.075	386.054	364.380	21.674	5.614
2006	64.520	8.332	360.187	360.179	0.008	0.002
2007	62.470	9.427	395.361	394.566	0.795	0.201
2008	57.540	10.288	393.074	396.621	-3.547	-0.902
2009	53.270	11.150	386.083	397.954	-11.871	-3.075
2010	52.180	11.050	375.101	386.315	-11.214	-2.989
Average emission values:		**8.871**	**374.204**	**371.832**	**2.372**	**0.651**

For the major part of the methane that is not mitigated by oxidation, data for the EU countries was examined and production and emission trends established. This data is extensive, being drawn from various sources (IEA, EPA, UNFCCC), covering open pit as well as underground mines, and is available for review if required. The chart in Figure 3 below showing the trends from underground mines provides a relevant example.

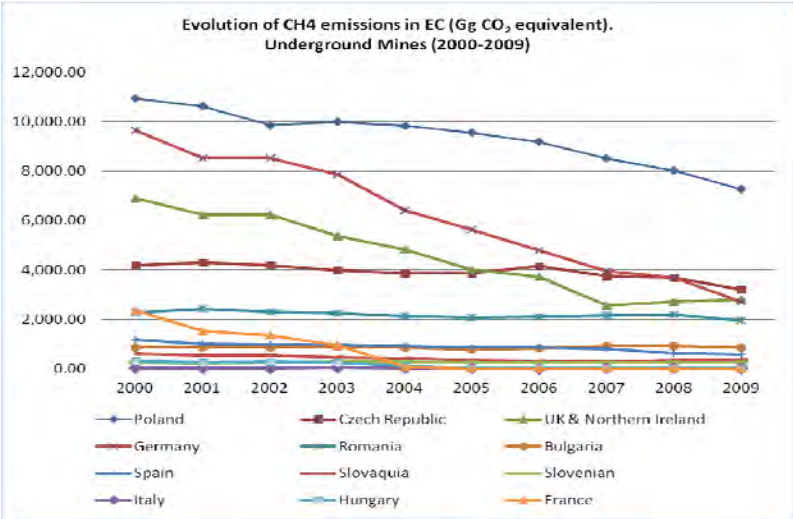


Figure 3: trends in methane emissions from EU underground coal mines $GgCO_2eq$

For the methane reserves assessment at Knurów-Szczygłowice colliery, using methods established for the Polish industry from research over several years at GIG, two seams (405/1 and 405/3) designed for exploitation during the forthcoming ten years have been analysed. The technical and operating parameters, including operational periods of the longwalls under analysis, were obtained from the operating schedules drawn up for the colliery for the years 2012-2022. The location of the panels in the selected seams, sections of exploration boreholes and results of methane content tests in the areas under analysis were obtained from colliery documents. For the calculations, average maximum methane content was obtained in individual seams along the planned extraction panel lengths and results of methane content tests in the seams present within the desorption zones of the respective longwalls, using data from exploration boreholes in development workings. Where tests were not performed or results were questionable, methane content values characteristic of neighbouring seams have been used. When simulating the vertical methane content distribution for the future longwalls, partial rock mass degassing from earlier extraction was taken into account. Appendix 4 shows the degassing ranges adopted for the predictions for each future longwall and assumed methane contents of seams, indicating whether it is primary or secondary. In general, the geological cross-sections through the methane-bearing strata were

drawn up so as to cover the whole area of the respective extraction panels. The magnitude of desorbable methane reserves should be determined for each longwall individually. It is also recommended to draw up a diagram of methane reserves. Therefore, the desorbable methane reserves have been determined for each of the longwalls presented in the first Table in Appendix 4, while the second Table shows desorbable methane reserves for the seams planned for extraction as well as of seams within the desorption zones (ranges) of the longwalls covered by the analysis. For further illustration, Appendix 4 also shows the desorbable methane reserves for longwall No. XIV in seam 405/1 in the bar chart form, broken down into individual seams and coal measures in the zone of desorption (along with the seam extracted), as well as an example of a geological cross-section. From the analyses performed, for planned extraction in seam 405/1 at the Knurów-Szczygłowice colliery, total desorbable methane reserves in seam 405/1 amount to cca. 29,376,000m^3 while in all seams and strata within the desorption zones cca. 78,544,000m^3. For planned extraction in seam 405/3, total desorbable methane reserves in the seam itself amount to cca. 17,865,000m^3 and in seams and strata within the respective desorption zones cca. 41,747,000m^3.

Exploitation and Impact: The reviews of methane emissions and mitigation technologies reinforce the need for aligning reporting standards, with the major Polish industry having already adopted standards to a higher accuracy than the norm, and the imperative to maximise installation of oxidation systems where possible, notably gas engines. Data for Knurów-Szczygłowice colliery will be very important to the mine and is an enabler for pre-mining drainage in some seams, as described in the remaining parts of this Work Package.

Task 2.2 Laboratory Testing

To provide a catalogue of methane content, porosity and strength for various types of coal; to define a method and to provide testing results for the geophysical location of highly gas-saturated coal seam zones.

Activities and Discussion: Comprehensive laboratory and underground investigation has been carried out in order to assess selected parameters of coal, i.e. compressive strength, porosity and methane content, in the KWSA mines in order to establish a database. Samples from the Velenje mine were also transported and tested. The aim to provide geophysical location of in-seam methane was in the event carried out under Task 2.3 below due to the relevance to the other work therein.

The Upper Silesian Coal Basin had originally been formed by the Variscan orogeny. Having been the mountains foredeep, it was transformed at the later stages of formation into the intermontane depression as a result of uplifting of the Myszków-Kraków fold zone. The depression was developed on the Upper Silesian Precambrian Massif and became filled with the coal measure formations composed of the assemblage of molasses stages. The carboniferous rock mass is heterogeneous and anisotropic. Heterogeneity of the rock mass relates to changes in its properties depending upon the observation point, while anisotropy relates to changes in the properties depending upon the direction. The carboniferous rock mass in the Upper Silesian Coal Basin (USCB) is featured by cyclic (repeatable) sedimentation. The cyclic sedimentation for carbonaceous formations can be expressed by particular sequences of lithological rock types and bedding types. The cyclothemes are differentiated in particular parts of the basin. Consequently, the carboniferous rocks, including coal, are featured by different properties in the vertical profile, according to the strikes.

Figure 4 shows the planned scope of investigation and the actual testing performance delivered.

Figure 4: Schedule and actual performance of coal sample testing

The database of test results is extensive. A summation of illustrative outputs is presented below.

Conclusions: The coals were examined in various lithostratigraphic groups of productive carbon and at different depths. There are bright, semi-bright, sometimes dull coals which consist of carbonates and sulphides. There were hard coals from ninety-three beds in seventeen mines of the Upper Silesian Coal Basin (USCB) and at different depths of deposition and various strata examined, such as: Cracow Sandstone Series - Laziskie beds; Siltstone Series - Zaleskie beds; Upper Silesian Sandstone Series - Rudzkie beds; Upper Silesian Sandstone Series - Siodlowe beds; Paralic Series - Porebskie beds; Paralic Series - Jaklowieckie beds. The coals were located at the depth of 288 – 1200 m.

Porosity (open, closed, fracture), methane content and uniaxial compressive strength vary with respect to age and depth. Examined coals are characterized by effective porosity (open) within the range from 0.68 to 12.48%, porosity closed: 1.24-14.92 %, porosity fracture: 3.29-18.17% (Table 2). From a chronostratigraphic viewpoint, shifting has been observed of the upper and lower limits of variability intervals of porosity towards higher values for the youngest coals. Higher values of porosity are characteristic for coals from younger stratigraphic groups (the Cracow Sandstone Series and the Mudstone Series). With the increase in depth, in general, there was a decrease in porosity.

Table 2: Summary of porosity & compression strength of coals by stratigraphic group

Stratigraphic groups	Depth D [m]	Uniaxial compression strength R_c [MPa]	Effective porosity p_o [%]	Closed porosity p_c[%]	Fracture porosity p_f [%]
Laziskie Beds - *seam 200*	345 - 684	19.6 - 51.5	5.90 - 12.48	2.20 - 10.27	10.47 - 17.33
Orzeskie Beds - *seam 301 - 326*	375 - 578	12,1 - 34,4	4.89 - 8.33	3.07 - 8.25	11.10 - 13.10
Zaleskie Beds - *seam 327 - 406*	530 - 974	9.7 - 28.3	1.63 - 7.95	1.68 - 10.78	3.99 - 12.97
Rudzkie Beds - *seam 407 - 419*	288 - 986	8.1 – 26.7	1.46 - 6.37	1.24 - 14.79	4.34 - 18.17
Siodlowe Beds - *seam 500*	330 - 922	8.4 – 28.2	1.07 - 6.37	1.62 - 14.92	3.29 - 16.06
Porebskie Beds - *seam 600*	720	11.5	1.40	2.67	4.07
Jaklowieckie Beds - *seam 700*	1103- 1140	10.6 - 22.1	0.68 - 2.38	6.33 - 11.81	4.46 - 12.94

Uniaxial compressive strength ranged from 7.6 to 51.5 MPa, most frequently within 10-30 MPa. With increase in compression strength, the value of porosity in particular stratigraphic groups generally decreases. However, no regular changes in mean uniaxial compressive strength with the increase in the age of subsequent stratigraphic groups were observed (see Appendix 5). Heterogeneity of the petrographic structure of coals influences their strength. A decrease in the fraction of the hardest microlitotypes (durain, clarodurain, and carbominerite) with depth results in a decrease in compression strength. For bright coal and semi-bright coal, an abrupt drop in compression strength with deposition depth of coal was observed. Methane content ranged from 0.000 to 21.100 m^3/mg (Table 3). At some mines and the related coal beds there were clear differences in methane content due to changing geological conditions in areas of a mine, which permitted classification of these beds into methane content categories (from I to IV). The coals examined from Laziskie, Orzeskie and Porebskie Beds fall into category I methane content. The other coals, Zaleskie, Rudzkie, Siodlowe and Jaklowieckie Beds, fall across all the categories (from I to IV). In general, in relation to the younger stratigraphic cells, in the older coals (Rudzkie, Siodlowe, Jaklowieckie Beds) methane content increases with the age of strata. There was no clear influence of depth of deposition of strata on methane content discovered. Graphical distributions of values for two examples are shown in Appendix 6.

For the lignite samples from the Velenje mine in Slovenia, porosity ranged from 16.42% to 17.26%, methane content was very low (from 0.002 to 0.003 m^3/Mg) and UCS was about 13.4 MPa. This information is useful to the company.

Table 3: Summary of UCS values and CH_4 content by stratigraphic group

Stratigraphic groups	Depth D [m]	Uniaxial compression strength R_c [MPa]	Methane content [m^3/Mg]
Laziskie Beds *seam 200*	345 - 684	17.5 - 51.5	0.000 - 0.002
Orzeskie Beds *seam 301 - 326*	375 - 590	7.6 - 34.4	0.000 - 1.078
Zaleskie Beds *seam 327 - 406*	450 - 985	9.7 - 28.3	0,008 - 21.100
Rudzkie Beds *seam 407 - 419*	288 - 1110	8.1 - 26.3	0.000 - 9.682
Siodlowe Beds *seam 500*	325 - 1028	9.6 - 33.9	0.000 - 18.800
Porebskie Beds *seam 600*	765 - 853	13.1 - 27.8	0.000 - 2.477
Jaklowieckie Beds *seam 700*	820 - 1200	10.6 - 20.2	0.236 - 9.211

Exploitation and Impact: The Polish data has been published and remains as an important resource for operations and research in the Upper Silesian Coal Basin, with particular relevance to planning for methane drainage operations, including pre-mining drainage and possibly virgin coal bed methane exploitation.

Bukowska M, Sanetra U, Wadas M (2012); Chronostratigraphic and Depth Variability of Porosity and Strength of Hard Coals of the Upper Silesian Basin; Mineral Resources Management; Vol. 28, no. 4.

Task 2.3 Numerical Modelling

To provide a computer programme and modelling results of methane drainage from coal seams of various permeability; to provide a computer programme and modelling results of methane and coal outburst with warning values for physical indices measured.

Activities and Discussion: Pre-mining methane drainage modelling Numerical modelling was undertaken to find optimum methane drainage parameters for pre-mining drainage, such as the length, diameter and spacing of in-seam boreholes and suction gradients for the main types of coal, identified from borehole sampling by methane content, sorption properties, porosity and permeability. This was used as design data for trials in Task 2.4. Petrel software (ver. 2010.1) was used to build a static model of a coal seam, while the demethanation process simulation was performed using the compatible ECLIPSE compositional simulator version. Following trial work for Szczygłowice and Brzeszcze collieries, a range of numerical simulations of the methane drainage process in a model based on a longwall in Brzeszcze was performed. Initially, longwall 193 in seam 510 which showed the highest methane content (up to 16.1m^3 CH4/t) had been chosen and modelling started. However, due to a spontaneous combustion event leading to a fire and sealing from active workings, a new site was selected for trials at longwall 121 in seam 364 and further borehole drilling carried out to provide input data. Input parameters for this seam, from which the total gas resource was calculated, are shown in Appendix 7, which also shows a table of outputs from the simulations for five different drainage options, images from which are shown in Appendix 8:

Case 1: 26 boreholes drilled from gateroad 551 and 25 boreholes drilled from ventilation gallery 552 drilled perpendicularly to the axis of the gallery. One year of operation.

Case 2: 25 boreholes drilled from tailgate 552 perpendicularly to the axis of the excavation. One year of operation.

Case 3: 30 boreholes drilled from maingate 551 perpendicularly to the axis of the excavation. One year of operation.

Case 4: As case 3, but in this case boreholes produced methane for five months and then the gradual disabling of boreholes commenced according to the progress of mining (5 metres per day).

Case 5: As case 3, but in this case boreholes produced methane over 842 days.

Based on the results of the modelling and GPR measurements (see below), a layout of boreholes for the underground trial was developed, as shown in Figure 5.

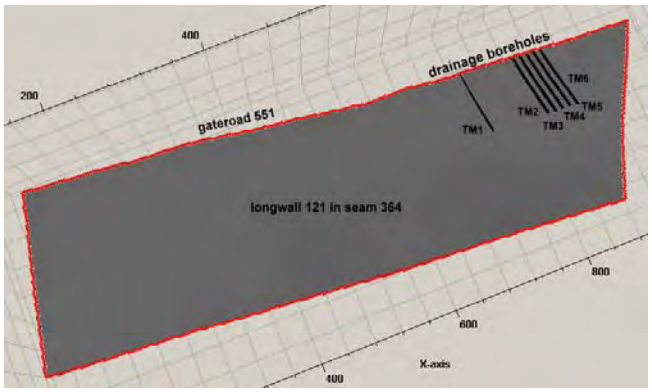


Figure 5: Borehole layout TM1-TM6 longwall 121 developed for field trial

Outburst modelling: Model development for coal outbursts was carried out using Particle Flow Code (PFC2D from Itasca) and a classification of coal and methane bursting tendency of coal seams was developed. From previous work to simulate movement and failures such as in roadway supports and floor heave, the hydraulic borehole penetrometer (HBP) had been developed at GIG to log resistance strength at points along a borehole. The HBP equipment can also be used in horizontal boreholes of standard diameter and therefore may be used for testing the strength of a coal seam in the close proximity to an opening in order to evaluate the maximal distance to which the coal seam is liable to be subject to coal and gas outburst. The continuous rock penetration log (rock strength profile) may be introduced to stress analysis equations including those used with PFC modelling. The model was 35 m long (including 17 m coal seam section and 18 m of empty roadway), 3.5 m high (from the floor to the roof) and 1.0 m wide. An example of outburst phases in the vertical cross-section of the model is shown in Appendix 10. The gas type is introduced to the model by density and viscosity, the flow is arranged through the gas pressure gradient between the left and right sides of the model, while at the right side (within an opening) it is always equal to one bar (atmospheric pressure). Information about local outburst risk level in the mine may be provided from laboratory testing of coal properties such as original porosity of coal and the uniaxial compressive strength of coal estimated with laboratory samples. As the measure of destructive power, the maximum kinetic energy may be taken as related to the mass of the model which was approximately 77 Mg of coal.

Detection of high CH_4 pressure zones by geophysical measurement: Ground penetrating radar 'in situ' test measurements were performed in a KWSA longwall and analysed in order to detect in-seam high methane pressure zones, together with some additional work at the HUNOSA Montsacro mine.

At KWSA, the GPR measurements were performed in the maingate and in longwall 121 in various time windows, the highest being 850ns for the 100 MHz radar transducer. The overall length of the conveyor gallery was about 600m and the data were recorded in sections with length varying from sixty to a hundred metres. The data from tests revealed that because of increasing attenuation of electro-magnetic energy with travel distance in the coal seam, the best way to present results is to apply a logarithmic gain curve to the data. This type of gain causes both the strong and relatively weak reflected signals to be visible on radar sections. The strongest reflected signals were reaching the antenna at a time close to 20ns. The segment recordings in the 850ns window are shown in a comprehensive radarogram of the coal seam in Appendix 11. The top display shows the raw data with HBG filter applied (Horizontal Background Removal – filtration of the data by moving average) and the lower display shows data processed by the Hilbert transformation in magnitude domain. Preliminary data analysis indicated that the fractured part of the seam (near the work side) is visible as a reflective horizon at a distance of 1m-2.5m from the side wall. In the deeper parts of the coal seam, a relatively strong group of reflections from irregularly shaped boundaries were recorded in many places, which probably correspond to the location of the anomalous regions of different physical and structural properties of coal. In these zones, the methane content in the coal (both adsorbed and free) may be elevated relatively to the ambient. Some additional survey was undertaken where unusual effects such as water movement affecting methane were observed, the boundaries of two such anomalous zones being shown in the seam map in Appendix 11, marked by blue arrows. The structure of the coal seam in the near wall face zone was obtained by measurements performed on a 9m profile located between the wall corner and a protective dam at

the conveyor. The radarograms of this profile are also shown in Appendix 11 as raw and processed data.

Underground GPR tests were also carried out at the HUNOSA Montsacro mine in Spain. At this mine, pressurized water is injected to the face to promote degasification just prior to mining and a theory was tested to discover if consequent changes to electrical properties might provide an opportunity to identify methane in the virgin seam by comparison with the degasified seam. With a 400 MHz antenna, nineteen scans were performed prior to the injection of water and repeated after the injection of water and consequent degassing. With a 200 MHz antenna, eleven GPR scans prior to degassing and ten scans after degassing. Examples of scans showing before and after comparisons are shown in Appendix 11.

Conclusions: Pre-mining methane drainage modelling: The Eclipse software simulations provided indications of the efficiency of the process of demethanation. There was no significant influence of the diameter of the drainage boreholes on the methane drainage process efficiency and assumptions regarding dependence of cumulative production of methane upon applied pressure in the drainage boreholes were confirmed. A significant increase in process efficiency was observed when the applied negative pressure equalled 0.25 bar. Much better results of simulations were obtained for the use of drainage boreholes positioned perpendicularly to the axis of the excavation. The best results for Brzeszcze were obtained from case 1, which uses an expensive perpendicular layout of boreholes drilled from two galleries with a length of 120m, diameter 76mm, pressure of 0.5 bar and with simultaneous activity of all boreholes for a period of one year (the efficiency of the methane drainage process achieved 45.8%). During the subsequent field tests (Task 2.4), CH_4 out-flow was measured for boreholes TM2-6 over 271 days and the comparable numerical simulations of methane drainage were examined. The results of the measurements and simulations are presented in Appendix 9. Some adaptation of initial parameter settings was made to obtain a closer match to actual conditions. It may be observed that the simulation is fairly close to the field data, concluding that the mathematical modelling and reservoir simulation method offers a useful tool for improving drainage efficiency, but requires iterative work with actual data to refine the process.

Outburst modelling: Three main factors affect the occurrence and severity of coal and methane outbursts in coal mine, namely methane content in coal, original porosity of coal and compressive strength of coal. High methane content, high porosity and low compressive strength of coal create conditions for coal and methane outburst generation and long horizontal transportation of dispersed coal masses. The high dynamic energy of coal and methane outbursts is generated mainly by violent desorbing of methane from breaking coal masses. A relationship was shown to exist between the original porosity of coal and the maximum kinetic energy of an outburst, with three classes of risk indicated (porosity – low <7%; medium 7-10%; high >10%). An index CII (Coal Instability Index) is proposed for use in a mine, related to the distance of the calculation point from the heading face. This index is received as a quotient of the horizontal stress (s11) and the vertical stress (s22) and the result being divided by the distance of the point selected from the heading face (in metres). Where vertical stress measurement in the coal seam is difficult, the general classification method may be used, based upon the three parameters of methane content in coal, original porosity of coal and the compressive strength of coal within the seam. A reference table for this is included in Appendix 10.

Geophysical measurement: The direct observations provided a first indicator that the radar survey can be considered as a tool for detection and contouring the regions within the coal seam structure which may have high methane content. Radar anomalies detected in a seam should be taken into account in analysis of numerical models simulating the methane distribution ahead of the face. However, the results of the GPR method are highly dependent on the lithology of rocks in contact with the coal bed. In all three data sets collected in underground galleries the coal was characterized by very low humidity. Also, geological conditions were similar with respect to roof and foot layers lithology (siltstones – clayish rocks). In such conditions the coal seam forms a waveguide for electromagnetic waves due to the structural geometry and electrical properties of beds. As reported in Task 2.4 below, the subsequent field test with borehole TM1 supported the hypothesis that this offers a useful method of methane identification. The results from Montsacro mine were inconclusive, in that there were changes in the wave reflections following degassing but it was concluded that the outcomes were not sufficiently robust to provide a reliable estimation of methane presence.

Exploitation and Impact: The pre-mining methane drainage simulation showed promise as a method of maximising efficiency once refined and the GPR experiment at KWSA offers a real opportunity for further development and use.

Kidybinski A (2011); The effect of porosity and the strength of coal on the dynamics of coal and methane outburst – the BPM modelling; Archives of Mining Sciences; Vol 56 No 3 pp 415-426

Kidybinski A (2012); Mierzalne czynniki zagrożenia wyrzutem węgla i metanu w przodku wyrobiska korytarzowego – wg badań na modelu numerycznym, Przegląd Górniczy nr 1/2012, str.20-26. (Measurable parameters of coal and methane outburst hazard in the heading face of a roadway; Mining Review; No.1 pp.20-26, – in Polish)

Task 2.4 Field Testing & Optimisation

To provide the results of coal mine trials and the optimisation of methane drainage technology; to provide an economic analysis of pre-mining methane extraction and guidelines for various coal mining conditions.

Activities and Discussion: In order to plan the underground trials of methane pre-drainage, mining conditions, coal seam parameters and extraction plans at KWSA collieries were analysed for extraction time, methane content, coal porosity, conventional methane drainage intentions, suitability for methane pre-drainage and drainage results monitoring. Two collieries were selected as possible sites for underground trials, Szczyglowice Colliery and Brzeszcze Colliery. The assumption was made that methane pre-drainage is methane drainage before longwalling or out of abutment pressure influence (in the Upper Silesian Coal Basin, normally about 150m ahead of the longwall). Potential sites are listed in Appendix 12. Longwall 193 in seam 510 which shows the highest methane content (up to 16.1m^3 CH4/t), where conventional methane drainage had been planned, was selected for trials. Boreholes were drilled but unfortunately, due to a fire caused by spontaneous combustion, this panel had to be sealed with fire dams. A new site was selected at longwall 121 in seam 364. Further borehole drilling was then carried out in longwall 121 panel to provide input data for the numerical modelling (see Task 2.3 above). Based on the results of the modelling, a layout of boreholes for methane drainage was developed. Six drainage boreholes 76mm in diameter and 100m in length (except TM1 and TM4 which were shorter due to technical problems) were drilled from “chodnik tasmowy sc. 121”, perpendicularly to the gateroad rib (see Appendix 12). Borehole TM1 was drilled to a distance of about 100m from the longwall termination line towards the longwall and, although this is a drainage borehole, the suspected high methane pressure zone detected by GPR (see Task 2.3) was also to be investigated. The remaining 5 boreholes TM2-6 were drilled close to the longwall termination line in 10m intervals boreholes TM2-4 towards the longwall (inby) with TM5-6 outby. During pre-methane drainage tests the CH_4 concentration in %, its out-flow in m^{3}pmin, pressure in a borehole, negative pressure and distance from the longwall were measured daily, separately for borehole TM1 and for boreholes TM2-6 in total. This started on the 24th April 2013 when the distance between longwall and drainage boreholes was more than 210m for the borehole TM1 and from 285m to 325m for the boreholes TM2-6 and were completed on 21st January 2014 when the last drainage borehole was blocked. The results of the measurements are presented in Appendix 13.

Conclusions: It was observed that CH_4 concentration in boreholes varied from 100% when measurements started to 0% when the rock mass around borehole was fractured as a result of abutment pressure and air entered. The first significant drop of CH_4 concentration was observed on day 48 of the trial, when methane concentration in boreholes TM1 – 4 dropped from the level of 85-90 % to 50-65%. The second was observed on day 100. Recorded drops were correlated with the negative pressure increases (respectively from 120 to 250mmHg and from 60 to 120mmHg) but not with distance from longwall 121 face. Regarding the most important parameter of CH_4 out-flow, maximum values were 0.5m^{3}min for borehole TM1 and 0.9m^{3}min for boreholes TM2-6 (0.18m^{3}min for single boreholes). These values were recorded for boreholes TM2-6 on day 149 of the trial when CH_4 concentration in boreholes reached 95% and negative pressure was fairly low (20mmHg). On day 152 a decrease to 0.5m^{3}min accompanied an increase of negative pressure to 150mmHg. In the following two days 153 and 154, negative pressure decreased and both the out-flow and CH_4 content again reached the maximum values (0.8-0.9m^{3}min and 80-85%). Although some interesting situations were observed, clear correlations between measured parameters were not found. This CH_4 out-flow is dramatically lower than out-flow recorded during classic methane drainage, when typically 6-7 boreholes about 60m long, drilled from the gateroad to above the gob area, give an out-flow of more than 8.6m^{3}min (i.e. 1.2-1.4 m^{3}min per single borehole). The efficiency of methane pre-drainage here is about ten times lower when compared to the classic

method and shows that this technology cannot be successfully implemented in the conditions of Polish hard coal mining. These low out-flow rates deeply affect the economics. The cost of the trial was calculated to be 428,000 PLN (€102,720), including 365,000PLN (€87,600) of direct costs of drilling, borehole piping and drainage parameters control and 63,000PLN (€15,120) indirect costs of methane pre-drainage design, laying and relaying pipelines, maintaining the network and equipment, including the methane drainage station and cost of methane utilisation. The cost of methane pre-drainage comes to 1.69PLN/m^3CH_4 (€0.41), compared with the classic methane drainage cost at Brzeszcze Colliery of 0.43PLN/m^3CH_4 (€0.10) for the years 2009-2013.

Ground Penetrating Radar: CH_4 out-flow at borehole TM1 (0.3037 m^3min) is significantly higher than at TM2-6 (0.4956 m^3min for 5 boreholes and 0.0991 m^3min for one borehole). This was the zone of high CH_4 pressure assessed by GPR (see Task T2.3), thus supporting the usefulness of GPR for detecting high concentrations of CH_4 within a seam.

Exploitation and Impact: The results of this underground trial indicate that pre-mining methane drainage technology in this form is not suitable for the conditions of Polish hard coal mining. There will be further attempts to improve viability with additional measures, i.e. hydraulic fracturing, open or cased hole cavitation, high pressure water jetting, use of explosives (see GasDrain - new RFCS project commencing July 2014).

Work Package 3 Mine Site Energy Generation

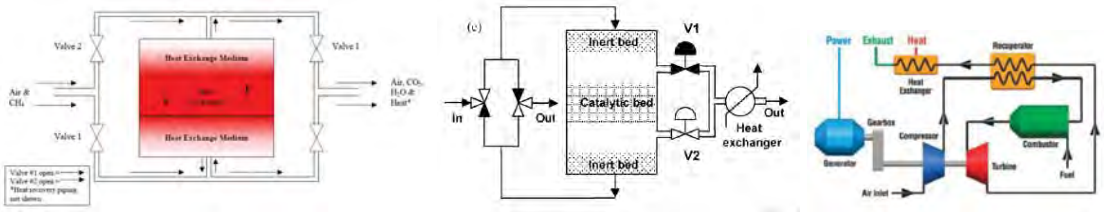
Objectives of WP3: *To examine the potential for the utilisation of coal mine ventilation air methane (VAM); to evaluate the potential for co-generation and tri-generation; and to evaluate the techno-economic criteria for candidate schemes.*

Comparison of initially planned activities and work accomplished: The work in WP3 has been accomplished. As a thorough review of attempts to harness gas turbines directly to ventilation streams revealed very little likelihood of success, the work in this area has been directed to a completely novel approach using standard machines running on imported gas.

Task 3.1 Examine various Ventilation Air Methane Utilisation Technologies

To review current combustion technologies; to undertake a survey of heat loads suitable for tri-generation applications; to investigate and to characterise catalytic foam flow reversal reactors.

Activities and Discussion: A detailed review has been undertaken of options for treatment of ventilation air methane (VAM). Three main technologies have been considered, the thermal (TFRR) and catalytic (CFRR) versions of the flow reversal reactor (FRR) and the gas turbine, as in Figure 6.



(a) (b) (c)

Figure 6: The three main VAM oxidation technology options

a) Thermal Flow Reversal Reactor TFRR b) Catalytic Flow Reversal Reactor CFRR c) Gas Turbine

The technology that has led the way for VAM oxidation is the FRR, operated by periodically reversing the flow direction of the exothermically reacting gas mixture over a heat retaining medium, which may include a catalyst (CFRR). The TFRR with its self-sustaining, high temperature reaction process is the most mature technology with a number of manufacturers offering products developed for volatile organics. Enquiries into gas turbine experiments suggest that this is not suitable for direct application to VAM (although this has led to some novel work described in Task 6.1).

A survey of known heating or cooling loads was undertaken, encountering some difficulties with confidentiality of data. The outcome is shown in Appendix 14. In general, apart from a few Polish deep mines, significant loads are not apparent in the EU and deliberate co-location of heating or cooling intensive industries next to mines would be the only viable approach.

Using a bench-scale CFRR, the combustion of methane has been studied with random packing, honeycomb monolith and foam beds. It was concluded that the best bed type is the honeycomb monolith due to higher thermal inertia for inert beds, although foams can also be used as catalytic beds. Tests were developed with different methane concentrations (1000-2000ppm) to observe and to determine kinetic parameters. Initially, a stabilization phase was necessary for the catalyst at a high temperature until constant conversion. Experiments were performed at temperatures from 350ºC to 470ºC and outlet methane was analysed by gas chromatography. The influences of the main operating parameters for this reactor, i.e. switching time and methane concentration in the feed, have been investigated.

MATLAB™ models have been built for the FRR and the gas turbine, images from which are shown in Appendix 15. Further physical experiments were performed in the laboratory CFRR to investigate the effect of humidity on the reaction and to reinforce the model.

Conclusions: The TFRR is the only VAM oxidation technology with any significant industrial uptake, based upon carbon credits, as CFRR, though in several ways superior, is currently expensive. Gas turbine experiments have mainly been operationally unsuccessful and have relatively high concentration requirements. This review did, however, lead to a novel gas turbine concept explored in Task 6.1.

For the specific CFRR research objective, light-off curves were obtained for different catalyst loadings and methane inlet concentrations. The results were successfully fitted to the models Mars–van Krevelen (MVK), Langmuir–Hinselwood and first order kinetics. The MVK model provided the best fit but the first order kinetic model can also be used for practical purposes. It was shown that in the presence of humidity, for a given switching time, higher methane concentrations (leading to higher adiabatic temperature rise, about 50ºC, corresponding to 1800 ppm of CH_4) are needed for stable operation with a Pd catalyst, enabling control concepts, such as decreasing switching time with increase in H_2O (see Task 6.1).

Exploitation and Impact: Two papers have been published on the CFRR science in academic journals by the team at the University of Oviedo. An industrial contact is available if a mine has the specific conditions for co- or tri-generation and wishes to pursue a design.

Marín P, Ordóñez S, Díez F V (2011); Performance of silicon-carbide foams as supports for Pd-based methane combustion catalysts; Journal of Chemical Technology and Biotechnology; doi: 10.1002/jctb.2726

Thomson C, Marin P, Diez F V, Ordóñez S; Evaluation of the use of ceramic foams as catalyst supports for reverse-flow combustors; Chemical Engineering Journal (in press, doi: 10.1016/j.cej.2013.01.080)

Task 3.2 Appraisal of current Sequestration and Gas Cleaning Technologies

To investigate pressure-swing and temperature-swing adsorption systems; to investigate gas-cleaning systems; to develop enhanced procedures for the operation of the flow reversal reactor with coal venting gases.

Activities and Discussion: Approximately 70% of the methane emitted from coal mines is released as the ventilation air methane (VAM). Unfortunately, due to the low methane concentration (0.1–1.5%) in ventilation air, mitigation or utilisation is very difficult. Effective methane separation and concentration as pre-treatment for further oxidation could be an important enabler for CH_4 treatment. Several separation technologies, such as absorption, adsorption, membrane and cryogenic separation have been studied. Because of the low energy requirement, cost advantage, and ease of applicability over a relatively wide range of temperatures and pressures, adsorption separation attracts particular interest. An example of this application is pressure swing adsorption (PSA), which requires two parallel beds with the adsorbent, in such a way that while adsorption is carried out at atmospheric pressure in one of the beds, the other regenerates at lower pressures. The same process can also be performed varying the temperature, in the so-called temperature swing adsorption (TSA). However, adsorption is not yet considered attractive for large-scale separation of CH_4 because the capacity and CH_4 selectivity of available adsorbents is still low.

Thus, the main objective here is the search for an adsorbent with high capacity and selectivity for CH_4 adsorption. Among the different materials proposed for this purpose (carbon nanotubes, silicas or zeolites), isoreticular metal organic frameworks (IRMOF) have been proposed as being especially promising for adsorption of different permanent gases (N_2, Ar, CO_2, CH_4, and H_2). These materials, constituted by metal ions located in vertices joined by organic linker molecules to form the IRMOF structures, are characterized by high surface area and pore volume, with wide possibilities of structure variations. Specifically, three IRMOFs were selected, constructed by tetranuclear Zn_4O clusters connected by rigid dicarboxylic linkers to create a cubic framework: IRMOF-1, IRMOF-8 and IRMOF-10. The linkers, terephthalic acid (IRMOF-1), 2,6-napthalene dicarboxylic acid (IRMOF-8) and 4,4′-diphenyl dicarboxylic acid (IRMOF-10), respectively, confer different open windows to the material. However, as the clusters are identical and no functionalities are introduced into the structure, variations in adsorption selectivities are mainly caused by differences in the organic units.

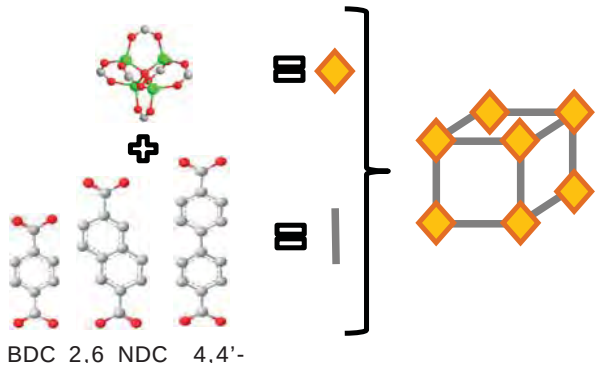


Figure 7: Cubic structure of IRMOFs; green = Zn grey = C red = O

These three materials were prepared and characterized for chemical and morphological properties. In this way, the pore volume and surface area was obtained by nitrogen physisorption; the crystallinity was confirmed by X-ray diffraction; the thermal stability of the materials was obtained by thermogravimetry; the chemical composition by ICP, and the affinity of the different structures for the adsorbate was studied by inverse gas chromatography. Adsorption behaviour for CH_4, and other small molecules present in the vent gases, was studied in a thermobalance.

Additionally, a study was made of 'plasma reforming' to enable gas cleaning with potentially beneficial by-products. Plasma is defined as an electrically neutral medium of positive and negative particles. Many of the particles are highly reactive chemically, hence the potential of plasma as a favourable method for bringing about chemical reactions. There are two distinct types of plasma known as equilibrium and non-equilibrium; the latter is particularly attractive for this application because of its low temperature. The plasma approach permits a much greater level of control over a reaction, allowing it to be fine-tuned towards generating a particular chemical product. Those chemical reactions of methane that could yield a commercially valuable chemical reagent have been reviewed. Reactions that yield a liquid or solid chemical product are preferred since this greatly simplifies the problems of separating the resultant chemical from the gas stream. Particular attention has been given to generating an atmospheric pressure plasma which would be required for reforming a continuous flow of gas. Suitable techniques include dielectric barrier discharge, pulsed corona, microwave excitation and gliding arc discharge. Potential products might include higher hydrocarbons, hydrogen, diamond, carbon nano-particles or oxygenates such as methanol or formaldehyde, managed by factors such as plasma reactor size, flow rate, catalyst use or thermal conditions. However, considerable energy input is needed that would need to be assessed.

The objective relating to flow reversal reactors is covered in Tasks 3.1 and 6.1.

Conclusions: IRMOFs are promising materials for CH_4 concentration. It was shown that the capacity for adsorption increases with the cavity size of the structures, but concurrently the selectivity for the higher volatile organic compounds also decreases with the size of the cage. Furthermore, the organic linkers of the IRMOFs have been demonstrated to be the most active sites for adsorption, thus this fact offers the possibility to vary the functionality of these molecules in order to favour CH_4 adsorption while hindering the retention of other molecules. While showing early promise at a technical level, it is noted that this is not a cheap solution.

The plasma reactor approach to gas cleaning for by-products offers some promise but would present major technical challenges that go well beyond the scope of this project.

Exploitation and Impact: Two scientific papers have been published on the scientific work with IRMOFs by the team at the University of Oviedo.

Gutiérrez I, Díaz E, Ordóñez S (2013); Consequences of cavity size and palladium addition on the selective hydrogen adsorption in isoreticular metal-organic frameworks; Thermochimica Acta. doi: 10.1016/j.tca.2013.01.007

Gutiérrez I, Díaz E, Vega A, Ordóñez S (2013); Consequences of cavity size and chemical environment on the adsorption properties of isoreticular metal-organic frameworks: An inverse gas chromatography study. Journal of Chromatography A 1274, 173-180.

Task 3.3 Evaluation of Co-generation and Tri-generation Potential

To evaluate the co-generation and tri-generation potential for the main competing methane utilisation technologies.

Activities and Discussion: The potential technologies for provision of electricity, heat and cooling from coal mine methane have been considered. With concentrated drained methane (around 35%), internal combustion engines are often employed for electricity and sometimes heat capture, even including cooling with absorption chillers at JWSA mines in Poland. For ventilation air, there have been hopes to develop a gas turbine option but trial systems in Australia and USA have been unsuccessful except for a small catalytic machine at CSIRO. Modelling related to a concept developed during this project to run standard co-generation gas turbines while burning VAM with exhaust heat is described in Task 6.1. As CFRR normally runs at too low a temperature to provide sufficient heat quality, the only direct proven option for VAM is the TFRR where sufficient energy is available. In order to go beyond simply sustaining the oxidation reaction, this requires VAM at concentrations of at least 0.45% to provide usable energy, which poses questions about improving drainage and limits the potential to less than one in ten mines worldwide, mainly situated in the Far East although the only system believed to have actually been built to a functioning level is in Australia.

Conclusions: Appendix 17 shows a generalised concept created in this project for a possible tri-generation configuration for VAM. This proposal is based upon recovery of eighty per cent of upcast air with a hood, fed to an array of TFRRs providing heat to a Waste Heat Recovery Boiler (WHRB) / steam turbine (up to 29% system efficiency but fairly expensive) and aims to achieve a reasonably closed system of steam at various pressures to generate electricity, radiator-style air heaters at the downcast shaft in winter or absorption chillers in summer. Another, cheaper option for heat processing would be the Organic Rankine Cycle (ORC), which provides only up to 14% efficiency but might be made to work at CFRR temperatures. Some technical restraints are VAM concentration to provide sufficient energy to maintain the system, the appropriateness of inputs (pressures, temperatures) to the various component devices and the proximity of shafts to keep expensive runs of insulated, pressure-resilient pipework as short as possible. See Task 6.3 for further work.

Exploitation and Impact: System design for such co-generation schemes is available through an industrial contact to industry where the unusual combination of VAM concentration and other factors exists. This formed part of a presentation of the project at the WMC 2013 conference.

Task 3.4 Techno-economic Criteria for all Candidate Schemes

To quantify the cost / benefit and the environmental performance of competing technology routes and options.

Activities and Discussion: As seen above, approximately 70% of the methane emitted from coal mines is released as ventilation air methane (VAM) at challengingly low concentrations. In this Work Package, different technologies for mine site energy generation using VAM have been researched and, although the CFRR and a novel gas turbine configuration have provided some very interesting research opportunities that are described elsewhere in this report, the clear conclusion is that the robust simplicity of the TFRR, without catalyst or pre-concentration, provides the only viable direction at the current state of the art. Knowledge of the cost of catalysts and direct enquiries with experimenters with lean burn gas turbines have supported this view.

Figure 8 shows the maximum heat recovery from the oxidation chamber of a FRR. However, as the thermal and catalytic versions have different operating temperatures, this is followed by consideration of how efficiently this may be converted to potentially usable power.

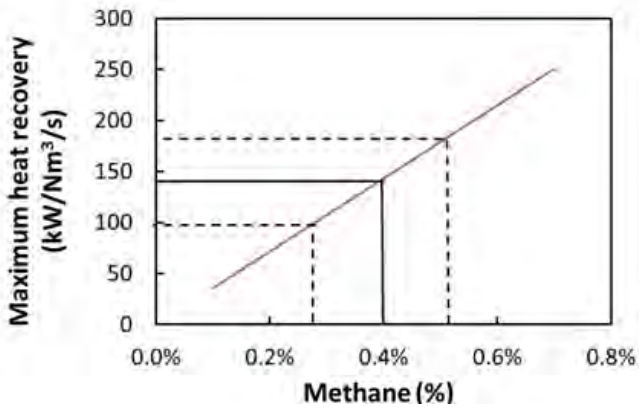


Figure 8: Maximum heat recovery by methane concentration

To assess the energy available from the heat exchanger, the Carnot cycle may be used for estimation:

$$Efficiency = 1 - \frac{T_f}{T_c}$$

Where T_f (K) is the temperature of the cold side and T_c (K) is the temperature of the hot side. The cold temperature can be considered T_f = 313 K = 40 ºC and, for the hot gas temperature T_f = 1123,2 K = 850 ºC in the TFRR oxidation chamber, and T_f = 723,15 K = 450 ºC in the CFRR oxidation chamber. This shows a working efficiency of 72% for the TFRR, compared with 57% for the CFRR.

Considering a ventilation air flow 200 m^3/s (720,000 m^3/h) the maximum heat that it is possible to obtain per Nm^3/s of VAM, and the efficiency, the power (kW) can be estimated.

Table 4: Maximum available power from VAM stream; TFRR vs CFRR

Reactor	Maximum heat from Nm^3/s VAM (nominal) (methane concentration aprox. 0,4%)	Efficiency	kW from Nm^3/s VAM (200 m^3 CH_4/s)	MWh
TFRR	136	72%	19,584	20
CFRR	136	57%	15,504	14.6

The primary drivers for current VAM projects have been the carbon markets for the reduction of GHG emissions under the UN Framework Convention on Climate Change (UNFCCC) and other voluntary initiatives and the reality is that in almost all cases of FRR use, heat is lost and carbon credit gains have supported the methane oxidation simply as a greenhouse gas mitigation exercise. The price per tonne CO_2eq during recent years (2008-2013) has been decreasing, with the most significant change being the decrease of prices between June 2008 and March 2009. This represented a very important adjustment to the future emissions rights prices in Europe, which over a period of about nine months were trading at about a third of the mid-2008 level. This change in trend during the second half of that year was due mainly to the beginning of the economic crisis, and especially the significant decrease in oil prices that took place during those months (Point Carbon, 2009).The link between the price of European emission rights and the price of oil can be seen by analyzing the correlation between the prices of the emission rights and Brent oil prices. However, although the current market for VAM destruction is soft, when carbon prices were as high as 23 euro/tonne (before 2008), the annual 300m $MtCO_2$eq of VAM emissions represented a potential annual market of $6 billion. Appendix 18 shows an estimation of global opportunities, within which Europe does not feature.

Conclusions: The TFRR remains the only viable technology at present and even this is not being installed at current very low carbon credit prices. If an effective financial subsidy mechanism should return, however, the global market remains a vast opportunity, though not the European one. Co-generation provides a prospect for improving the financial appeal at the small proportion of mines with VAM in excess of 0.45% (mainly in China), but the investment requires confidence in a long-term mine life and probably some additional support from carbon credits (see Task 6.3). The work on methane concentration and gas cleaning for useful by-products provided interesting science but at present is not at a viable stage.

More work on the techno-economic aspects is shown in Task 6.3.

Exploitation and Impact: This section provides supporting information for the other work.

Work Package 4 Improving Energy Efficiency

Objectives of WP4: To gather energy usage demand data from various sites, build a 'load-profile' database and carry out modelling/simulation work on energy load forecasting in order to determine the benefits of employing an energy management strategy; to examine intelligent demand reduction and responsive strategies for underground coal mining; to investigate the use of intelligent ventilation control for improved efficiency; to provide improved pumping efficiency using intelligent control including scope for integrating the system with demand response strategy; to investigate other short term energy efficiency strategies at the surface facilities/buildings

Comparison of initially planned activities and work accomplished: The work has been completed, and in some areas such as mine electricity demand monitoring and thermodynamic fan efficiency estimation moved ahead of the requirement.

Task 4.1 Gathering Data, Simulation & Forecasting Energy Demand of Mining Operations

To develop a comprehensive load profile database in a mine operation; to model and to forecast the regional demand profile to compare with the load curve of an underground mine to understand the effects of load-shifting and load reduction.

Activities and Discussion: This Task involved collection of mine electrical power usage data for use in Task 4.2. A formal agreement was reached with UK Coal for CSM to design and install a data acquisition system at Kellingley Colliery. The installation would, a) provide the detailed underground coal mine energy data for the project, and b) provide Kellingley with much needed electrical energy monitoring. The colliery opened in the 1960's with standard analogue energy meters, which provided no power data and had to be read manually. These were replaced with digital power meters, Appendix 19 shows the original energy meter and a new digital power meter. 22 digital power meters were installed at Kellingley for this project, monitoring all the surface equipment and the underground supply, also being integrated into Kellingley's existing SCADA system with MODBUS communications. The MODBUS communication wiring was separated into eight channels to increase the bandwidth and sampling rate of the data acquisition. Appendix 19 also shows the wiring diagram of the MODBUS system. The eight channels required the use of two MODBUS gateways to interface with the server and the data logging software. The system also consisted of two Dell Power Edge servers, one in the substation and one at CSM. The system also uses various software packages, these are: a database server, OPC Top Server, OPC logger, SSH Tunnelling and the colliery SCADA software, Appendix 20 shows a schematic of the software system. An OPC logger was used to log all the data produced by the power meters into a database. The average sampling rate was two times a second for every meter. During the setting up of the system there were problems with the reliability of the data and it was found that the meter wiring was causing the problems. A test meter was built to aid in the troubleshooting (and initial setup of the software). A procedure was created to ensure all the meters were wired correctly. Once the installation was complete, the data from the meters was compared with the utility electric bill. This comparison proved the system to be over a period of a month accurate to less than 1%. Two different months were used and this accuracy was proven to be consistent. From the data a demand load profile database was compiled of the various loads in the mine (Table 5)

Table 5: Demand load profile example data

Machine	Max Demand (kW)	Average Demand(kW)	Constraint
Shearer	553	330	Production
AFC	4224	976	Production
Conveyor	317	142	Transport
Coal Hoist	2293	799	Transport
Surface Ventilation	509	495	Safety

During the project a questionnaire was distributed to the coal mine partners and partners that had coal mine connections, to gain knowledge of the European coal mines, especially the size of the mines and their electrical demand (Table 6).

Table 6: Mine data from questionnaire

Mine	Method	Depth(m)	No. of Faces	Tonnes/Hr.
CMV (Solvenia)	Longwall	530	2	2200
Ziemowit (Poland)	Longwall	701	3	1200
León 03 (Spain)	Face Workers	149	7	20

Conclusions: The agreement between UK Coal and CSM was key to the completion of this Task, as this gave it the research edge needed over the more basic data originally planned. The advanced installation at Kellingley was complex and can be considered a complete success. The company now has a comprehensive energy management system at Kellingley Colliery and CSM have the data required to continue the research into Task 4.2.

Exploitation and Impact: The live production data from Kellingley is unique and very desirable from a research perspective. Even mines themselves would not have data of such detail and it will also provide a base for further research into mine power management, including a new RFCS project entitled M SMART GRID.

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Task 4.2 Intelligent Demand Reduction & Responsive Technology for Underground Coal Operations

To examine smart grid initiatives across the EU; to appraise demand reduction & management strategies; to assess uncertainty in the energy industries and potential impact; to evaluate relevant technology systems; to evaluate the potential for carbon footprint reduction and to examine the cost / benefit.

Activities and Discussion: The main activity of this Task was the development of an intelligent demand reduction and responsive control system for underground coal operations. This was achieved systematically, firstly with the definition of the demand reduction pyramid (DRP). The DRP gives the procedure in its logical hierarchy form. Each level is proportional to how important each stage is on the overall success of the final demand reduction strategy.

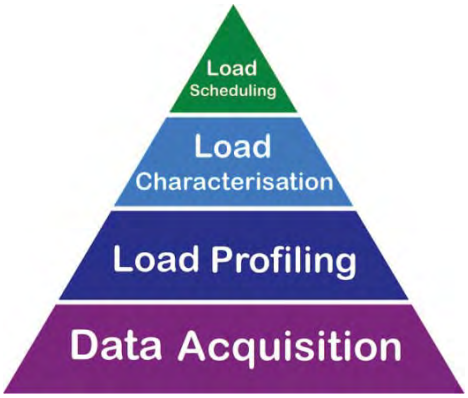

Figure 9: The Demand Reduction Pyramid

Considering the DRP, 'Data Acquisition' is the most import process of the system and has been completely covered by Task 4.1. 'Load profiling' has been traditionally used by utility companies to forecast, schedule, and define tariffs for various customer groups. The approach to a load profiling process for demand reduction is different as the load profiles are for individual loads within the mining operations and are reduced in detail to only the required demand information with a process known as model skeletonisation. This process defines all the loads with their demand requirements. This simplification of detail permits quick computation and application of multiple loads. The underground power data from Kellingley is not for the individual loads, instead the underground load data is aggregated together. This poses a problem when defining the individual load profiles of the underground loads. To aid in the load profiling, research was undertaken into the disaggregation of the aggregated data. A simulation was built in MATLAB and Simulink to generate profile data, wherein the data can be considered and solved as simultaneous linear

equations; this approach was applied to the live data. However, the results with the live data showed the approach required further research. For 'Load Characterisation', once the load profiles have been defined, the loads are then characterised, the steps for this process are as follows:

Step 1 Main Categories.
Step 2 Allocate Load Group.
Step 3 Define Load Characteristics.

Step 1: The first step is to define the main categories for which the loads fit into. Three main categories have been defined; these are Constant Loads, Free Loads and Cyclic Production Loads:

- Constant loads are loads that exist in the system, although their operation cannot be changed or removed. The load does not need to be running constantly. However, it must have a fixed operation cycle that cannot be changed, for example due to safety or production/operational constraints. Main ventilation fans are a good example of a constant or fixed load.
- Free loads are loads that can operate without safety or production/operational constraint. The load can have constraints on how often it must operate; however, these time constraints can be managed so the load can operate at different times. Scheduling of free loads mainly focuses on the 'valley filling' (smoothing) and price advantages.
- Cyclic production loads are loads that the mine operation and production rely on and cannot continue without. These loads may be in groups or individual; however, these most likely have constraints in that they need other loads or group of loads to operate before, during or after their operation.

Appendix 21 shows the flow diagram for main category allocation

Step 2: Allocate Load Group, whereby the production of an underground coal mine is more or less set and follows a constant routine of extraction, underground transportation, and finally transportation to surface, either by hoisting or conveyor, depending on the depth of the mine. Appendix 21 also shows the flow diagram for allocating load groups.

Step 3: Define Load Characteristics, for which the loads have now been categorised and grouped. The last step is to define the characteristics of these grouped loads. The load characteristics define the way that the load exists with the network and its group. The following list outlines possible load characteristics.

Load Running Time.
Load Running Interval.
Regular or Random Load.
Any Linked Loads or Patterns.
Overlap Running of Linked Loads.

For 'Load Characterisation', as we have seen, the three steps of load classification involve manual input to complete the process. A fully automated system is the ultimate goal. If a system could take a data set and produce the optimised load schedule with minimal manual input this would have a enormous impact on all industrial processes, not only mining. The Load Classifications MATRIX (LC MATRIX) was developed to increase the potential for automating the entire network optimisation system from data analysis to scheduling.

The last stage of the network optimisation is 'Load Scheduling'. This involves ordering the load start-up and running to give the maximum production with minimum electrical demand. The LC Matrix was further developed for the off-line scheduling. Loads are allocated a type using their characteristics, these types are as follows:

A Loads:	Unavoidable and Dependency Ranking order of avoidability. No adjustment to profile position is made.
B Loads:	Unavoidable and Independent. Ranking order of avoidability. Whole operational profile is shifted to minimum demand.
C Loads:	Avoidable and Dependency. Ranking order of dependency. Each operational block is shifted to minimum demand.
D Loads:	Avoidable and Independent. No ranking. Load is used for valley filling.

Appendix 21 shows how the LC MATRIX Scheduling operates. The simulation used in the load profiling research was also used to develop and test the LC Matrix. The simulation results gave excellent results. The process was then applied to the real-time data. The LC MATRIX optimisation of the face and hoist machinery reduced the maximum demand by 2850 kW. The demand reduction is nearly 3 MW, being is a significant impact and a constant demand reduction. A central control system with the optimised schedule to operate the load will always see this reduction. During peak periods the mine has a peak of 18 MW, including the un-optimised face and hoist profiles. However, if the optimised peak face and hoist profile is operated, this would be reduced to 15 MW and save the mine over £2000 per month in availability charges alone.

Conclusions: The demand reduction process outlined here has been proven, not only in simulation, but also using the real-time data. The demand reduction system is working, although it is not complete and would benefit from further research. The system provides a foundation for automation of demand reduction.

Exploitation and Impact: The system offers an impact on the real-time processes of underground mining operations and made distinct cost and demand savings. Further development will improve the system and allow underground mining to move into the future with integration into smart grid technology, such as in the RFCS project M SMART GRID

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Task 4.3 Ventilation Improvement

To design the ventilation network and to develop required algorithms; to automate the system; to introduce robotic control devices; to provide measures to reduce ventilation circuit leakage; to provide warning systems and requirements; to define escape ventilation characteristics; to consider cost effectiveness.

Activities and Discussion: The Velenje lignite mining method is a unique system exploiting a very wide seam. The mine has around 50 km of roadways and inter-connections, a lot of underground features, relatively high air speeds and airflow and high levels of dust and gases, including carbon dioxide, that render the ventilation complex.

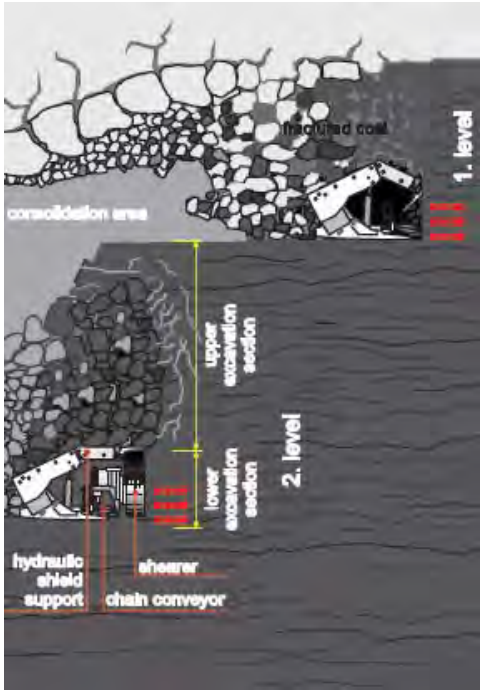

Figure 10: The Velenje mining method

The underlying aim was to achieve optimal efficiency of the fans operation while assuring the proper ventilation of the mine (gas concentration, regulations etc.). Different parts of the Velenje mine are ventilated with two main fans drawing different air from five air entry sites (shafts, declines). Several aspects were under consideration:

- Gas occurrence from different mine areas[2]
- Airway networks covered by the two main fans and their interdependence
- Machinery aspects (wear, maintenance)
- Electrical aspects (options for automated control)
- Safety aspects of ventilation control

In 2010 CMV implemented SCADA - central energy surveillance system (CESS) (energy consumption and cost monitoring). An algorithm was developed and the CESS upgraded for easier ventilation consumption monitoring and obviating the need for manual summation of the fans. Changes of the fans operating points were monitored for consumption and mine behaviour, including reduction in ventilation during holidays and weekends. Additionally a SCADA surveillance system (VTIS) (gases and technical parameters monitoring) was used. The first step towards optimisation was to assure accurate data about real-time airflow for which new air speed sensors were installed at entering and exiting airways, mine faces and some other locations. Initial anomalies in the data obtained were investigated by comparing different sources, times and conditions. This enabled effective calibration and positioning of sensors according to airway type (obstacles, junctions, curves, airspeed, etc.). Frequency of cleaning and monitoring in the dusty environment was formalised. Measurements were taken at the faces, analysing and comparing gas intensity. Verification was done with a multi-functional measuring instrument and data logger and became a significant aspect of the project (temperature, humidity, air speed, pressure, etc.):

- Control and calibration of connectors in the CMV laboratory and at the manufacturer
- Control measurement of static air speed sensors
- Measurements of air speed at the mine faces and their exiting airways (several measurement points)
- Control measurements of the main fans at Šoštanj and Pesje shafts
- Measurements of short circuits / resistance using the barometric method

At the main fans, four Pitot-Prandtl tubes (sonds) simultaneously measure air speed. Early issues (faults with connectors and the device algorithm) were taken up with the supplier and manufacturer. New measurements holes were drilled at the fans to reduce the influence of turbulence.

A measurement protocol for short circuits and losses through doors / barriers was implemented. Resistance data was used for updating the Zračenje programme.

When the main Pesje fan was turned off for a rescue exercise, ventilation behaviour was measured by eight teams and simulations and sensors tested.

The ageing CMV ventilation software programme Zračenje was fundamentally upgraded (software, operating speed, formulae, input data, working interface). Comparison was made with the VenPri programme from project partner AITEMIN. Initial programming work was carried out to develop AutoCAD tools to work with the existing Zracenje linear network map. Further work went into a new concept of a New Linear Airway Network Map (NLANM). Attention was given to creating more representative resistance factors. The existing mine ventilation map can now be imported to a new AutoCAD map. Airways characteristics can now easily be set or corrected. Definition of objects and calculation algorithm were enhanced. An Excel exchange table for interactive work and AutoCAD objects were defined. Databases were cleansed and a basic concept of a mask for setting and changing airway parameters was implemented. Control calculations for monorail roadways and other views which may change were included. Object types in the roadway were verified comparing old equipment with ALMEMO and additional very precise VAISALA micromanometer and pressure sensor, showing only a very small difference.

Accurate measurements at Šoštanj main fan at non-production conditions were performed (pressures, airflow, humidity, temperature, power) to set corrective factors in VTIS, to assess the condition of the fan related to existing characteristic curves and to find out the impact of fan plate changes on mine faces. Parallel to CMV measurements, CSM measurements were performed by Poirson technique of hydraulic efficiency. Additionally airflow data from mine faces, exiting airways and entering airways was analysed. This is described in Task 6.1.

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[2] The mine is divided into three areas; also the Velenje longwall mining method can vary (excavating the lower coal layer only or both upper and lower coal layers).

A proposal to be studied was if the majority of ventilation can be covered by the big Šoštanj fan, potentially using less mine roadways, different fan operating points and a predicted new coal transport shaft. With the ventilation software Zracenje, three different mine situations were calculated: (i) using existing fans (Pesje and Šoštanj); (ii) substituting Pesje fan with Hrastovec fan (smaller); (iii) removal of some airways.

Conclusions: Knowledge of improved calibration and positioning of sensors was gained and a cleaning and monitoring schedule was implemented. Airflow behaviour with the Velenje mining method is generally disorganized, even chaotic, and tends to form eddies, affecting resistance, de-gasification and de-dusting. A modification of the Atkinson equation should be used for mine faces. A lot was learned from the rescue exercise, leading to checking of air speed sensors in case of reversed airflow, gas sensors with wider measurement scales, additional sensors and remote opening doors for fan stopping in emergency situations to guide airflow past the mine faces. For the ventilation software, improvements were made with operating principles and database linkages, while upgrades created faster and easier use and object type entries were enhanced. For cross-sections the conclusion was that ATEX measurement lasers are too expensive and with special safety conditions non-ATEX lasers may be used. Software was corrected with new measured data and using more precise equations which are very complex, especially for the resistance of objects in airways, junctions resistance and resistances at mine faces. Improvements were made to assure accurate, effective data in the VTIS system. Fan operating efficiency is now better understood for different fan plates angles plus changes at mine faces, considering the whole mine as a system. One specific finding was that changing fan plates at Šoštanj fan increases airflow on the first mine face but slightly decreases it on the other face, because both faces are using same shaft air intake. Insight was gained as to how much additional airflow at mine faces costs. From the results it was seen that the fan plates are not in the best condition and should be replaced with new ones.

From the simulations, it was seen that removing Pesje fan and installing a smaller fan at a different location would not be economical, despite a better operating point for Šoštanj fan. This is because Šoštanj fan would cover the majority of mine ventilation which includes very distant airways and hence increased cumulative resistance. However, the mine could be separated into two independent parts wherein some risks and issues would need to be considered (rescue paths, mine connections, etc.).

Exploitation and Impact: The Velenje mine is benefiting from these improvements, including greater understanding of fan costs and quicker and more effective management of the mine ventilation as a total system. This learning might potentially be carried forward to other, similar underground lignite sites where CMV offers consultancy on the 'Velenje mining method'.

Task 4.4 Pumping Efficiency Improvement using Intelligent Control

To identify / develop and appraise technology to measure efficiency and emissions of turbo-machinery in real time; to integrate temperature, pressure and electrical power technology into the mine network demand reduction strategy; to conduct trials on a test rig; to integrate real time acquisition and control of turbo-machinery into the demand reduction strategy.

Activities and Discussion: Pumps are designed to operate at a 'head' that corresponds to the best efficiency point (BEP) of the pump, a function of the system and other pumps operating. Even at installation, many industrial pumps are not near the BEP. This research has been based on the Poirson technique of measuring small changes in pressure, temperature and electrical power to obtain the hydraulic efficiency. The hydraulic (pump) efficiency is determined by assessing the actual increase in enthalpy across a pump and comparing this to the theoretical minimum enthalpy required to generate the same pressure at the given flow rate. To this end, a 'mine simulator' has been constructed at the Camborne School of Mines Test Mine. Appendix 22 shows the housing with the white header tank and some of the internally installed equipment.

The starter panel houses the circuit leakage and over-current protection, machine starting circuits and power metering. The provides the degree of electrical circuit safety required by law, but also the 3-phase power analysis of the motors / drive prime movers of each process. The voltage is connected directly from each phase and the current is inductively measured by a current transformer on each phase with a ratio of 50:1 with 2 turns of the primary. Communication is via RS485 MODBUS with each meter assigned an ID and wired on a common bus. The motors are started locally via a contactor in the panel. The test control centre houses the control PC, touch panel screen, signal conditioning for transducers, regulated power supplies, digital communications

and router. The platform is an IP55 rated 19” rack with keyboard tray and lower enclosure for the electronics. Due to the potentially wet and arduous nature of the testing regime a touch panel screen has been installed to allow test progression and user driven editing to be safely performed. The PC is a 2.4GHz, 2GB RAM, 500GB hard disc, 19” rack mounted construction, selected to store the test data generated. Due to the steady state nature of all proposed tests a large memory is not required. Primary measurements were acquired to support, calibrate and, most importantly, validate prototype measurement systems developed on the test rig, including reference sensors for:

- Volumetric fluid flow rate (electromagnetic flow meter)
- Static fluid pressure (before and after the pump and before and after the throttling calorimeter)
- Electrical power (3 phase power meters with current transformers)
- Barometric pressure (to support the determination of fluid properties)
- Atmospheric temperature (to support the determination of fluid properties)
- Relative humidity (to support the determination of fluid properties)

Reference measurements of flow rate, pressure and power have been installed to be able to check computations of pump efficiency. ½” BSP tappings are present either side of the pump to enable the inserting and exposure of sensors. A computer controlled, motorised valve has been placed in series to enable defined cycles to be operated for precision positional control, using a PID loop, for throttling calorimetry experiments. The pump under test is a 5kW Lowara centrifugal pump with a low specific speed for high head experiments. The suction and discharge lead to, and from, the same 7000 litre header tank on the roof. The volume has been designed to limit heating up problems (through losses) for longer duration tests.

The absolute pressure and temperature are only required to determine the fluid properties, but changes across the turbo-machine are required to a much higher accuracy. Based upon the qualities of high overpressure rating for transients, high electrical output simplifying signal conditioning and good thermal properties, Silicone on Sapphire sensing technology was identified for the mining industry. The differential temperature across the machine is required to a challenging accuracy of 1/1000th °C and potential technologies identified were the Platinum Resistance Thermometer (PRT) and the Thermistor. After a calibration process, test sensors were mounted in the simulator pumping system and operated for 100 hours in the same stretch of pipe. This identified the thermistor, as spectral analysis showed a distinct offset in measurement with the PRT, implying physical change over time.

Conclusions: The proof of the system was in the successful computation of hydraulic efficiency and flow rate by the Poirson method with low cost hardware design. This has been verified by the reference instruments on the mine simulator using the conventional (pressure – flow rate) method, more accurate in the very controlled simulator conditions than in typical working situations. A key comparator, the flow rate, is shown in

Table 7 as the outcome of some of the tests.

The following conclusions were drawn from this set of tests:

- There is a degree of variability in the results, both Poirson and Conventional, and so the process would benefit from multiple tests and statistical treatment
- Despite sensitive pre-calibration the operating difference between the two thermometers was 7.47mK, which had to be removed by a zeroing process that will be included in future procedure
- As the flow rate was reduced the error became greater which could be due to either moving away from the full scale deflection of the transducers or moving to position on the motor load – efficiency curve which is less well defined than at rated conditions

After an heuristic correction in motor efficiency the Poirson method agreed well with the flow meter highlighting the importance of motor efficiency estimation.

Table 7: Comparison of pump flow rate; conventional vs Poirson

REFERENCE Volumetric Flow Rate (l/s)	POIRSON #1 Volumetric Flow Rate (l/s)	POIRSON #1 Error (%)	POIRSON #2 Volumetric Flow Rate (l/s)	POIRSON #2 Error (%)
15.3	14.9	-2.6%	15.3	0.0%
14.0	14.0	0.0%	14.1	0.7%
12.5	12.4	-0.8%	12.6	0.8%
11.0	10.8	-1.8%	11.0	0.0%
9.6	9.5	-1.0%	9.5	-1.0%
8.0	8.0	0.0%	7.9	-1.3%
6.4	6.6	3.1%	6.2	-3.1%
5.2	5.1	-1.9%	5.1	-1.9%
3.5	3.6	2.9%	3.4	-2.9%
2.2	2.1	-4.5%	2.1	-4.5%

The comparative hydraulic efficiency results are shown in graphical form in Appendix 23 and may be compared with field trials in Task 6.1. The overall outcome showed that, with rigorous testing, the Poirson method using low cost sensors showed potential for efficiency measurement of mine pumping systems, providing a means of generating data essential to energy monitoring and improvement programmes. The sensor system was further refined before the field trials briefly described under Task 6.1.

Exploitation and Impact: This system, which exploits low-cost modern sensor technology to apply a theory first proposed a century ago, is now available for efficiency measurement of industrial pumps in the harsh environment of underground mines and awaits take-up by the wider industry. The work has been presented at mining conferences in Cornwall, UK, and Canada and awaits publication in a mining journal.

Task 4.5 Surface Facilities Energy Efficiency

To assess the potential for carbon dioxide emission reduction at a modern colliery site by consideration of construction standards, energy systems, fuel switching, renewable energy and decreasing demand through behaviour or technology.

Activities and Discussion: A review has been carried out of the various measures that can be adopted to improve the energy efficiency of the surface facilities at European mines, most of which had been built when energy use was not the major issue it is today. Consideration has been given to various levels of refurbishment of surface buildings, some of which can be carried out at moderate cost, capable of making substantial reductions in energy consumption. In addition, the role of various intelligent building control systems has been investigated.

A minor refurbishment could involve the following measures:

- Changing space layout to enhance daylight, ventilation and zone controls.
- Improving lighting arrangements.
- Improving window performance by adding blinds.
- Using lighter coloured interior surfaces and furnishings to enhance lighting.
- Seek passive energy efficiency measures, such as natural ventilation.
- Opportunities for better personal control of the local environment.

- Zoned areas where there are high heat gains or special environmental requirements.
- Limiting heat loss across roof, wall, floor, windows and doors, by suitable insulation.

A major refurbishment might include the following more substantial measures:

- Adding additional insulation, including energy efficient glazing using low emissivity glass, with argon or vacuum filling.
- Adding atria and sun spaces to increase natural ventilation and daylight.
- Increasing the use of passive strategies in air conditioned buildings.
- Removing (fully or partially) air conditioning through changes to fabric, lighting and controls, e.g. zone controls.
- Specifying an efficient and fully insulated hot water system.
- Introducing passive measures to reduce external heat gains while maximising daylight, e.g. replacing windows, providing shading, introducing atria and roof-lights.
- Changes to the ventilation strategy to minimise the use of mechanical ventilation.
- Installing energy efficient plant, such as condensing boilers and combined heat and power (CHP).
- Installing energy efficient lighting and lighting control systems.
- Improving building services monitoring and controls.

Smart meters for electricity and possibly gas can provide a service greater than the simple logging of usage for billing, especially where sub-meters are installed. Real-time detailed information on consumption can enable active management of loads, tariff switching and, in some circumstances, energy export. This can be taken further to a computerised Building Energy Management System (BEMS), which may be linked to a SCADA. Voltage optimisation, especially for fixed-speed devices, can save up to 25% on electricity. Fuel-switching, including to on-site renewable sources, provides another option. It is suggested that the thermal energy in upcast air or from heavy-duty electrical equipment might be exploited, although effective heat exchangers are not cheap to install. Upgrading of old machinery such as compressors or the use of efficiency monitoring (Task 4.4) can make significant savings, noting the cube relationship between speed and energy in many cases. Variable speed drives on conveyors are a well-understood efficiency mechanism. The matching of high loads such as winders to low tariff periods may be possible (Task 4.2).

Human behaviour, from energy targets for managers to posters and meetings campaigns, provides a major enabler to energy saving. Even simple acts such as switching off office computers and printers can make noticeable savings over time.

Conclusions: The increase in energy costs should stimulate a search for new opportunities, but also a review of earlier energy savings projects which were previously considered to be of marginal viability. Justification criteria should be reviewed against the continuing rise in energy cost on a regular basis. Appraisal of low carbon systems should be carried out on whole-life cost analysis, an approach increasingly used in building design rather than simply looking at initial capital costs. Consideration should be given to devices that supplement the data from site fiscal metering. A multi-disciplinary energy efficiency design team should be appointed to use targets and life cycle costing. Appointing an energy champion at a senior level is an excellent start to sustaining and demonstrating commitment and achieving an energy saving culture within an organisation. However, the challenge in changing attitudes and policies in an industry in decline should not be under-estimated.

Exploitation and Impact: This work provides one of the elements from this project that feeds into a further RFCS project investigating smart grid concepts for coal mine sites.

Work Package 5 Energy Storage

Objectives of WP5: *To provide a comparative review of energy storage options; to provide a critical evaluation of compressed air scheme options and system optimisation; to provide guidelines for, and subsequently identify, potential sites for Compressed Air Energy Storage (CAES); to evaluate structural risks for specific sites; to examine heat storage options for underground mines.*

Comparison of initially planned activities and work accomplished: This Work Package has been delivered in full at a theoretical level. Where detailed areas for investigation were specified, these have been reviewed and key aspects given attention. The scale of the concept means that an actual application was never within the scope of this project, and, although a suitable site has in fact been identified and subjected to analysis, it has not become available for this use. As heat storage for CAES has been covered in Task 5.2, Task 5.5 was in the event used to examine heat storage for thermal reactor design.

Task 5.1 Comparative Review of Energy Storage Options

To provide a comparative review of energy storage options to assess the potential application of Compressed Air Energy Storage (CAES).

Activities and Discussion: A comparative review has been carried out for various energy storage options which are available for immediate deployment as well as technologies which are currently under development. The review has covered the overall scenarios of energy systems; the importance of electricity and other forms of energy; the strategies for integrating renewable energies; and the added system costs due to the increasing penetration of renewable energies into the main grids, in order to help to understand the demand for energy storage. The time dimensions have been reviewed and the control strategies of energy storage have been identified. Energy storage of all types addresses three primary market-related roles:

Power Quality - Where stored energy is utilised for seconds or less, as required, to assure continuity of power quality/stability.

Bridging Power - Stored energy in these applications is used for seconds to minutes to assure continuity of service when switching from one source of energy generation to another.

Energy Management - Here the storage media offers an extended duration and allows the timing of generation and consumption of electric energy to be substantially decoupled.

There are various options or methods available to store energy from high generation periods to release at other times (Dincer I and Rosen M A, 2002). These include:

(1) *Mechanical options*:

— *Pumped hydro* where the pumps pump the water upward through pipe using off-peak energy and the stored water moves downward during peak time to rotate turbines/motors to generate electricity. Berry estimates the cost of pumped storage at approximately £3,000/MWh (Berry G, 2004).

— *Compressed-air* where air is compressed and stored in large underground reservoirs such as naturally occurring caverns, salt domes, abandoned mine-shafts, depleted gas and oil fields or artificial caverns (Dincer i and Rosen M A, 2002). A commercial underground CAES plant of 1080 MWh capacity costs around $1000/kW in terms of cost per unit power and $125/kWh in terms of cost per unit storage capacity. An increase in the capacity of similar plant to 2700 MWh increases the cost per unit power to around $1250/kW but the overall cost per unit storage capacity decreases to $60/kWh (EPRI, 2010).

— *Flywheel* where a wheel of relatively large mass rotates and stores rotational kinetic energy. For large-scale energy storage systems, flywheel can demonstrate 80-90% efficiency (Dincer i and Rosen M A, 2002). Low speed flywheels cost between £150/kWh and £200/kWh and high speed flywheels with higher energy densities cost as much as £17,000/kWh, being in the early stages of development (Gonzalez A et al., 2004).

(2) *Chemical options*:

— *Electrochemical batteries* which store energy in systems composed of one or more chemical compounds. Commercial sodium-sulphur batteries of 300 MWh capacity with 75% efficiency and 4500 cycles life-time cost around $3100-3300/kW in terms of cost per unit power and $520-550/kWh in terms of cost per unit storage capacity. Lead-acid batteries of 250 MWh capacity with 85-90% efficiency and 4500 cycles life-time cost around $4600-4900/kW that means $920-980/kWh (EPRI, 2010).

— *Organic molecular storage* uses the photosynthesis process, in which an endergonic photochemical reaction followed by exergonic regeneration can convert radiative energy into storable chemical energy (Dincer i and Rosen M A, 2002).

(3) *Biological option* which uses bioconversion process to store energy in the form of chemical.

(4) *Magnetic options* use superconducting materials to store energy in a magnetic field. The cost has been stated at approximately £200/kW (Gonzalez A et al., 2004).

(5) *Thermal energy storage options*:
— Sensible heat storage where energy is stored by elevating or lowering the temperature of a substance.
— Latent heat storage which uses phase change materials that release and absorb energy in the form of heat when the state of the material changes.

The operation of energy storage is controlled by certain principles known as control laws for storing and withdrawing energy from storage. Both capacity measures and performance measures are used to evaluate energy storage performance. In an adiabatic compressed air energy storage (A-CAES) plant, the thermal energy in the compressed air is stored in a specially designed thermal energy store, then re-used to heat up the air entering the expander so that no fuel needs to be burned. Improved efficiencies are possible using A-CAES, though an A-CAES plant has a higher initial cost than a conventional CAES plant because of the need for a thermal energy store and heat exchanger.

Conclusions: From the study of the alternatives above, CAES, especially A-CAES, occupies an important position in that it offers a storage technology of strategic scale, both in terms of the operating duration and power level of the energy delivered, with only pumped hydro standing out as a mature technology with very good operational performance. This provides justification for the further work below to explore the application to coal mine shafts.

Exploitation and Impact: This work provides an underpinning rationale for the subsequent research.

Task 5.2 Compressed Air Scheme Options & System Optimisation

To evaluate CAES options and technology requirements for system optimisation, taking into account a specified range of options, limitations and other factors.

Activities and Discussion: Energy storage of all types addresses three primary market-related roles; (i) power quality and reliability down to very short periods, (ii) bridging power (continuity of service when switching), and (iii) energy management to decouple the timing of generation and consumption. The technical and commercial drivers behind the optimal reuse and deployment of abandoned mine-shaft infrastructure for Compressed Air Energy Storage (CAES) systems have been identified, showing that CAES is probably the lowest-cost utility-scale bulk storage option for dealing with the intermittency of non-hydro renewable energy sources. CAES facilities typically demonstrate a ramp rate which is three times faster than combined cycle gas turbine facilities. The large-scale and fast reaction capabilities of CAES permit such systems to make frequent start-ups and shutdowns and the concept is clearly a leading contender in the energy storage hierarchy.

As mentioned above, adiabatic CAES uses a thermal energy store, then heating the air entering the expander, so that no fuel needs to be burned although initial investment cost is higher. As fuel and carbon emissions costs increase, A-CAES will become more attractive. In the diabatic CAES (D-CAES) system, the extra heat gained from the compression process is released to the environment using an intercooler prior to store. When the cold compressed-air is released from the storage to generate electricity, additional gas firing is required to heat up the compressed-air so that it reaches the turbine operating temperature. Using the stored compressed-air along with additional gas firing helps saving about two-thirds of otherwise needed gas as well as helps achieving low part-load losses and high flexibility (Gatzen, 2008). Currently, there are two D-CAES systems operating in the world. One of the main findings of this work for the LOWCARB project is that it is very attractive to have a CAES plant with the option either to burn fuel or simply to use the existing (relatively low grade) stored heat. At times where there is likely to be a sustained strong demand (i.e. high price) for electricity for an extended period, burning fuel can more than double the amount of electrical output which would otherwise have been achievable with a given quantity of high pressure air.

Compressed-air can be stored in pressurized containers under various processes, primarily isochoric or isobaric. The isochoric is a thermodynamic process in which the contents of a closed system formed with sealed and inelastic container maintain constant volume with the addition or the removal of heat. An isobaric process is a thermodynamic process in which a pure substance within a closed system formed with a cylinder and a leakproof piston of constant weight remains under constant pressure with the addition or the removal of heat. The ratio of stored energies for the same physical volume and the same maximum cavern pressure typically favours isobaric over isochoric storage. However, in view of the significantly increased engineering input and related cost of creating an appropriate shuttle pond at this site to receive the displaced water, a decision was taken in the context of the theoretical case study to treat the shaft for isochoric air storage. A surface-level tank such as would be required on this flat surface would be very expensive, approaching cost levels of £500,000/MWh, in the same order as electrochemical energy storage options. The proposed isochoric system will provide an energy return to the grid in the order of sixty per cent efficiency against input.

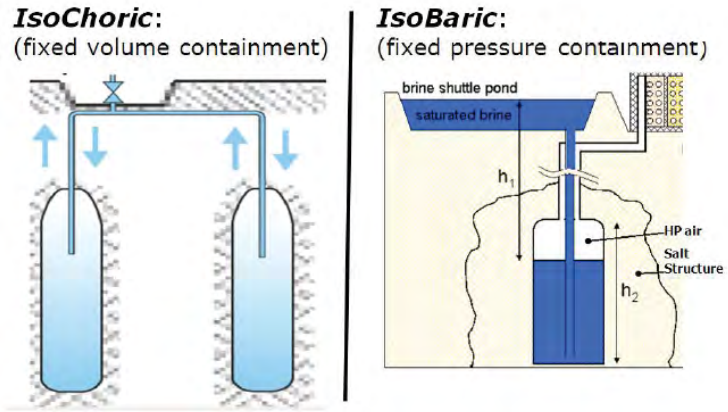

Figure 11: Isochoric and isobaric underground compressed air storage

The aim is to perform compression in multiple stages as the total amount of work put in to achieve a given quantity of cool high-pressure air is reduced if the air is compressed in small ratios and then cooled after each compression stage. Similarly, the amount of work extracted from a given quantity of cool high-pressure air is increased if the air is pre-heated before each expansion stage up to a maximum possible temperature. A minimum amount of preheat before an expansion stage is required to ensure that the air does not leave the expander stage at temperatures below freezing. This would cause problems for the machinery – mainly associated with the build-up of ice in the low-pressure end. High pressure air is never stored hot. The main reason is that it would lose its heat quickly anyway but a secondary reason is that even if the heat could be retained, the volume of storage required for the same mass of air and the same pressure is directly proportional to the temperature so there is better 'value' from storing cool air. Appendix 24 shows the approach to turnaround efficiency and also diagrams of a system using sensible heat storage and the two likely heat store options of a solid storage medium within the pressurised circuit or an exchange to a separated medium. Following review of water, molten salts, heat transfer fluids and solids such as rocks or concretes as storage media, this has led to the concept of an underground thermocline store using available broken rock as the storage medium for the mine-based CAES, fed from a heat exchanger carrying heat from the pressurising air in ambient pressure air to the rock. A rock-filled sensible thermal storage can store heat with temperature ranges between 100°C and 1000°C (Hasnain, 1998). Another potential use of available materials would be the application of thermal output from one of the methane oxidation technologies elsewhere in the LOWCARB project for top-up heat where required. Design work for the heat exchanger feeding the store has suggested the steelwork and labour for this to be a significant capital cost for the project. The work included a MATLAB-based calculation suite. Calculations for one specific case indicate costs of ~£120K/MW for the heat exchanger. While significant, it may not be prohibitive given that open-cycle gas-fired generation plant typically exhibits costs of £500K/MW of output power.

Necessary heavy equipment such as compressors, expanders, motor/generator, recuperator has been reviewed. Appendix 25 shows some of the items.

Conclusions: Using operating data from the McIntosh CAES plant in the USA applied to UK electricity price fluctuations 2005-2009, with a turnaround efficiency of 70%, we find that the

maximum possible yearly profit varies between £1.64/(kWh.year) (in 2009) and £3.69/(kWh.year) (in 2008), with the average being £2.16/(kWh.year). It should be noted that the expenses simply consist of spending on input electricity, and operating expenditure is not included here. Although initial capital expenditure is clearly high, ongoing earnings should also be considerable and, depending upon future energy price predictions, profitability is possible. Where electricity prices are announced in advance, a predictive strategy can be used to optimise the function, e.g. using an algorithm to control the input and output power of the plant at each time interval in the forthcoming 24 hours that will maximise profits.

Exploitation and Impact: This work forms the basis to provide the actual CAES plant to work with the compressed air store under investigation in the following Tasks.

Task 5.3 Site Selection, Sealing & Preparation of Storage Volume

To investigate a specified range of factors to provide a critical review and to identify the key factors that need to be considered / evaluated for the development of a CAES facility.

Activities and Discussion: The comprehensive list of aspects proposed for the project was reviewed and individual items were all addressed, with some of the most significant factors such as the review of mines and shafts, the suitability of cappings, shaft stability, lining stress, modelling of pressure changes, geology and sealing technologies becoming the major focus of the research. The other items initially listed in the Technical Annex such as insertion of boreholes and shafts into aquifers and hydraulic connectivity, were found to be of lesser or insignificant importance if a resilient, isolated pressure chamber could be proven. Operational items such as pre-mining abandonment measures and maintenance were addressed during the case-study (using Maltby No. 3 shaft - see below). A series of numerical models was run to investigate the effect of geotechnical and construction factors on the application of a circular cross-section mine shaft as a CAES facility. Appendix 26 shows the factors that were considered within the sensitivity study and a table giving all variations modelled for each factor. It was assumed in these preliminary models that the ground horizontal stresses are equal in different directions and the shaft was supported by a concrete liner with a uniform thickness. Therefore, a simple axisymmetric FLAC^{2D} model was employed in this research. As this research focused on the behaviour of the shaft liner and shaft caps under gas pressure, the surrounding soil and rock field was simplified to a single type of rock, in this case siltstone, rather than a complex mix of different types of rock strata. The preliminary numerical modelling provided the following outcomes:

1. The key to the success of the shaft CAES facility is the integrity of the concrete caps placed in the shaft. The lower cap is the most vulnerable area (it suffers the largest displacements) when there is backfill material above the upper cap. However, the upper cap becomes the vulnerable area when there is no backfill material above it.
2. Not surprisingly, the higher the gas pressure, the larger the deformations of shaft liner and shaft cap. The range of deformation of the lower cap was from 1.42 cm under a gas pressure of 4 MPa to 3.24 cm under a gas pressure of 8 MPa. The results also illustrated that the higher the gas pressure, the more a plastic area occurred in the shaft caps, shaft liner and surrounding rock adjacent to the upper cap.
3. High strength concrete caps (with uniaxial compressive strength of 100 MPa) remained intact in all the modelling cases (whatever the shaft length, gas pressure, cap width) whereas normal strength concrete (with uniaxial compressive strength of 35 MPa) did not. Under a gas pressure of 4 MPa, the largest deformation of the shaft NSC lower cap was 2.27 cm while that of the shaft high strength concrete lower cap was 1.42 cm.
4. Also unsurprisingly, the thicker the shaft liner, the lower the deformations of shaft liner and shaft cap. When the gas pressure was 8 MPa, the largest deformation of the lower cap was 3.15 cm when the shaft liner was 0.4 m thick. This value decreased to 2.92 cm when the shaft liner increased to 0.8 m thick.
5. The modelling suggested that cap width had a very limited influence on the deformations of shaft liner and shaft caps.
6. The longer the shaft length for gas storage, the smaller the deformations of shaft liner and shaft caps when there was backfill material above the upper cap, but this is very small.

7. The distance between the shaft upper cap and ground surface had a significant effect on that of the shaft upper cap. This distance must exceed 5 m. A distance of 10 m was found to be adequate in the case of deformations of the shaft upper cap and the backfill material above it.

Following on from this initial work, further numerical models were developed. These models, included modifications to the shaft cap width, the selection of a weaker rock material in which the shaft is modelled, the effect of the liner-rock interface (especially at the position close to the ground surface) on the behaviour of the shaft when there is no grout behind the liner and the effect of geological faulting in the ground intersected by the shaft. The following conclusions were drawn from this later work:

- The results supported the reasonable assumption that the thicker the shaft cap, the smaller its displacement under the same gas pressure.
- The weaker ground condition had very little influence on the shaft liner's displacement and limited influence on the lower cap's displacement.
- The vertical displacements of the shaft upper cap and ground surface increase significantly under internal gas pressure when the liner-rock interface is introduced into the model. This result illustrates that the bond between the liner and rock is very important for the stability of the shaft upper cap and the backfill material above it.
- The modelling implied that the presence of a geological fault had very limited effect on the deformations of the shaft caps. However, it does have an effect on displacements of the shaft lining and surrounding rock nearby to it. Under inner gas pressure, the shaft lining "expands" due to "outwards" displacement. On the other hand, the existence of the geological fault causes a weaker ground condition leading to higher loading on the shaft lining, especially at its joint with the lining, reducing this "expansion".

Relevant properties of concrete and related sealants were reviewed, with a view to achieving greater understanding of potential sealing for cyclical compressed air containment. A test rig was constructed for concrete research (see Appendix 27).

A review of site availability in current and abandoned coal mine shafts in the UK was conducted that identified Maltby No. 3 shaft at a currently closing mine as the best prospect.

Conclusions: It is considered that the formation of a Compressed Air Energy Storage facility in a disused coal mine shaft is feasible. It is likely that a modern, concrete lined shaft, recently abandoned (but before the mine entrance is treated and before the infrastructure of the mine had been removed) would be the most promising site to construct an experimental facility. Although detailed work would be necessary to estimate likely costs of such a facility, it is at least possible that much of the necessary shaft work would be carried out as part of the normal closure work, thus minimising the additional cost of the civil / mining component of the total project costs. The Maltby No 3 shaft will be the subject of customised analysis (Task 5.4).

Epoxy-based surface sealant could be applied to the concrete lining following high-pressure water-jetting, or a silica fume possibly used as an admix to cement. With the test rig it had been intended to raise the pressure to 80bar, but in the event air leakage occurred that was traced to the joint between the concrete sample and the sealant fixing it to the vessel. No leakage occurred through the actual concrete. A learning point from this was the need to make extra provision where concrete sections abut in the shaft lining or any other such inconsistency in the structure.

It is likely that costs could be contained within the order of magnitude of the costs of fabricating and installing a purpose made steel sphere or cylinder at ground level, which presumably would be benchmarked as the alternative approach.

Exploitation and Impact: Actual development is unlikely. The greatest problem, at least in the UK, is that the most suitable sites (Selby, Maltby) are already earmarked for alternative energy projects. Other possibilities (e.g. Daw Mill, Harworth) are unlikely to still be available for long, but there may be alternative options available elsewhere in the EU. It is noted in particular that coal mines in Germany, where there is a high penetration of renewable energy, are expected to close.

Task 5.4 Evaluation / Assessment of Site Structural Risks

To apply empirical and numerical modelling to case examples to demonstrate the impacts of identified factors as appropriate.

Activities and Discussion: Following review, the specific areas identified for detailed attention were the hazards posed by static electricity, possible migration of gases and an examination of the

permeability of the concrete lining. Other issues listed in the Technical Annex were largely dealt with by reference to the findings of Task 5.3 or considered minor or insignificant when dealing with a sealed, isolated pressurized container, which the CAES shaft would become during operations. There is a risk of static and hence a spark due to the flow of a gas at high pressure through a nozzle, in this case not be due to the air itself but would depend on the tribo-electric properties of any particles or droplets carried in the air, of the reservoir walls and any separate lining that may be used and of the pipework and nozzle through which the air is introduced into the reservoir. For the gas migration hazard, depending on the scenario and the time since abandonment, recommendations are made to assess the need for additional vents. Further numerical modelling based on the generic model in Task 5.3 but specific to the Maltby No. 3 shaft case study was conducted, together with a review of the best approach to undertaking the work that showed the need to plan it in closely with normal closure activity.

Conclusions: Preventative measures against static such as using conductive materials for the pipework and providing a low resistance path to earth could be put in place at low cost. Static mitigation measures associated with conditioning the incoming air stream are available. Ionising the incoming air, for example, would be expensive for the volumes involved and pose either a health concern due to the use of polonium 210 or could itself pose an explosion risk due to the use of high voltages. Physical filtering would pose a static risk in itself whereas electrostatic filtering involves the use of a high voltage. This leaves just two methods – humidifying the air and de-humidifying it – which are mutually exclusive and need further work to determine which, if either, of the approaches may prove effective.

When a new gas vent is deemed necessary, measures such as ensuring that it is widely separated from the CAES air inlet and in the opposite direction to the prevailing wind should be taken. In the event of a catastrophic failure of the lining, some means must be provided by which compressed air can be vented to the atmosphere. In all probability, this will take the form of solenoid-operated valves in the upper cap which are triggered if the rate of loss of pressure in the reservoir exceeds a threshold. Methane sensors within the CAES facility would be installed as well as monitoring the flow rate and methane concentration from the mine vents.

At the time of the research, the closure of the Maltby mine had been announced. The concrete lined No 3 shaft is 991 m deep with internal diameter of 8 m and was sunk in the early 1990's. Two-dimensional (2D) numerical modelling was designed and conducted based on strata and lining support information from Maltby. Appendix 28 shows the 2D model developed from the original plan for Maltby 3 shaft and some of the modelling output with variations in top plug depth and pressure tracking deformations in the ground surface, the lower plug and the shaft lining. It was concluded that:

1. The ground surface had a significant 0.11 m uplift under 8 MPa gas pressure when the top plug was located at a depth of 30 m. This significant uplift decreased to approximate 0.04 m when the top plug was moved to a depth of 80 m. When the gas pressure was reduced to 4 MPa, there was only 0.01 m uplift of the ground surface when the top plug was located at a depth of 30 m and zero uplift when the top plug depth was over 40 m.
2. For the same depth of the top plug, the higher the gas pressure, the larger the lateral displacement of the shaft liner. When the gas pressure was 8 MPa and the depth of the top plug over 50m, the depth of the top plug had little effect on the maximum lateral displacement of the shaft liner. With the same gas pressure, the maximum lateral displacement of the shaft liner was in excess of 0.15 m, regardless of the depth of the top plug. In practice, this displacement would indicate a failure of the shaft liner.
3. For the shaft liner, two areas were found to be the critical ones and needed more attention in a CAES system. These were where larger lateral displacements occurred close to the top plug and close to the depth of 470 m underground where the shaft liner's thickness changed suddenly from 1.2 m to 0.4 m.

The overall conclusion was that the concept would be viable if the above outcomes were to be respected and further customised. A modern, concrete lined shaft such as Maltby No 3, recently abandoned but before the mine entrance is treated and before the infrastructure of the mine had been removed, would be the most promising site to construct an experimental facility. Although detailed work would be necessary well in advance for project planning and to estimate the costs of such a facility, it is at least possible that much of the necessary shaft work would be carried out as part of the normal closure work, thus minimising the additional cost of the civil / mining component of the total project costs.

Exploitation and Impact: No insurmountable engineering hurdles were identified to the selection of a current concrete lined former coal mine shaft for conversion into a CAES facility. The chief problem is one of timing. Identifying the shaft closure date far enough in advance of closure to

ensure that the installation of the shaft caps and other equipment could be planned to synchronise with access being still available at the site is essential. Recent mine closures in the UK have not been planned events that were known about well in advance. In these circumstances mine owners and operators are anxious to salvage the mine and abandon the facilities very quickly. Closed and subsequently abandoned shafts are available but would be more difficult to engineer and therefore more expensive due to re-entry costs. As much of the basic work in Tasks 5.3 and 5.4 was completed using a generic model, then, although later work was conducted on a specific UK case study, Maltby No. 3 shaft, all of the findings of the research could be of equal validity in the development of such a facility using an abandoned mine-shaft anywhere in Europe. The main conclusion of the work was that a resilient, isolated pressure chamber must be engineered to give any chance of success.

Task 5.5 Exploitation of Heat Storage for Energy Generation System Efficiency

To review the current span of established heat-store methodologies with emphasis on liquid-solid phase-change approaches; to consider cavern-lining methodologies for enhanced thermal insulation; to examine pressurised thin-walled tank possibilities; to consider 'nested' heat storage configurations; to undertake basic modelling of cost-performance; to integrate simple heat-store and heat-exchange models into system dynamic models.

Activities and Discussion: The options for heat storage for CAES were described in Task 5.2. This task investigated heat storage for energy generation, including sensible, latent and chemical options. Appendix 29 shows the categories as well as a diagram of the behaviour of water to explain latent heat related to phase changes, although suitable substances are typically paraffin waxes, hydrated salts or even certain metals. Different sensible systems were compared.

The flow reversal reactor (FRR) has emerged as the proven VAM energy generation system for detailed study. The use of a heat storage system with the FRR, capable of storing excess heat at high methane concentration periods and returning it to the reactor during low concentration periods, can greatly improve the efficiency and stability. After examining the different heat storage technologies proposed, sensible heat storage in a ceramic packed bed was found the most appropriate as, while not being the most efficient, it is very flexible, easily dealing with dynamic changes in temperature associated with the operation of the reactor.

Figure 12 shows the working principle of the heat storage system proposed for the flow reversal reactor. Beds A and B are responsible of heating the feed up to the reaction temperature and cooling down the hot gases. When the feed concentration is high (left figure), a valve diverts part of the hot gas from the reactor to bed C, where the excess of heat is stored at high temperature (~900ºC in thermal combustion and ~500ºC in catalytic). When the feed concentration is low (right figure), a valve diverts part of the cold feed through bed C to extract the stored heat stored. The valves are regulated using PID controllers in order to obtain the desired temperature in the reaction zone (between beds A and B).

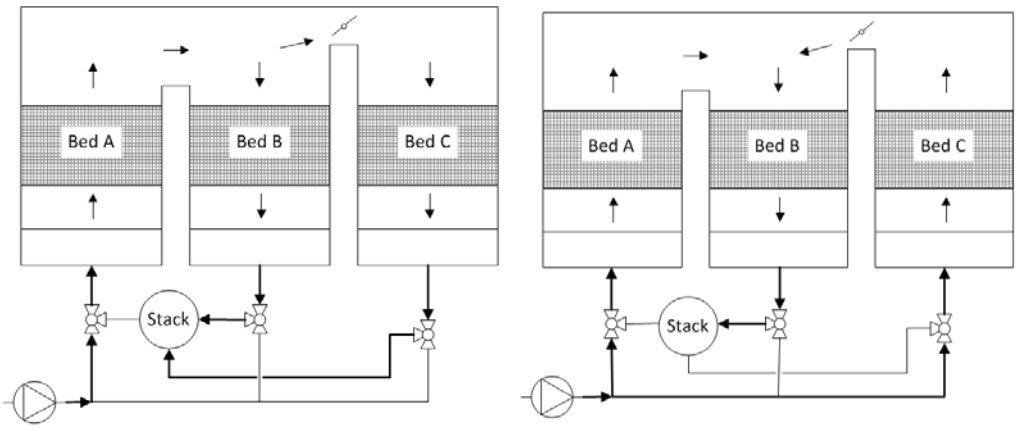

Figure 12: Working principle of the novel heat storage system

The heat storage system has been modelled using a mathematical model obtained from the conservation principles (mass and energy). The model has been integrated into the FRR model (Tasks 3.1; 6.1) to simulate the dynamics of the combined system, enabling heat storage bed design, operation optimization and tuning of the controllers.

Conclusions: It was concluded that latent heat storage concepts are currently too expensive for practical application and hence sensible heat storage would be pursued. A comparison of active and passive systems is shown in Appendix 29. Passive ceramics were identified for this work.

The simulation of a TFRR equipped with a heat storage system under real coal mine ventilation air conditions indicated that the proposed heat storage system is capable of achieving stable reactor operation when CH_4 concentration varies in the 0.5-0.25% range. The results are summarized in Appendix 30, the most important being that the system maintains the gas temperature in the reaction zone within a narrow interval centred on 950ºC, improving the efficiency of energy recovery and any associated generation system.

Exploitation and Impact: This heat storage system concept is in publication and is ready for the next research stage at pilot scale to confirm the simulations before proposing a commercial application.

P. Marín, F.V. Díez, S. Ordóñez, "A new method for controlling the ignition state of a regenerative combustor using a heat storage device", Applied Energy (accepted for publication, 2013)

Work Package 6 Potential Impact of Low Carbon Mining

***Objectives of WP6**: The output of this WP is to evaluate and to quantify the impact of the low carbon approach as defined within this project, investigating the exploitation of each of the sub-themes. Specifically: to test and evaluate low carbon mining technologies and initiatives including operational trials or simulations of energy efficiency systems, mine site energy generation and compressed air energy storage / heat storage; to conduct an economic analysis of methane extraction prior to mining and to produce guidelines for various coal mining conditions; to evaluate the techno-economic model of implementing a 'low carbon' approach in underground mine operations.*

Comparison of initially planned activities and work accomplished: The work has been completed, including a wide range of modelling and field trials in Task 6.1*, an economic report for the trial of pre-mining methane drainage in Task 6.2 and a technical and economic assessment of selected areas considered appropriate for the techno-economic model defined. This last Task may have been originally conceived as a total application of all elements, but this has proved to be impractical as some are mutually incompatible, insufficiently developed for such detailed consideration or already considered from a cost viewpoint in previous Tasks.

*Although referred to additionally as part of Task 6.1, as extensive modelling work for compressed air energy storage has been reported in Work Package 5 above it will not be repeated in this Work Package.

Task 6.1 Testing & Evaluation of Low Carbon Mining Technologies including Operational Trials & Simulations

To test and to evaluate: mine site energy generation (WP3); energy efficiency systems (WP4); compressed air energy storage / heat storage (WP5)*

Activities and Discussion: The simulation of flow reversal reactor (FRR) performance uses the code developed under Task 3.1 to solve a mathematical model. This is updated with the specific geometrical and physical properties of the inert bed material, the ventilation air properties, and the reactor design. Figure 13 shows the simulated profiles of concentration and temperature between two consecutive flow reversals at 5 m^3/s with 0.25% methane.

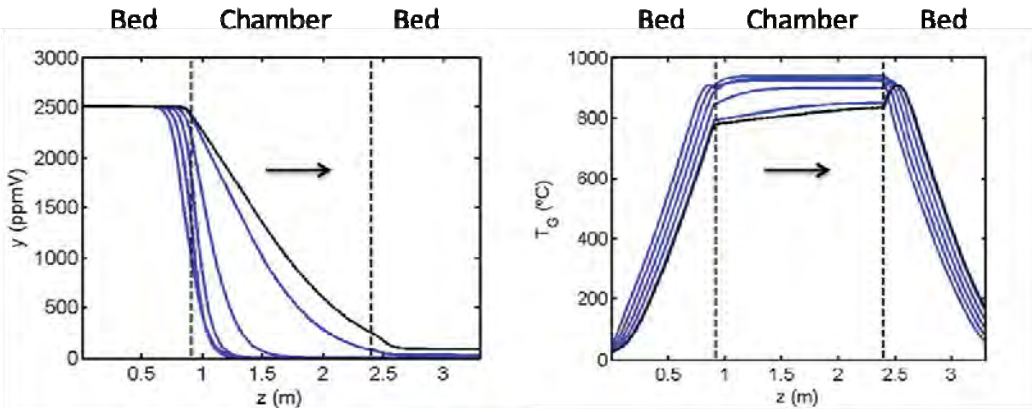

Figure 13: Evolution of concentration and temperature between flow reversals

At the pseudo-steady state and in the absence of further disturbances, this evolution will be repeated cycle after cycle. The methane concentration profiles indicate that some combustion takes place already in the inert bed, but conversion is higher in the chamber. Complete methane conversion is achieved by the end of the reactor. The influence of methane feed concentration on the stability of the reactor has been studied by means of additional simulations. Appendix 31 shows a comparison of heat extraction strategies to achieve reactor stability.

Review of options for harnessing the gas turbine principle with ventilation air methane had led to the concept shown in Appendix 32 that uses an array of small standard cogeneration gas turbines running on clean imported gas providing electricity and oxidising the VAM with the exhaust heat. The oxidation depends upon the process of heat capture in a combustion zone taking recycled heat from a heat exchanger feeding back to a pre-heater, aided by supplementary gas-fired igniters. Subsequent to the intended total VAM combustion in this zone, a steam turbine may be used for further electricity generation from the combined heat output together with other heat load uses. In

addition to combustion modelling, much of the modelling work was to design the primary combustion zone heat exchanger. To find 'ideal energy', mass flow rate and conversion to metric tonnes/hour were supplied for 600°K, 700°K and 800°K steam temperatures at the flow rate of the ventilation air of 100m^3/s. The availability of steam quoted in tonnes/hr is an industry standard quantification typically used to characterize the steam generated by a boiler. The calculation is based on raising the temperature of water initially at 288.15°K to steam at 600°K, 700°K or 800°K. A loss factor is then applied where it is assumed that usually about 93% of the energy is captured. The energy transferred from the hot gas is calculated based on an average density and average specific heat capacity over the temperature range. The thermodynamic characteristics of each fluid and the metal all have an effect on the overall heat transfer coefficient. The characterization of a heat exchanger by the Log Mean Temperature Difference (LMTD) is a standard method of heat exchanger design (Lienhard IV and Lienhard V, 2000). When the overall heat transfer coefficient and absorbed energy are known, the contact area of the flow tubes can be determined. Designs followed extensive computation; the heat exchanger structure in Solidworks© shown in Appendix 32 shows an example of the output, while Appendix 33 shows some examples of the combustion and heat exchange modelling. A modest field test was also attempted to test and adapt the principle of hydrocarbon oxidation in the heat of the exhaust stream (Appendix 34), but unfortunately the small gas turbine available did not function sufficiently well to obtain usable results.

Pump efficiency measurement using the Poirson method refined with modern sensor technology in the CSM mine simulator (see Task 4.4) has been subjected to two field trials. It had been hoped to test the pumps at Velenje coal mine but this proved not to be possible for operational reasons and the ventilation fan was investigated there instead. Thus the pump at the CSM Test Mine and a large pumping operation for high-pressure extraction at a china clay (kaolin) pit in Cornwall using comparable pressures to a deep mine were tested:

- At the Test Mine, the natural ingress and production water gravitates to a central sump and the three-phase 7.5HP centrifugal pump transfers this water to surface tanks to supply the production network, excess water being drained away via an overflow. As this pump is old and the manufacturer is no longer available, estimation of motor characteristics was necessary. Results are shown in Appendix 35.
- At the china clay pit, a rack of four 355kW horizontally-mounted centrifugal pumps is used in various combinations to power a recycling water ring main to provide high-pressure monitor (hosing) operations to wash out the clay. Here, although records were scant, information was available from site and manufacturer engineers to help to specify motor efficiency. Results are shown in Appendix 36.

For the ventilation improvement work at Velenje mine, described in Task 4.3 above, data had been taken from various sources: the fan; mine air speed sensors; in situ measurements (in the mine and at the fan). As the goal is to maximise efficiency, electricity consumption and cost has been correlated. Comparison of field data had revealed some mismatches and a trial exercise was set up for a non-production day. Appendix 37 shows the trial comparison between different measuring techniques related to fan plate angles and the power consumption increases. In production time, usually the fan plate angles are changed if the gas situation at the mine face becomes unacceptable. The Poirson thermodynamic approach was also tested (Appendix 38), with the sensors developed in the CSM mine simulator being inserted across the fan and readings taken over a small range of fan blade angles. Data analysis was complicated due to the up-cast shaft driven air being in a saturated condition and a correction had to be made for increase in temperature per unit pressure. Appendix 39 shows the comparison of the conventional and thermodynamic methods in overlaid graphical form.

*The extensive modelling work related to compressed air energy storage is reported in detail in Work Package 5 above and will not be repeated here.

Conclusions: The further FRR simulations show that below 0.25% methane the reactor cannot be operated at autothermal conditions and extinction takes place progressively. This is an important finding, because if the concentration decreases only for a short period of time and then increases above 0.25%, the reactor will not extinguish. When the concentration at the feed increases above 0.30%, the temperature in the combustion chamber increases above 1000ºC, which can be dangerous. For this reason, the temperature must be controlled in order to prevent over-heating of the reactor. The two heat extraction strategies proposed are:

a) Part of the hot gas is drained from the middle of the reactor with no return. The heat is recovered as sensible heat from the drained gas.
b) The whole gas stream is partially cooled down in a heat exchanger and then returned to the reactor.

The reactor was simulated for both strategies. Option (a) produces very little disruption on the reactor performance, apart from the desired temperature decrease, while option (b) breaks the parabolic profile of the reactor, resulting in a loss of symmetry and hence a decrease on the reactor efficiency. Nevertheless, (b) has an important advantage in that all the energy is recovered at high temperature, giving higher efficiency for subsequent use. For this reason, (b) may be favoured for applying to heat loads whereas (a) is more efficient for typical methane oxidation initiatives for carbon credits (see Appendix 31).

For the gas turbine configuration (VamTurBurner©), results of a numerical investigation, using large-eddy simulation, of the key process, based on the ignition and combustion of preheated VAM by means of a primary fuel injection acting as a pilot or primary flame, have been studied (Mira-Martinez et al, 2014). The VAM mixture is supplied as a co-flowing stream and after the interaction with the primary flame, it ignites provided certain conditions are provided. The preheating temperature and methane concentration were studied and it was found that the preheating temperature has a significant influence on the ignition characteristics of the VAM mixture. By varying the methane concentration from 0.5 % to 2.0 % it was shown that a secondary flame is developed by the oxidation of the methane contained in the VAM, while for the low methane concentration of 0.5%, the combustion only takes place when the temperature of the mixture is above 500 K. Ignition timing was also described and discussed for the different mixtures providing some insight into the flame dynamics in each case. A description of the flame structure and mixing evolution is also addressed using time-averaged results for the main reactants and products as well as the 'unmixedness' parameter. This study proves that the ventilation air methane flow ignites under certain conditions and motivates further analysis of the combustion characteristics of such mixtures. The concept would require a reliable source of clean gas and a complete redesign of a mine's approach to energy management, but remains an interesting concept for future consideration and adaptation.

For the pump efficiency measurement by Poirson technique, the pump results shown in Appendices 35 and 36 show the hydraulic efficiencies. Table 8 shows a financial assessment for the CSM Test Mine pump, where a direct pump refurbishment to original conditions would present an unnecessary increase in flow rate and head for the work done, suggesting that a refurbishment with associated reduction in diameter to current dimensions is the favoured approach for this very worn pump. This would reduce annual energy cost by 25%, achieving close to BEP for a modest financial outlay. Considering the hydraulic efficiency outcomes for the four clay site pumps (Appendix 36), the first two are approaching 80% and hence are working at a reasonably efficient level. However, the third and certainly the fourth in the array would benefit from some targeted refurbishment that will provide significant energy savings over time.

Table 8: CSM Test Mine pumping energy assessment

Parameter:	Current condition	OEM 'post refurbishment' condition	Diameter change and refurbishment	Units
Head	25.7	28.8	25.7	m
Flow rate	10.5	12.3	10.5	l/s
Electrical power	6.2	6.4	4.6	kW
Hydraulic efficiency	50.4	63.4	66.6	%
Operating time	37.0	31.5	37.0	%
Volume pumped		122,517		m^3/year
Energy	20095.4	17628.7	15062.0	kWh/year
Tariff		0.125		£/kWh
Energy cost	2511.93	2203.58	1882.76	£/year
Specific power	0.021	0.018	0.015	kWh/m^3
Reduction	-	12.3	25.0	%

The Velenje ventilation trials highlighted the need for a total mine system approach. This could be seen from the trial measurement results where increase of air flow at one place (e.g. mine face)

influences performance in other places. A systematic approach is needed where the efficiency of both fans will be considered in such a way that optimum fan operating points are found. From such measurement comes an understanding of the costs of additional airflow at mine faces. It is proposed that before changing the fan plate angles, which is usually triggered by a request for more airflow at a mine face (e.g. due to measured gas increase), a due diligence update should be carried out of data in the ventilation software (e.g. changes in airways cross sections, resistances) and then a simulation of the suggested change quickly run to assess the overall impact, using the Zračenje software upgraded during this project. If everything is positive then the change will be carried out or alternatives proposed. Fan plate angle changes must be carried out gradually with enough time between changes for mine behaviour to be effectively monitored.

For the Poirson thermodynamic analysis of the mine air fan, the size of the temperature differential was large as compared to water applications, with an increase in temperature across the fan in the order of 1-2°C. This is due to the low density and most importantly the low specific heat capacity of air. The results proved very sensitive to the moisture content of the air and a saturated condition equation had to be used to describe successfully the thermodynamic conditions across the fan. Following adjustment, the results, shown in Appendix 39, show a generally good agreement between the thermodynamic method (green dots) and conventional method (blue dots) for all three graphs (differential pressure, electrical power and pneumatic efficiency all against volumetric flow rate). The thermodynamic method shows a greater differential pressure at each point and so, with solution of this anomaly, the results could show closer agreement. The electrical power curves are not on top of each other. As the same electrical power was used for each method then this must either be a (synchronous) sampling issue or an error in flow rate for one or both methods. The pneumatic efficiency curve shows very good agreement. The general conclusion for this novel application is that, while the method is not yet as refined as for water pumps, with growing understanding of the impact of shaft conditions the thermodynamic approach does offer an effective low-cost fan efficiency monitoring method.

Exploitation and Impact: The greater understanding of the processes within the flow reversal reactor have enabled improved reactor design, with potential for construction at a Hunosa site if suitable funding should become available.

The novel gas turbine configuration concept has been presented at an international mining conference and is the topic of a submitted journal paper awaiting assessment.

As stated above, the water pump efficiency work has been presented at two mining conferences and is available for uptake by the coal industry. Further consideration will be given to the novel adaptation of this principle to air fans, where shaft conditions have proved to be challenging.

Velenje mine has greater control over the management of ventilation at this large and unusual underground lignite operation, although a financial benefit has not yet emerged.

Cluff D L, Kennedy G A, Bennett J G (2013); Capturing Energy from Ventilation Air Methane; Proceedings of the 23rd World Mining Congress; ISBN: 978-1-926872-15-5

Mira-Martinez D, Cluff D L, Jiang X (2014) ; Numerical investigation of the burning characteristics of ventilation air methane in a combustion based mitigation system; submitted to Fuel, January 2014

Task 6.2 Economic Analysis of Pre-mining Methane Drainage Technology for Underground Coal Mines

To conduct an economic analysis of the costs associated with pre-mining methane drainage technology (WP2) to produce a model and guidelines for mining operations.

Activities and Discussion: For extraction from coal seams in high methane risk conditions it is inevitable to incur additional related costs, e.g. the necessity for special development, extraction and ventilations systems, maintaining extended departments for methane hazard prediction, identification and control, execution of expanded safety precaution measures, application of appropriate machinery, devices and materials. This imposes a considerable additional cost on each tonne of coal extracted. This additional cost may be reduced by application of methane drainage (before or during extraction) and sale or utilisation of captured methane. This requires a methane drainage department to drill boreholes, lay and relay pipelines and maintain the network and equipment, including the methane drainage station. Captured methane enhances safety and mining

conditions underground and is next sold or used for energy/heat production at mine methane based energy production unit. In order to assess the costs and benefits of pre-mining methane drainage, the various costs incurred by the collieries were identified and analysed, based on the data of Brzeszcze Colliery. Brzeszcze Colliery is rated among the gassiest collieries in the Polish as well as European mining industry. The mean methane release intensity in 2012 amounted to 197.5m^3 CH4pmin, of which 73.1m^3 CH4pmin (37% - ie. 38.1 mio.m^3) were captured by means of methane drainage systems and sold to an external purchaser. The yearly coal production amounted to 1.783 mT thus the mean cost of methane drainage was 8.7 PLNpT of coal.

A working model for pre-mining methane drainage economy assessment was produced according to the UNIDO methodology and Dynamic Generation Cost (DGC), Net Present Value (NPV) and Internal Rate of Return – (IRR) were proposed as an assessment criteria.

Conclusions: Based on the investigations at Brzeszcze Colliery (data for years 2009-2012), the following main cost groups were identified relating to the methane drainage department: wages with contributions and additions (71.3%), energy (21.1%) and other costs (7.6%). In the year 2012, the captured methane was sold to external purchaser for 289.91 PLN per thousand m^3, covering 76% of methane drainage costs. The cost of the underground trial of pre-mining methane drainage was twice as high as that of classic methane drainage technology. Calculated values of NPV (-1,833,900 PLN) and DGC (1.37 PLNpm^3 for pre-mining and 0.43 PLNpm^3 for classic) indicates that pre-mining drainage is not economically efficient. The results of a sensitivity analysis, when the deviations of methane pre-mining drainage efficiency (% of captured methane/absolute methane bearing capacity) and direct costs were simulated, show that even if the efficiency increases to 50% (higher than for the classic method), it is still not profitable. The increase of distance between drainage boreholes from 10 m (tested value) to 20 m, with the assumption that the CH_4 out-flow remains at the same level, or the increase of a CH_4 out-flow from the boreholes (more than 7 times) makes it profitable. Based on this assessment, it can be stated that the longer pre-mining drainage lasts the higher the efficiency (the calculations were made for a period up to 800 days). But it must be emphasised that if a CH_4 out-flow increase is not achieved it will be still not profitable.

Exploitation and Impact: This analysis supports the conclusion from Task 2.4 that pre-mining methane drainage technology in this form is not suitable for the conditions of Polish hard coal mining. There will, however, be further research including hydraulic fracturing, open or cased hole cavitation, high pressure water jetting or use of explosives, in particular within GasDrain, a new RFCS project commencing in July 2014.

Task 6.3 Evaluate the Techno-economic Model of Implementing a 'Low Carbon' Approach in Underground Mining Operations

To construct a techno-economic model on the feasibility of a 'low carbon' approach to underground mining operations against representative mining systems in the EU.

Activities and Discussion: Taking the concept of a viable mining 'model', this review covers the proven technologies from the project with possibilities of early practical implementation, i.e. flow reversal reactor (FRR); potential heat output from FRR to co-generation; pump efficiency. Other areas were considered either more speculative concepts (CAES, gas turbine configuration, where some financial assessment was included in earlier Tasks) or applied to unique mine situations (Velenje ventilation management). Figure 14 shows a proposed high-level techno-economic model. The inputs and outputs are clear enough. The technical viability presents the first question, then the efficiency at a suitable scale. From this, the question of possible subsidy (e.g. carbon credits) is an input to a traditional investment review of capital and operating costs against benefits over a chosen time period. 'Future concepts' are included in case of either further development or the emergence of new 'game-changing' technologies. The area of ventilation air methane oxidation has been chosen for in-depth study, leading to work on the thermal flow reversal reactor (TFRR) and subsequent potential to harness this for co- or tri-generation of electricity, heat and cooling, and some economic savings from the trial work on pumping improvement through improved monitoring. It should be noted that considerable technical and economic work had gone into the more esoteric areas of energy storage, gas turbine configuration and catalytic solutions in earlier Tasks, while the methane drainage work in Poland is covered in Task 6.2.

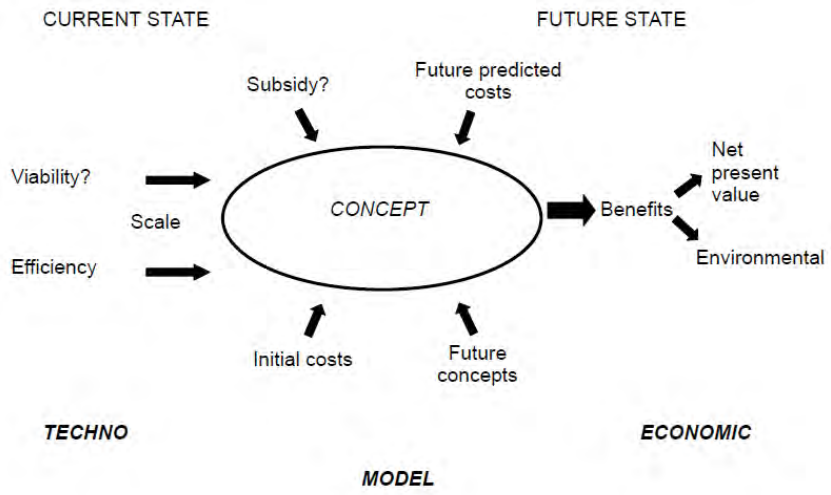

Figure 14: High-level techno-economic model

Regarding the social and political dimensions, coal mine methane is the area within the project that is most significant, with a geo-political dimension, although the potential of energy storage and electrical demand management will be critical to plans to implement smart grids in the future. The strategic objective to oxidize as far as possible drained and abandoned mine methane was described in Task 1.1, also pointing out the environmental damage from coal power station emissions that can be reduced by greater electrical efficiency. A recent collapse in the price of carbon credits such as the EU Allowance (EUA) has run contrary to the expressed strategies of governments and many major companies and also had a huge impact upon the economic projections herein.

Flow Reversal Reactor: Arising from a partner's expertise and the consequent specification of the LOWCARB project, a considerable amount of analysis and experimental work had been given to the catalytic flow-reversal reactor (CFRR), described in other Tasks above. Although offering a number of advantages, this technology is currently not viable due to the high prices of the catalysts (over €300,000 for one bed), so this section is based upon the thermal reactor (TFRR). As VAM concentration is critical, a programme of measurement took place at the five HUNOSA upcast shafts, showing that only one, the Maria Luisa mine, fell consistently within the range 0.18%-1.0% methane suitable for TFRR. To provide a technical basis for the model, the Spanish partners then undertook a complete TFRR design exercise based upon this shaft. Some images from this work are shown at Appendix 40. The zones of distribution of the gas stream have polygonal form with dimensions 2.5 m x 3.75 m, and 3.75 m in height. There are deflectors to achieve a uniform distribution of the flow. For the bed design, a cordierite bricks 'honeycomb' is proposed. These bricks have a high thermal resistance (maximum 1300 ºC) and thermal shock resistance (500 K). The specifications of the material include a density of 2,400 kg/m^3, a caloric capacity of 1,000 J/kgK and a thermal conductivity of 2 W/mK.

With this technical proposition for the shaft, economic cases were considered, some output being shown in the conclusions below.

Tri-generation: The graph at Appendix 17 shows the percentage of heat energy recoverable in a flow reversal reaction. Reactor designs have differing bed characteristics and hence critical VAM concentrations for heat by-pass to become necessary, but this exercise takes an accepted general minimum of 0.45% VAM for energy to become available for consequent use in electricity generation. This chart also shows the unsuitability of the catalytic reactor for co-generation, with such reactors working at much lower temperatures with energy trapped in the catalytic process rather than being released. The schematic shown in Appendix 17 forms the basis for the following assessment, the technical elements having been briefly described in Task 3.3. A ventilation air flow of 720,000 m^3/hour, typical of a large mine, with VAM averaging 0.5% provides 3,600m^3 CH_4 per hour (2.42 tonnes), a potential of 38MWh of energy (from 39MJ/m^3 for methane). An unconnected hood can safely capture around 80%, providing around 30MWh of potential energy in an hour. The distance from the hood to the TFRR inlet is governed by pressure drop, duct diameter and primarily the safety system. A good quality modern laser methane detector takes 1 second to respond to a methane cloud greater than 1.5% concentration and the isolation damper to cut out the flow to the

reactors takes 3 seconds, so 5 seconds is a reasonable minimum safety interval. A well-designed and constructed TFRR should perform at around 95% efficiency in oxidising the CH_4, but with a parasitic load from the fan (e.g. in the order of 120kW) that draws the air through the system. The next stage in the system is a heat recovery steam generator (HRSG) to capture the thermal energy in usable form. This comprises a modular system of economizer, evaporator, super-heater and water pre-heater. Flows, pinch and approach points and surface exposures should be carefully modelled to the intended use to create the modular design. High pressure steam, e.g. 22,000kg/hour; 350^{0}C; 21bar, drives a steam turbine that turns an electricity generator. Applying the general process with losses at all points to the theoretical mine, it is estimated that a customised range of TFRRs could generate sufficient heat to produce a continuous 4MW of electrical power. The extension to air heaters in winter or absorption chillers to create ice-water for hot face conditions or perhaps off-site sale (generating cooling power from heat input) using the low pressure steam from the back of the HRSG/turbine is not believed to be in use anywhere but might offer an tri-generation option where shaft proximity enables insulated steam pipes at a tolerable length for cost and heat retention. A 500 ton (1.76MW output) absorption chiller with a typical coefficient of performance (COP) of 0.7 would require around 4,100kg/hour of low pressure steam at around 115^{0}C to input 1.76 / 0.7 = 2.51 MWh of heat energy in an hour. With 22,000kg/hour high pressure steam entering the HRSG, this should be well within the capability.

Based on known mines, a figure has been taken of €500,000 annual saving in purchased electricity costs from the continuous provision of 1MW of electrical power (i.e. 8760MWh in a year). For the above 'VAM to electricity' installation, fed by an array of seven TFRRs including redundancy and legal and other project costs, a capital investment of €10m has been estimated (rounded as legal and other management costs may vary). Maintenance and repair cost take €300,000 per year. TFRR fan costs are significant (4MW – 0.72MW for six concurrent fans = 3.28MW overall output). To include shaft heating and chilling, an absorption chiller would cost €140,000 and a further €40,000 for heating radiators, ducting, insulation, etc. A spreadsheet was developed to assess the investment opportunities, some output from which is shown in the conclusions below.

For the pumping trial of four pumps at the china clay pit described in Task 6.1, the hydraulic efficiencies observed, in the solo configurations, were 78.2%, 80.5%, 74.5% and 68.0% for pumps 1-4 respectively. Therefore the best machine (pump 2) was 5.8% below the optimum case and the worst machine (pump 4) is 18.3% below. The normal operating regime is running two pumps in parallel and so, assuming the pumps are rotated evenly in time, the hydraulic and economic conditions are shown in Appendix 41.

Conclusions: TFRR: For a project duration of six years (the anticipated life of the mine at the time) with a single reactor at a subsidy rate of 7.1 euro per tonne CO_2eq (proposed by the Spanish Office of Climate Change but with certain limitations that in the event were too stringent), the following projection was made on the basis of the reactor's capability for saving 114,346 tCO_2eq.

R=4%	year 0	2013	2014	2015 - 2018
CASH FLOW (euro)	**- 250,000**	-24,690	59,750	100,310

NPV	**113,600 €**
IRR	**14 %**

The NPV was then tracked for the above technical specifications for different EUA credit prices, revealing €5.92 as the point where NPV became positive. The then current market price of €4.59 is added to the chart on the right to demonstrate that it was in a negative area.

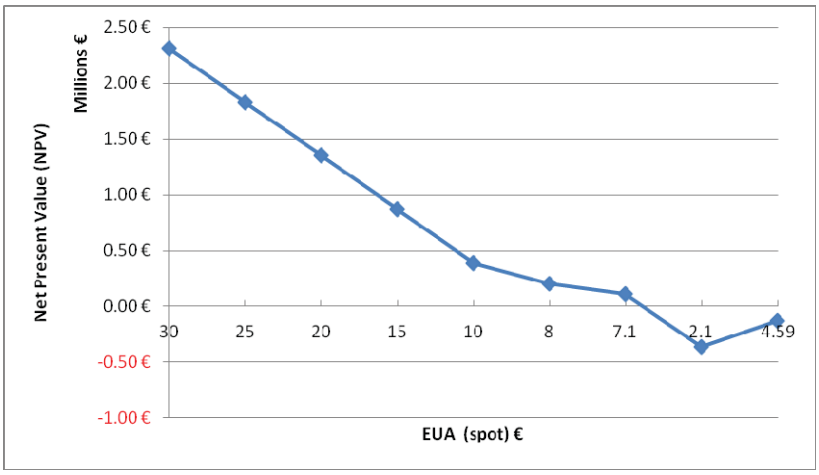

Figure 15: NPV for TFRR vs C credit prices

Tri-generation: The following two charts (Figure 16), one to show the impact of different rates of change in electricity prices and the other to show different (low) credit prices at a consistent 2% electricity price, demonstrate that carbon credits are needed to support the investment, but that at current low rates even these do not make it attractive.

The further chart (Figure 17) shows that, even if credits return to a higher rate, while the basic investment in methane mitigation for credits would be profitable (as it was considered by the investment community only a few years ago), the tri-generation option does not provide an attractive financial addition.

Repair & maintenance increases 2%pa Propane price increases by 1.5%pa

Capital expenditure Year 0 = €8,124,000	Discount rate 5%	Discount rate 7.5%	Discount rate 10%
Electricity price increase per year:	Net Present Value €s (Energy savings only/no C credits)		
2%	10 years 3,540,019 15 years 8,455,978	10 years 2,135,200 15 years 5,762,619	10 years 960,102 15 years 3,658,370
3%	10 years 4,284,835 15 years 9,959,122	10 years 2,770,189 15 years 6,955,001	10 years 1,505,301 15 years 4,616,616
4%	10 years 5,073,431 15 years 11,600,699	10 years 3,441,544 15 years 8,253,035	10 years 2,080,899 15 years 5,656,400
NOTES: Even assuming an average price rise of electricity of four per cent (unlikely over a long period), the return is completely inadequate for simply installing such a system.			

Repair/maintenance increases 2%pa Electricity price increase 2%pa Propane price increases 1.5%pa

Capital expenditure year 0 = €8,124,000	Discount rate 5%	Discount rate 7.5%	Discount rate 10%
Carbon credit price per tCO_2eq:	Net Present Value €s (Energy savings & C credits)		
€1	10 years 6,657,174 15 years 12,531,061	10 years 4,945,252 15 years 9,280,122	10 years 3,512,552 15 years 6,737,500
€3	10 years 12,867,555 15 years 20,634,037	10 years 10,544,934 15 years 16,277,601	10 years 8,599,898 15 years 12,865,528
€4	10 years 15,972,746 15 years 24,685,526	10 years 13,344,775 15 years 19,776,341	10 years 11,143,571 15 years 15,929,542
NOTES: Here the complete tri-generation system, backed with sale of credits at current low rates of around €4, fails to provide an attractive pay-back over 10 years, especially if capital pushes the discount rate to 10%, which is likely.			

Figure 16: NPV for tri-generation with & without C credits

Repair/maintenance increases 2%pa Electricity price increase 2%pa Propane price increases 1.5%pa

	Capex	C Credit / tCO_2eq	Discount	NPV 10 years
7 TFRRs; hood; pipes	€5,000,000	€13	10%	€28,067,750
Trigen system	€10,180,000	€13	10%	€34,036,629
Difference=	€5,180,000		Difference =	€5,968,879

Figure 17: Comparison TFRR system for oxidation only vs full Tri-generation system

One aspect that emerges from this study is that a strong annual increase in the cost of electricity (e.g. 3% a year rather than 2%) only marginally increases the NPV. It is the carbon credits that make the real difference for the investors, which is unfortunate in a way as this is an artificial market rather than a basic resource. There may, of course, be other reasons for investing in self-sufficiency in energy other than simply profit if the system covers its cost, such as the South African situation where grid supply is uncertain.

Exploitation and Impact: The Spanish partners had been seeking funding subsidy to consider building the proposed reactor design at Maria Luisa. However, the subsidy was not sufficient and now the expected mine life is too short. The co-generation concept of VAM to electricity and potentially the full tri-generation can be pursued to design stage through an industrial contact for a mine with suitable VAM concentrations, which may become viable in regions of China if carbon credit prices rise. The pump monitoring sensor system is now a proven technology available to any site with large industrial pumps and with particular relevance to deep mines with very high 'head'.

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4. FIGURES & TABLES

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5. ACRONYMS & ABBREVIATIONS

ACAD	Automated Computer-aided Design (design software)
ADELE	Adiabate Druckluftspeicher für die Elektrizitätsversorgung (German CAES project)
ALMEMO™	Ahlborn universal measuring instrument
AMM	Abandoned Mine methane
ATEX	Appareils destinés à être utilisés en ATmosphères EXplosives (EU Directive)
BEMS	Building Energy Management System
BEP	Best Efficiency Point (pumping)
BET	Brunauer, Emmett & Teller (gas adsorption theory)
CAES	Compressed Air Energy Storage
CAPEX	Capital Expenditure
CDM	Clean Development Mechanism (Kyoto Agreement)
CESS	Central Energy Surveillance System
CFRR	Catalytic Flow Reversal Reactor
CHP	Combined Heat & Power
CMM	Coal Mine Methane
CSIRO	Commonwealth Scientific and Industrial Research Organisation (Australia)
CT	Current Transformer
DSM	Demand Side Management
ECLIPSE	software platform
EIEO	Electricity In – Electricity Out
ER	Eley-Rideal (kinetic model)
ETMDC	Electromagnetic Transients including Direct Current (calculation software)
EUA	European Union Allowance (C credit)
$FLAC^{2D}$	Fast Lagrangian Analysis of Continua (geotechnical modelling software)
GPR	Ground Penetrating Radar
GT	Gas Turbine
HBG	Horizontal BackGround remover (GPR processing technique)
HP	High Pressure (air)
HSC	High Strength Concrete
HYSYS	Process simulation software
IEA	International Energy Agency
IPCC	Intergovernmental Panel on Climate Change
KS	Kolmogorov-Smirnov Test (statistical test)
LCD	Liquid Crystal Display
LH	Langmuir-Hinselwood (kinetic model)
LMTD	Log Mean Temperature Difference (heat exchanger design))
MATLAB	Matrix Laboratory (modelling software)
NSC	Normal Strength Concrete
MODBUS ®	Modicon electronic messaging protocol
MPa	MegaPascal (material strength)
Mtoe	Megatonnes Oil Equivalent
MVK	Mars-Van Krevelan (kinetic model)
OPC	Open Process Control
ORC	Organic Rankine Cycle
PETREL	Geophysical software
PFC	Particle Flow Code (distinct element modelling software)
PSCAD®	Power Systems Computer-aided Design (design software)
PT	Potential Transformer
SCADA	Supervisory Control & Data Acquisition system
SIMULINK	MATLab simulation option
SMES	Superconducting Magnetic Energy Storage
SQL	Structured Query Language (software language)
TES	Thermal Energy Storage
TFRR	Thermal Flow Reversal Reactor
UCS	Uniaxial Compressive Strength
VAM	Ventilation Air Methane
VESC	Volumetric Energy Storage Capacity

VOC	Volatile Organic Compounds
VTIS	Velenje safety & technology information system
WHRB	Waste Heat Recovery Boiler
XRD	X-Ray Diffraction

6. LIST OF PUBLICATIONS & PATENTS

Bennett J G, Kennedy G A, Foster P J, Williams N C, Cluff D L, Clifford T P (2013); Low Carbon Coal Mining – A Contradiction or an Opportunity?; *Proceedings of the* 23rd World Mining Congress; ISBN: 978-1-926872-15-5

Bukowska M, Sanetra U, Wadas M, 2012; Chronostratigraphic and Depth Variability of Porosity and Strength of Hard Coals of the Upper Silesian Basin; Mineral Resources Management; Vol. 28, no. 4.

Cluff D L, Kennedy G A, Bennett J G (2013); Capturing Energy from Ventilation Air Methane; *Proceedings of the* 23rd World Mining Congress; ISBN: 978-1-926872-15-5

Díaz E, Fernández J, Ordóñez S, Canto N, González A (2012); Carbon and ecological footprints as tools for evaluating the environmental impact of coal mine ventilation air; Ecological Indicators; 18, 126

Gutiérrez I, Díaz E, Vega A, Ordóñez S (2013); Consequences of cavity size and chemical environment on the adsorption properties of isoreticular metal-organic frameworks: An inverse gas chromatography study. Journal of Chromatography A 1274, 173-180.

Gutiérrez I, Díaz E, Ordóñez S (2013); Consequences of cavity size and palladium addition on the selective hydrogen adsorption in isoreticular metal-organic frameworks; Thermochimica Acta. doi: 10.1016/j.tca.2013.01.007

Kidybinski A (2011); The effect of porosity and the strength of coal on the dynamics of coal and methane outburst – the BPM modelling; Archives of Mining Sciences; Vol 56 No 3 pp 415-426

Kidybinski A (2012); Mierzalne czynniki zagrożenia wyrzutem węgla i metanu w przodku wyrobiska korytarzowego – wg badań na modelu numerycznym, Przegląd Górniczy nr 1/2012, str.20-26. (Measurable parameters of coal and methane outburst hazard in the heading face of a roadway; Mining Review; No.1 pp.20-26, – *in Polish*)

Marín P, Díez F V, Ordóñez S; A new method for controlling the ignition state of a regenerative combustor using a heat storage device; Applied Energy (accepted for publication, 2013)

Marín P, Ordóñez S, Díez F V (2011); Performance of silicon-carbide foams as supports for Pd-based methane combustion catalysts; Journal of Chemical Technology and Biotechnology; doi: 10.1002/jctb.2726

Mira-Martinez D, Cluff D L, Jiang X (2014); Numerical investigation of the burning characteristics of ventilation air methane in a combustion based mitigation system; *submitted to Fuel, January 2014*

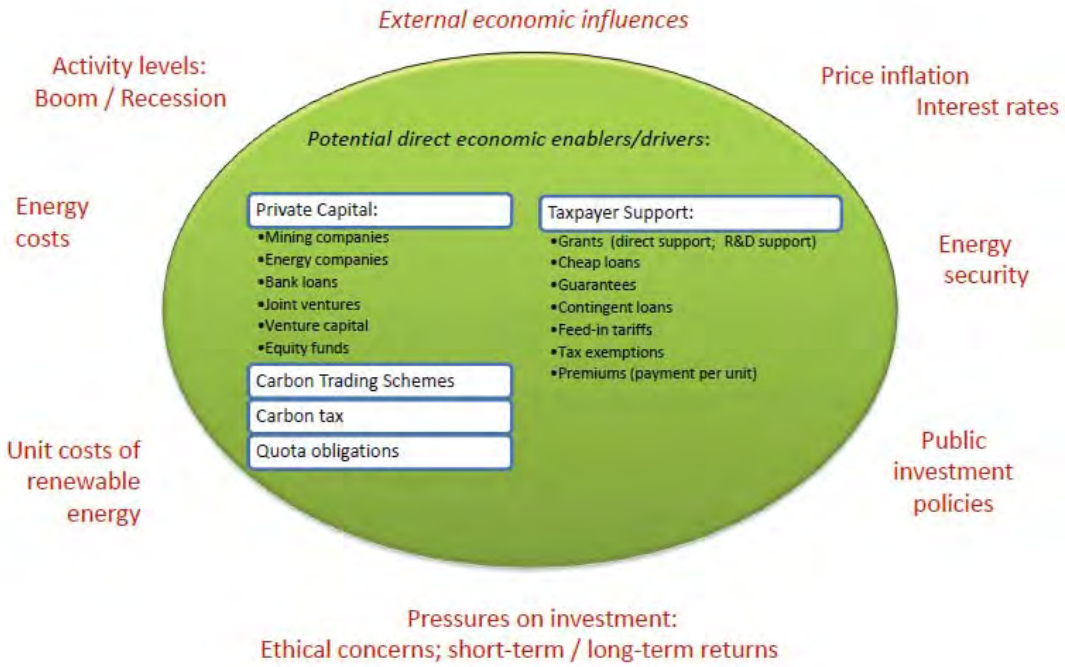
Thomson C, Marin P, Diez F V, Ordóñez S; Evaluation of the use of ceramic foams as catalyst supports for reverse-flow combustors; Chemical Engineering Journal (in press, doi: 10.1016/j.cej.2013.01.080)

Williams N C, Kennedy G A, Foster P J (2013); Electrical Demand Reduction & Response Strategies for Underground Mining; *Proceedings of the* 23rd World Mining Congress; ISBN: 978-1-926872-15-5

7. APPENDICES

Appendix 1 Economic & Financial Influences upon Low Carbon Initiatives

(Task 1.3)



Appendix 2 CH_4 Emissions from Hard Coal Exploitation in EU Countries 2001-2010

(Task 1.4)

Year	Number of CMM Coal Mines in the year	Hard coal output in the CMM Coal Mines	Methane emission from active Coal Mines				Methane emission from abandoned mines (AMM) (2006 IPCC code 1B1ai3 *Abandoned underground mines*: Includes methane emissions from abandoned underground mines.
			Mining (Emission Factor)		Post-mining Mine gas emissions (Emission Factor)		
			Ventilation methane emission (VAM) (2006 IPCC code 1B1ai1 *Mining:* Includes all Mine gas emissions vented to atmosphere from coal mine ventilation air)	The emissions from degassing systems (2006 IPCC code 1B1ai1 *Mining:* Includes all Mine gas emissions vented to atmosphere from coal mine degasification systems)	The emissions post -mining activities ((2006 IPCC code 1B1ai2 *Post-mining Mine gas emissions:* Includes methane emitted from post-mining activities)	The emissions from heaps of mine ((2006 IPCC code 1B1ai2 *Post-mining Mine gas emissions*: Includes methane emitted from heaps of mining wastes.)	
-		mln t	m^3 $tonne^{-1}$	m^3 $tonne^{-1}$	m^3 $tonne^{-1}$	m^3 $tonne^{-1}$	mln m^3 $year^{-1}$
BELGIUM (Hard coal output http://www.eia.doe.gov/iea/coal.html and Source: U.S. Energy Information Administration (July 2010)							
2001	NA	0.24	NA	NA	NA	NA	NA
2002	NA	0.19	NA	NA	NA	NA	NA
2003	NA	0.14	NA	NA	NA	NA	NA
2004	NA	0.20	NA	NA	NA	NA	NA
2005	NA	0.12	NA	NA	NA	NA	NA
2006	NA	0.03	NA	NA	NA	NA	NA
2007	NA	0	NA	NA	NA	NA	NA
2008	NA	0	NA	NA	NA	NA	NA
2009	NA	0	NA	NA	NA	NA	NA
2010	NA	NA	NA	NA	NA	NA	NA
CZECH REPUBLIC (Source: Mining yearbook of the Czech Republic 2001-2011 (in Czech), MONTANEX, a.s.,)							
2001	5	14.246	9.6	5.2	NA	-	NA
2002	5	14.200	9.1	5.7	NA	-	NA
2003	5	13.614	9.2	5.5	NA	-	32.7[1)]
2004	5	13.272	8.9	5.8	NA	-	32.4[1)]
2005	5	13.227	10.1	5.6	NA	-	NA
2006	5	14.280	10.1	5.4	NA	-	NA
2007	5	12.886	9.6	5.5	NA	-	NA
2008	5	12.662	10.3	5.4	NA	-	NA
2009	4	11.001	12.4	5.2	NA	-	NA
2010	4	11.584	10.7	5.4	NA	-	NA
[1)] dane pochodzą z kopalń, gdzie układy wydechowe zostały zainstalowane i są ogólnie dostępne (http://www.cbusbs.cz/docs/projekty/zprava023.pdf)							
FRANCE (Source: http://unfccc.int/national_reports/annex_i_ghg_inventories/national_inventories_submissions/items/5888.php)							
2001	NA	2.20	41.646	-	4.017	-	NA
2002	NA	1.63	51.046	-	4.017	-	NA
2003	NA	1.74	32.664	-	4.017	-	NA
2004	NA	0.16	32.664	-	4.017	-	NA
2005	NO	NO	NA	-	0	-	NA
2006	NO	NO	NA	-	0	-	NA
2007	NO	NO	NA	-	0	-	NA
2008	NO	NO	NA	-	0	-	NA
2009	NO	NO	NA	-	0	-	NA
2010	NO	NO	NA	-	0	-	NA
GERMANY (Source: http://unfccc.int/national_reports/annex_i_ghg_inventories/national_inventories_submissions/items/5888.php)							
2001	NA	27.361	19.856	NA	0.803	-	-
2002	NA	26.363	20.650	NA	0.803	-	-
2003	NA	25.873	19.368	NA	0.803	-	-
2004	NA	25.880	15.668	NA	0.803	-	-
2005	NA	24.907	14.250	NA	0.803	-	-

2006	NA	20.674	15.402	NA	0.803	-	-
2007	NA	21.307	11.480	NA	0.803	-	-
2008	NA	17.077	13.603	NA	0.803	-	-
2009	NA	13.766	12.342	NA	0.803	-	-
2010	NA	NA	NA	NA	NA	NA	NA
HUNGARY (Source: http://unfccc.int/national_reports/annex_i_ghg_inventories/national_inventories_submissions/items/5888.php)							
2001	NA	5.87	2.887	-	2.622	-	NA
2002	NA	5.45	3.236	-	0.335	-	NA
2003	NA	4.74	3.375	-	0.349	-	NA
2004	NA	2.77	2.552	-	0.265	-	NA
2005	NA	1.42	0.934	-	0.098	-	NA
2006	NA	1.49	0.865	-	0.084	-	NA
2007	NA	1.47	0.865	-	0.084	-	NA
2008	NA	1.36	0.865	-	0.084	-	NA
2009	NA	0.96	0.865	-	0.084	-	NA
2010	NA	NA	NA	NA	NA	NA	NA
ROMANIA (Source: http://unfccc.int/national_reports/annex_i_ghg_inventories/national_inventories_submissions/items/5888.php)							
2001	NA	8.65	16.36	-	2.287	-	NA
2002	NA	8.18	16.36	-	2.287	-	NA
2003	NA	7.98	16.36	-	2.287	-	NA
2004	NA	7.60	16.36	-	2.287	-	NA
2005	NA	7.36	16.36	-	2.287	-	NA
2006	NA	7.51	16.36	-	2.287	-	NA
2007	NA	7.73	16.36	-	2.287	-	NA
2008	NA	7.85	16.36	-	2.287	-	NA
2009	NA	6.96	16.36	-	2.287	-	NA
2010	NA	NA	NA	NA	NA	NA	NA
SLOVENIA (Source: CMV)							
2001	2	0.832	1.61	NA	NA	NA	NA
2002	2	0.776	1.53	NA	NA	NA	NA
2003	2	0.739	2.33	NA	NA	NA	NA
2004	2	0.742	3.10	NA	NA	NA	NA
2005	2	0.722	3.41	NA	NA	NA	NA
2006	2	0.714	2.50	NA	NA	NA	NA
2007	2	0.587	2.82	NA	NA	NA	NA
2008	2	0.594	1.82	NA	NA	NA	NA
2009	2	0.621	2.61	NA	NA	NA	NA
2010	2	NA	1.31	NA	NA	NA	NA
SPAIN (Source: AITEMIN)							
2001	9	2.41	38.2	NA	NA	NA	NA
2002	9	1.98	38.2	NA	NA	NA	NA
2003	9	2.16	38.2	NA	NA	NA	NA
2004	9	1.80	38.2	NA	NA	NA	NA
2005	7	1.43	22.5	NA	NA	NA	NA
2006	7	1.44	26.5	NA	NA	NA	NA
2007	7	1.04	26.5	NA	NA	NA	NA
2008	7	1.41	26.5	NA	NA	NA	NA
2009	7	1.32	26.5	NA	NA	NA	NA
2010	7	1.21	26.5	NA	NA	NA	NA
UK (Source: CSM / MRSL) N.B. One major working mine not reported for confidentiality reasons							
2001	NA	NA	NA	NA	NA	NA	61.2
2002	16	15.145	14.226	5.046	NA	NA	52.5
2003	14	14.743	12.879	3.811	NA	NA	35.7
2004	13	12.013	14.760	4.676	NA	NA	24.2
2005	16	8.823	15.456	2.778	NA	NA	28.2
2006	8	8.910	12.779	2.422	NA	NA	38.5
2007	5	6.425	9.184	1.791	NA	NA	34.3
2008	5	7.167	11.373	2.200	NA	NA	31.0
2009	5	6.751	12.374	2.199	NA	NA	23.4
2010	5	6.826	11.809	1.310	NA	NA	23.7

POLAND (Source: GIG)							
2001	30	72.366	7.010	0.990	0.590	0.026	0.91
2002	30	72.129	7.284	0.996	0.590	0.013	0.73
2003	29	65.708	8.457	1.353	0.606	0.011	-
2004	29	69.167	7.640	1.281	0.601	0.017	-
2005	24	67.347	8.075	1.500	0.601	0.018	-
2006	24	64.518	8.332	1.966	0.577	0.013	-
2007	23	62.465	9.427	1.637	0.586	0.021	-
2008	23	57.537	10.288	2.137	0.586	0.012	-
2009	23	53.271	11.150	1.815	0.586	0.032	-
2010	21	52.184	11.050	1.777	0.579	0.013	-

Appendix 3 Polish methodology for estimation of Methane Emissions to meet IPCC Guidelines 2006

(Task 2.1)

In Poland, the National Greenhouse Inventory 2007[3] was published to meet Poland's obligations resulting from point 3.1 Decision no. 280/2004/WE of the European Parliament and of the Council of 11 February 2004 concerning a mechanism for monitoring EC greenhouse gas emissions. As insufficient data existed concerning methane emissions from Polish collieries, this led to the detailed study of methane emissions from mines in Gornoslaskie Zaglebie Weglowe – GZW (Upper Silesian Coal Basin - USCB) (LOWCARB Task 1.4). The methodology applied to the methane emission estimation (Task 1.4) from GZW covered the four basic emission sources, based upon IPCC[4] guidelines of 2006 and the amended 2010 version (IPCC 2006[5]). Emissions estimated with the national (Polish) method have been compared with the results obtained with the use of equations from the IPCC[6] 2006 method. There follows a description of the methodology for estimating methane emissions in Polish coal mines (active and abandoned) in the years 2001-2010, considering:

- Methane emission during the mining process as ventilation emission and drainage systems emission,
- Post-mining methane emission and surface methane emission (from landfills).

Estimation of ventilation emission – Ventilation emission E_w (m^3) was calculated for the given coal mines on the basis of the equation:

$$E_w = W_e * Q_w$$

where: Q_w – hard coal output (tonnes),

W_e – ventilation emission factor (m^3 CH_4 /tonne of coal).

The amount of methane from the source (E_w) was estimated on the basis of data provided by the hard coal mines and collected in the *Annual Report* [7] (2002-2011). The final value of ventilation emission of methane was converted to standard conditions (293°K) with the equation:

$$E_{w(N)} = E_w * 293 / (273 + t_p)$$

where: $E_{w(N)}$ – ventilation emission converted into standard conditions (20 °C), [M m^3],

E_w – ventilation emission [M m^3],

$t_p = 30$ – annual average temperature of ventilation air [°C].

The emission factor expresses the volume of methane released to the atmosphere per weight unit (tonne) of extracted coal, which is then converted to a weight unit with the use of conversion coefficient of 0.67 (tonne/M m^3) (Revised[8], 1996).

Assessment of methane emission from degassing systems –The volume of methane from the source is calculated on the basis of data provided by individual

---

[3] Krajowa inwentaryzacja emisji i pochłaniania gazów cieplarnianych za rok 2007. Raport wykonany na potrzeby Ramowej konwencji Narodów Zjednoczonych w sprawie zmian klimatu oraz Protokołu z Kioto. KRAJOWY ADMINISTRATOR SYSTEMU HANDLU UPRAWNIENIAMI DO EMISJI. KRAJOWE CENTRUM INWENTARYZACJI EMISJI. Instytut Ochrony Środowiska. Warszawa Maj 2009.

[4] IPCC (2006). Guidelines for National Greenhouse Gas Inventories. 2006

[5] IPCC (2006). Guidelines for National Greenhouse Gas Inventories. 2010

[6] IPCC (1997). Revised 1996 IPCC Guidelines for National Greenhouse Gas Inventories. Reference Manual. IPCC 1997; IPCC (2000). 2000 IPCC Good Practice Guidance and Uncertainty Management in National GHG Inventories; IPCC (2006). 2006 IPCC Guidelines for National Greenhouse Gas Inventories; Revised 1996 IPCC Guidelines for National Greenhouse Gas Inventories: Reporting Manual. IPCC/OECD joint Programme

[7] *Annual Report* (for the years 2001-2010) on the State of Basin natural and Technical Hazards in the Hard Coal Mining Industry. Central Mining Institute, Katowice 2002-2011

[8] Revised 1996 IPCC Guidelines for National Greenhouse Gas Inventories: Reporting Manual. IPCC/OECD joint Programme

coal mines and is presented in the *Annual Report* [9] (2002-2011). Emission from degassing systems (E_o), where degassing is conducted, is calculated as the difference between the volume of methane recovered (M_{ui}) and used (M_{wi}) or burned, according to the equation:

$$E_o = M_{ui} - M_{wi} \text{ [M m}^3 \text{ CH}_4\text{]}$$

The final value is given after converting the methane emission values indicated by the coal mines to standard conditions (293°K).

Although IPCC guidelines do not estimate *post-mining methane emission* with a specific method (a national average approach is advised), post-mining methane emission and surface methane emission (from industrial wastes), between 2001-2010 in Poland, were calculated on the basis of the data taken from *Annual Report*[10] (2002-2011) and *Rocznik Statystyczny (Statistical Yearbook)*[11] (2002-2010) in *Environmental Protection* [12](2011), with the use of a specific method for GZW. In the calculations of surface methane, emission data concerning the total post-mining waste output in hard coal mines in a given year were used.

Estimation of post-mining methane emission – Post-mining methane emission from the extracted coal is estimated separately for each of the coal mines in each of the years analysed, with the use of a national methodology which assumes that on the surface only residual methane is emitted, whereas the value of residual methane content in given coal mines was estimated on the basis of the data from the work of Kwarcinski[13] (2005). Depending on the relationship between the average methane content of a given seam of coal (G_{pk}, m^3/t clean coal) and the value of the residual methane content (G_r, m^3/tonne clean coal) the following was assumed:

– In seams (or parts of them), where the average methane content of a given seam of coal (G_{pk}, m^3/tonne clean coal) is higher than or equal to the residual methane content (G_r, m^3/tonne clean coal), the volume of surface methane emission (e_{pko}, m^3) is calculated according to the equation:

$$e_{pko} = M_w * (100 – W – A)/100 * G_r$$

where:
M_w - extracted coal mass (Mt)
W - average total moisture (%)
A - average ash content (%)

– In seams (parts of seams), where average methane content in coal (G_{pk}, m^3/tonne clean coal) is lower than residual methane content (G_r, m^3/tonne clean coal), the volume of surface methane emission (e_{pkl}, m^3) was calculated with the equation:

$$e_{pkl} = M_w * (100 – W – A)/100 * G_{pk}$$

where:
M_w - extracted coal mass (Mt)
W - average total moisture (%)
A - average ash content (%)

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[9] *Annual Report* (for the years 2001-2010) on the State of Basin natural and Technical Hazards in the Hard Coal Mining Industry. Central Mining Institute, Katowice 2002-2011

[10] *Annual Report* (for the years 2001-2010) on the State of Basin natural and Technical Hazards in the Hard Coal Mining Industry. Central Mining Institute, Katowice 2002-2011

[11] Rocznik Statystyczny Wojewodztwa Slaskiego (2001-2010). Urzad Statystyczny w Katowicach, Katowice

[12] Environmental Protection in the Slaskie Voivodship in Years 2007-2010. Statistical Office in Katowice. Katowice 2011

[13] Kwarciński J. (2005). Ocena rzeczywistej emisji metanu do atmosfery spowodowanej eksploatacją węgla kamiennego. Państwowy Instytut Geologiczny, Sosnowiec, 2005.

Post-mining methane emission was estimated on the basis of an empirically obtained relationship between coal sample methane emission and average methane content of the exploited seam. This assumes that on one hand it is possible to extract coal which still contains dynamically desorbable methane, while on the other hand extensive degassing of coal (below residual methane content) takes place underground, and partial emission of residual methane on the surface. The volume of methane emission from the analysed seam (e_{pkn}) is:

$$e_{pkn} = M_w * (100 - W - A)/100 * (0.2144177 * G_{pk})$$

The final estimated volume of methane emission from an exploited coal seam/part of a seam (e_{pk}) is defined by an arithmetic mean of the two results.

$$e_{pk} = (e_{pko} + e_{pkn}) / 2$$

Methane emission in the whole coal mine (E_{pk}) is a total of emission in individual coal seams /parts of seams. Post-mining emission factors (W_{ep}) for individual coal mines were calculated with the equation: $W_{ep} = \Sigma [e_{pk} / Q_w]$

Estimation of industrial waste site methane emission: This was conducted taking into consideration the annual amount of production wastes from coal exploitation and coal preparation processes and assuming a methane content in the organic substance and average content of coal matter in industrial waste of 15% (after Kwarcinski[14], 2005). Methane contained in pores (or fissures in waste rock) in the form of free gas and most of sorbed methane is released underground and becomes a part of ventilation emission.

The formula for estimating the volume of methane emitted from industrial waste (E_{eso}) in coal mines is:

$$E_{eso} = S_o * B * W_{ep}$$

where: E_{eso} - amount (volume) of methane emitted from industrial waste (M m^3)

S_o - amount of produced mineral wastes (M t/year),

B - share of coal matter (coal) in the waste (% of weight),

W_{ep} - post-mining emission factor (m^3/tonnes csw)

Methane emission measurement and analysis also took place at an abandoned coal mine (KWK Niwka Modrzejow Sp. z o.o.). The volume and share of methane emission from this mine were included in the collective analyses from the years 2001-2002. Both input data (coal production, methane content in individual coal mines) and the data on the amount of post-mining industrial waste in individual years, come from credible sources (i.e. *Annual Report*...[15] 2002-2011).

[14] Kwarcinski J. (2005). Ocena rzeczywistej emisji metanu do atmosfery spowodowanej eksploatacją węgla kamiennego. Panstwowy Instytut Geologiczny, Sosnowiec, 2005.

[15] *Annual Report* (for the years 2001-2010) on the State of Basin natural and Technical Hazards in the Hard Coal Mining Industry. Central Mining Institute, Katowice 2002-2011

Appendix 4 Desorbable CH_4 reserves

Predicted Methane Contents and Desorbable Reserves

Seams 405/1 and 405/3, Knurów-Szczygłowice (Task 2.1)

Methane contents predicted for seams 405/1 and 405/3 – areas to be extracted to 2022

Ser. No.	Longwall No.	Seam	Extraction time period	Length, m	Degassing range, m		Methane content, m^3CH_4 / Mg csw	
					above:	below:	primary:	secondary:
1.	31	405/1	2012-2013	250	161	61		2.398
2.	XIV	405/1	2012-2013	200	134	50	7.267	
3.	32	405/1	2014-2015	250	161	61		4.891
4.	XV	405/1	2014-2017	205	134	50	7.267	
5.	28	405/1	2018-2020	250	161	61	5.866	
6.	33	405/1	2020-2022	250	161	61	7.318	
7.	6	405/3	2012-2013	245	159	60		0.077
8.	XXV	405/3	2013-2015	210	140	52	7.647	
9.	8	405/3	2014-2015	250	161	61		3.258
10.	31	405/3	2017-2019	250	161	61	7.873	

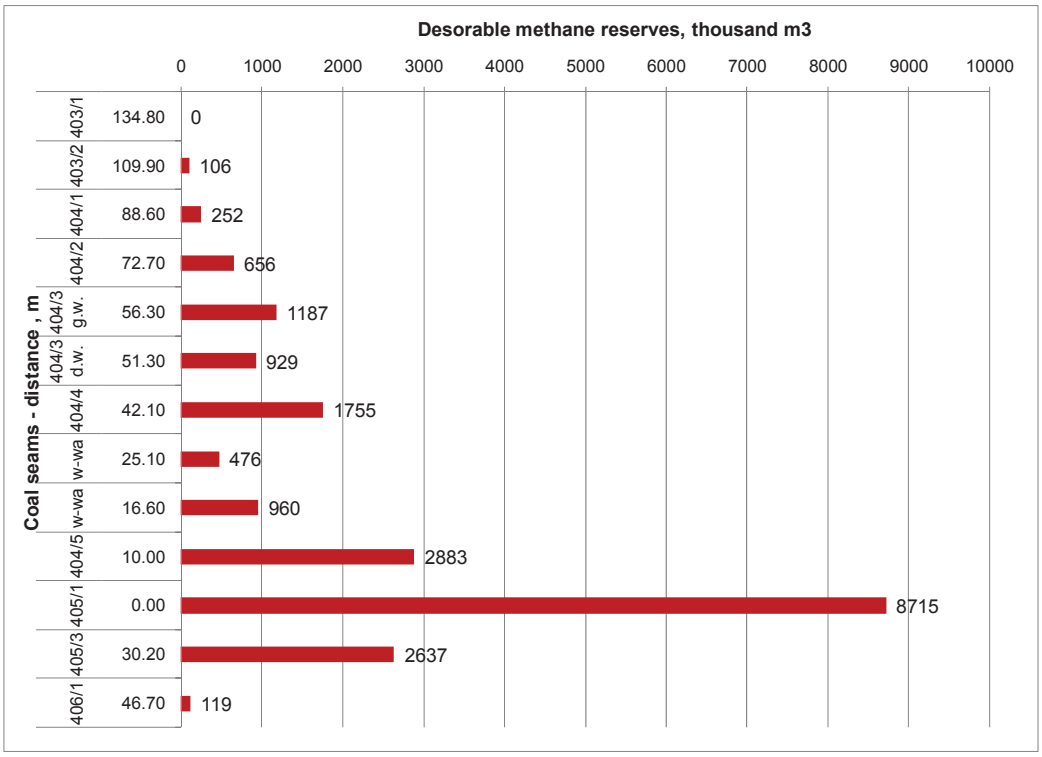
Desorbable methane reserves in seams 405/1 and 405/3

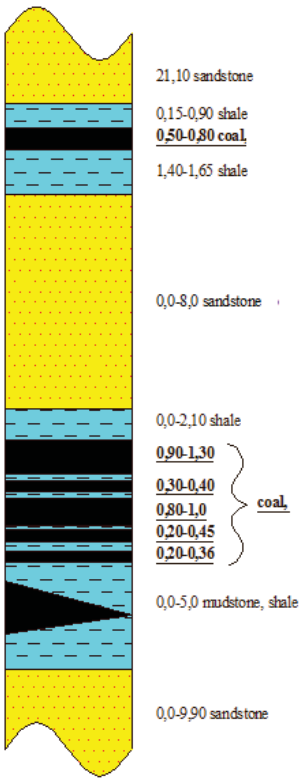
Ser. No.	Longwall No.	Seam	Extraction time period	Longwall length	Panel length	Methane reserves of seam to be extracted	Desorbable methane reserves in zone of desorption
				m	m	thousand m^3	thousand m^3
1.	31	405/1	2012-2013	250	1000	543	2259
2.	XIV	405/1	2012-2013	200	2135	8715	20697
3.	32	405/1	2014-2015	250	1250	1935	6135
4.	XV	405/1	2014-2017	205	2175	9053	21861
5.	28	405/1	2018-2020	250	1850	4132	13198
6.	33	405/1	2020-2022	250	1500	4998	14394
7.	6	405/3	2012-2013	245	1000	7	1121
8.	XXV	405/3	2013-2015	210	1490	7347	15067
9.	8	405/3	2014-2015	250	760	1095	2354
10.	31	405/3	2017-2019	250	1650	9423	23205

Continued....

Desorbable CH_4 reserves

Longwall XIV, seam 405/1, Knurów-Szczygłowice (Task 2.1)





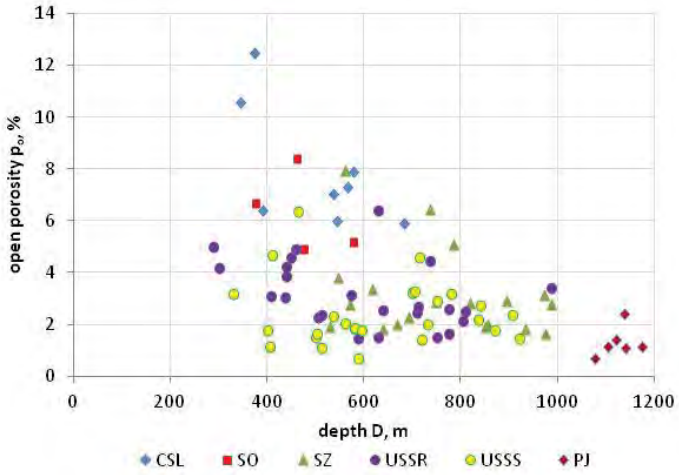
Example of geological cross-section

Appendix 5 Relationships between open porosity, UCS and depth

(Task 2.2)

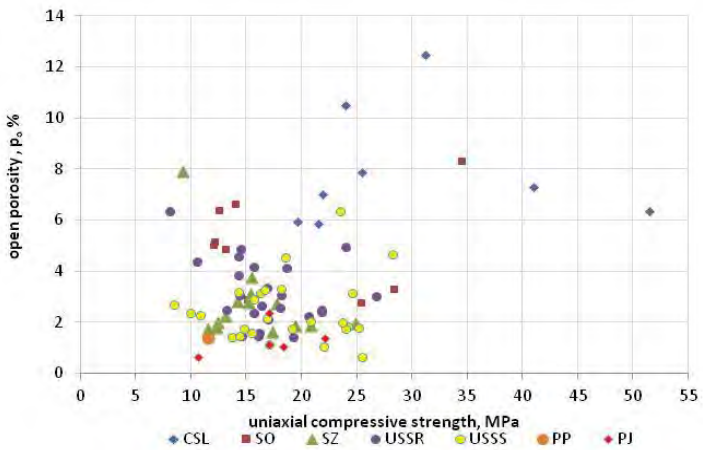
Coal open porosity and compressive strength

CSL - Cracow Sandstone Series (Laziskie Beds); SO - Siltstone Series (Orzeskie Beds); SZ - Siltstone Series (Zaleskie Beds); USSR - Upper Silesian Sandstone Series (Rudzkie Beds); USSS - Upper Silesian Sandstone Series (Siodlowe Beds); PP - Paralic Series (Porebskie Beds); PJ - Paralic Series (Jaklowieckie Beds)



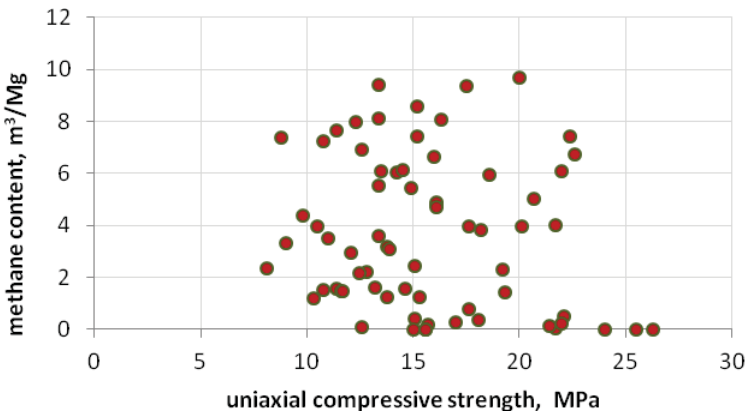
Coal open porosity and depth

CSL - Cracow Sandstone Series (Laziskie Beds); SO - Siltstone Series (Orzeskie Beds); SZ - Siltstone Series (Zaleskie Beds); USSR - Upper Silesian Sandstone Series (Rudzkie Beds); USSS - Upper Silesian Sandstone Series (Siodlowe Beds); PP - Paralic Series (Porebskie Beds); PJ - Paralic Series (Jaklowieckie Beds)

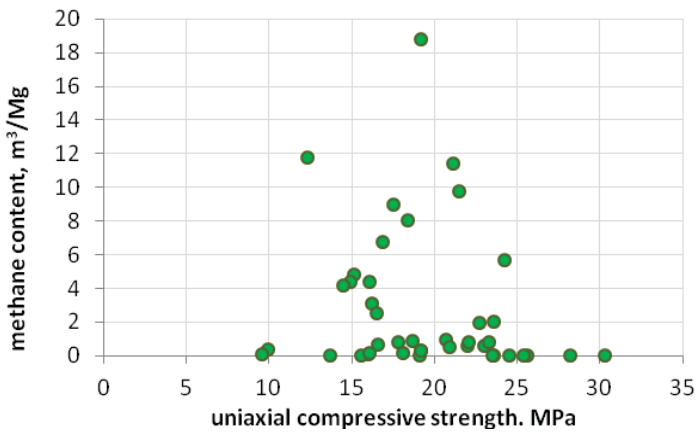


Appendix 6 Relationship of UCS with methane content

(Task 2.2)



Uniaxial compressive strength - methane content, Rudzkie beds
From the Upper Silesian Sandstone Series, Rudzkie Beds



Uniaxial compressive strength - methane content, Siodlowe beds
From the Upper Silesian Sandstone Series, Siodlowe Beds

Appendix 7 Longwall pre-mining drainage modelling

(Task 2.3)

Parameter	Value	UNIT
Coal seam thickness	4.0	m
Depth	750	m
Surface area	196 938	m^2
Horizontal resolution of grid	2x2	m
Vertical resolution of grid	2	m
Number of cells	64815	-
Porosity	6.3	%
Permeability	0.5 – 5.0	mD
Initial pressure	3 - 40	bar
Pressure in the galleries	1	bar
Temperature	34	°C
Coal density	1386	kg/m^3
Moisture content	7.5	%
Ash content	3.2	%
Extended Langmuir isotherm:		
Methane volume V_L	0.007	sm^3/kg
Methane pressure P_L	3.0	bar

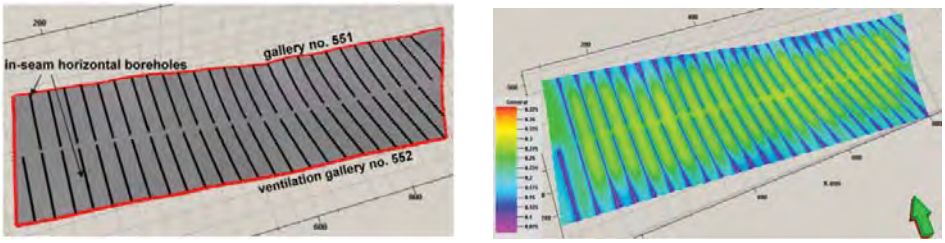
Modelling parameters and coal properties, longwall 121 seam 364 Brzeszcze colliery

Initial parameters										
	Initial average pressure [bar]	Pressure in boreholes [bar]	Operation time of boreholes	Diameter of boreholes [mm]	Length of boreholes [m]	Total gas in place [10^6 x sm^3]	Free gas in coal fractures [10^6 x sm^3]	Volume of methane per ton of coal [m^3/t]	Methane production [10^6 x sm^3]	Efficiency of methane drainage [%]
-	28.11	-	-	-	-	3.704	0.538	7.37	-	-
Post methane drainage parameters										
Case	Post drainage average pressure [bar]	Pressure in boreholes [bar]	Operation time of boreholes	Diameter of boreholes [mm]	Length of boreholes [m]	Total gas in place [10^6 x sm3]	Free gas in coal fractures [sm^3]	Volume of methane per ton of coal [m^3/t]	Methane production [10^6 x sm^3]	Efficiency of methane drainage [%]
1	3.98	0.5	1 year	76	120	2.007	0.091	3.99	1.697	45.8
2	11.71	0.5	1 year	76	120	2.790	0.273	5.55	0.913	24.7
3	19.49	0.5	1 year	76	100	3.245	0.461	6.46	0.458	12.4
4	19.88	0.5	5 months and then gradual disabling	76	100	3.325	0.470	6.62	0.379	10.2
5	18.22	1	842 days	76	100	3.186	0.430	6.34	0.518	14.0

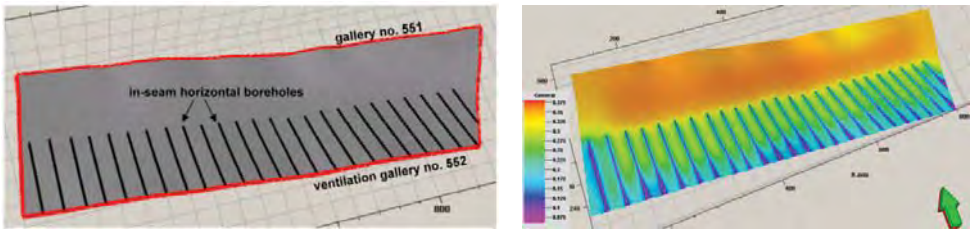
Initial and final parameters for various drainage simulation options

Appendix 8 Images from pre-mining methane drainage simulation options

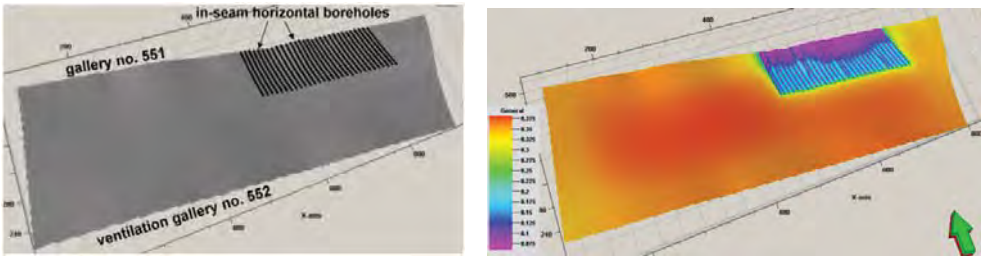
(Task 2.3)



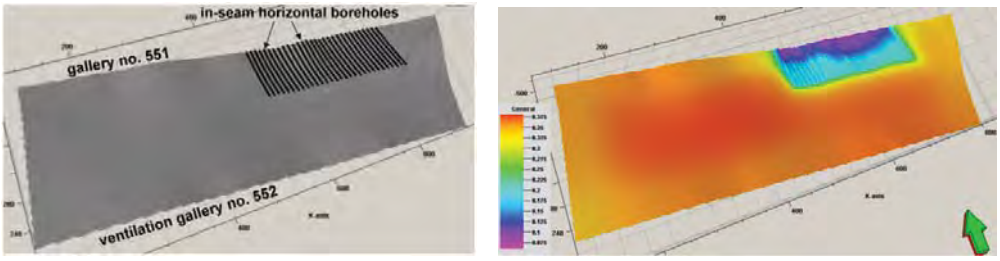
Simulation of Case 1 borehole pattern (left) & CH_4 final concentration (right)



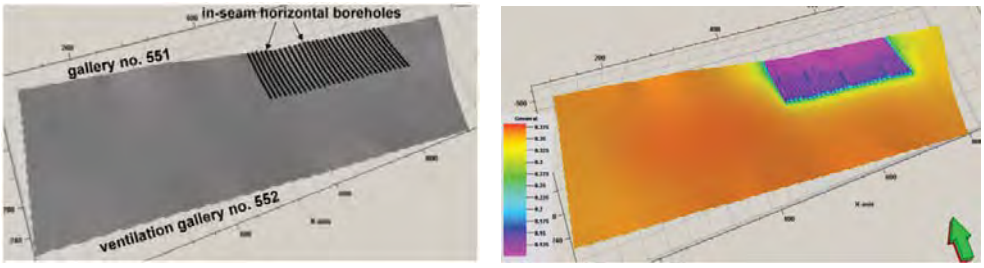
Simulation of Case 2 borehole pattern (left) & CH_4 final concentration (right)



Simulation of Case 3 borehole pattern (left) & CH_4 final concentration (right)



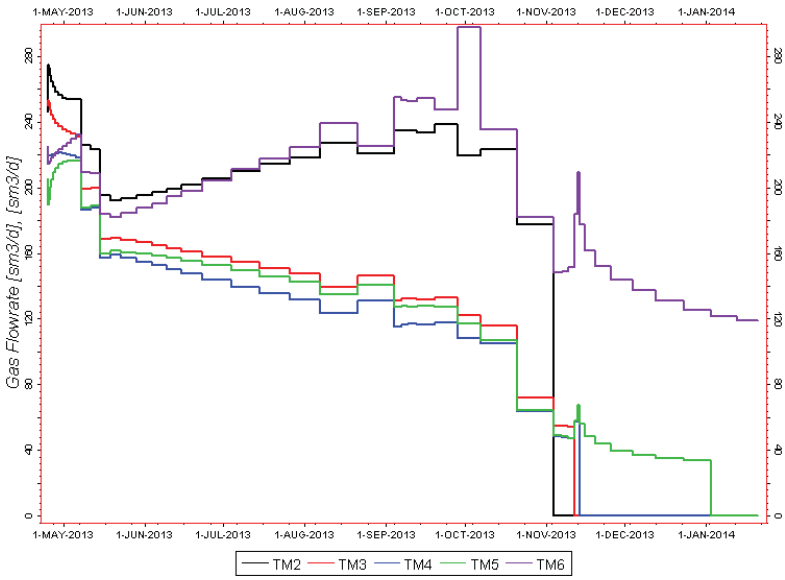
Simulation of Case 4 borehole pattern (left) & CH_4 final concentration (right)



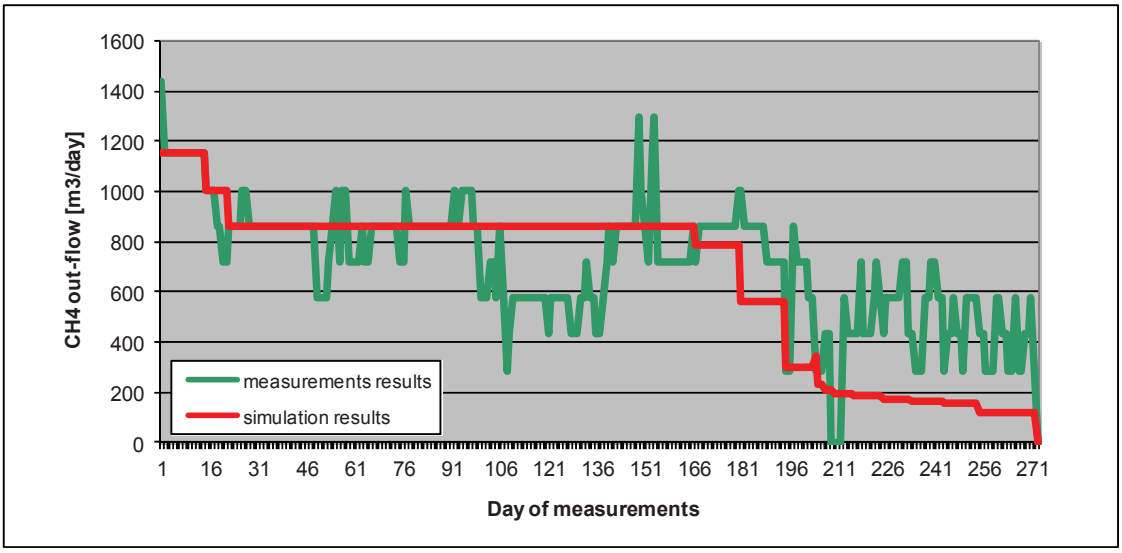
Simulation of Case 5 borehole pattern (left) & CH_4 final concentration (right)

Appendix 9 Measured methane flow from boreholes & comparison with simulation

(Task 2.3)



Methane flow rate for boreholes TM2-TM6 m^3/day



Boreholes TM2-TM6; field measurements vs simulation results

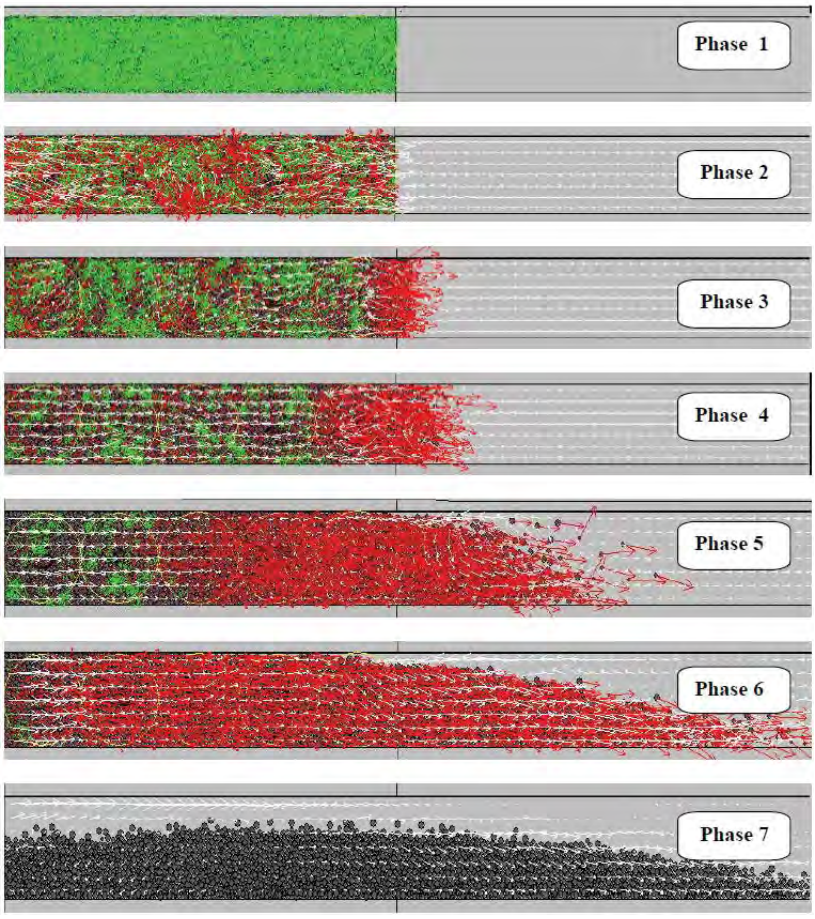
CH4 concentration, out-flow, pressure in borehole and negative pressure were measured to verify the numerical model. CH_4 out-flow rates of the simulation were matched to the field rates. In order to match methane out-flow rates the following input assumptions and parameters of model were modified:

- permeability distribution: permeability was assumed to be 5 mD (maximum result of laboratory test was 5,8 mD),
- porosity distribution: porosity was assumed to be 9 % (maximum result of laboratory test was 11,33 %),
- pressure in boreholes, negative pressure and diffusion coefficient were updated,
- methane out-flow at the initial stage of production was limited.

These changes covered either the whole model area or local regions of the model.

Appendix 10 Outburst modelling

(Task 2.3)



Consecutive phases of coal and methane outburst in the PFC model

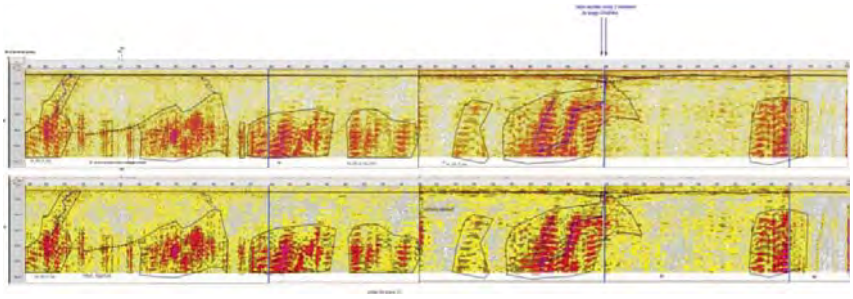
Coal seam	Methane content (Mc), m^3 /Mg of coal	Original porosity of coal (Op), percent	Compressive strength of coal (Rc), MPa
Highly outburst prone (Hop)	> 10 (10)	> 10 (8)	<10 (6)
Medium outburst prone (Mop)	5 - 10 (6)	7-10 (4)	10 - 20 (3)
Low outburst prone (Lop)	< 5 (2)	< 7 (1)	> 20 (1)

Highly outburst prone seam 24 points;
medium outburst prone seam 13 points; low outburst prone seam 4 points.

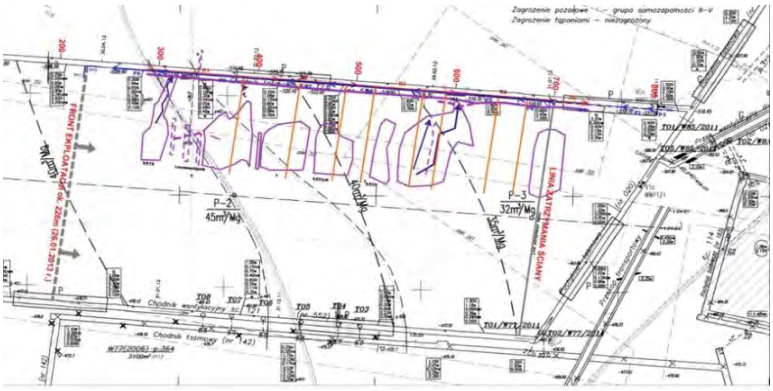
Classification of coal and methane bursting tendency of coal seams

Appendix 11 Geophysical experiments for in-seam methane detection

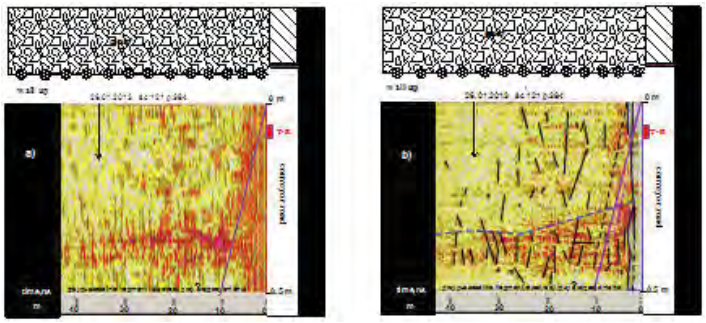
(Task 2.3)



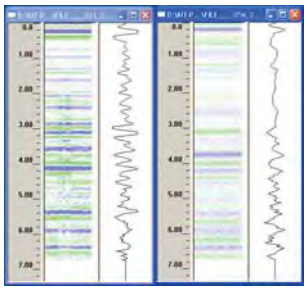
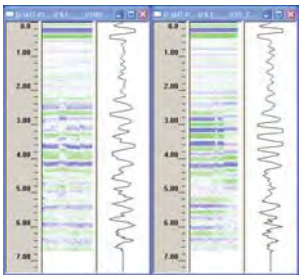
Radar sections from conveyor road side at 850 ns time window



Anomalous zones at longwall 121 seam 364



Radarograms of KWSA wall face: a) raw data; b) processed by HBG

Profiles 11 (before)/56 (post) 200 MHz antenna	Profiles 10 (before)/55(post) 200 MHz antenna
	

Montsacro profiles before and after degasification

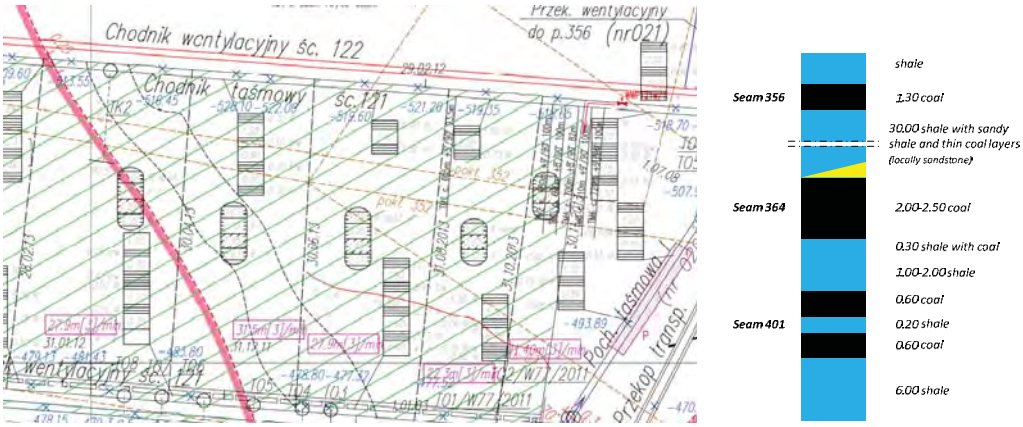
Appendix 12 Pre-mining methane drainage

(Task 2.4)

No.	Longwall	Seam	Star of extraction (mm/yyyy)	End of extraction (mm/yyyy)	Methane content m^3 CH_4/t	Methane drainage (yes/no)
SZCZYGŁOWICE Colliery						
1	XIV	405/1	01/2012	03/2014	2.2÷7.9	yes
2	XXII	401/1	04/2012	07/2013	0.9÷3.5	yes
3	XII	407/3	02/2011	04/2012	2.8÷7.6	yes
BRZESZCZE Colliery						
1	193	510	2011-11	2013-05	5.1 – 16.1	yes
2	194	510	2013-08	2014-02	5.3 – 9.6	yes
3	128	401	2012-01	2012-09	4.4 – 5.6	yes
4	812	352	2012-04	2013-02	6.0 – 10.0	yes
5	123	364	2011-08	2012-01	2.4 – 7.8	yes
6	121	364	2012-11	2013-09	2.4 – 8.0	yes

Potential sites for pre-mining methane drainage

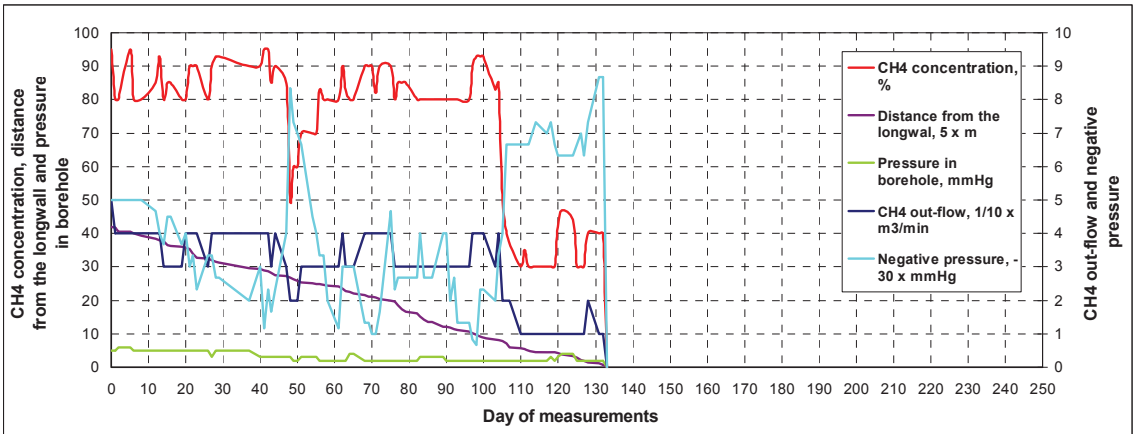
Initial selection: Longwall 193 in seam 510 Brzeszcze; map & profile



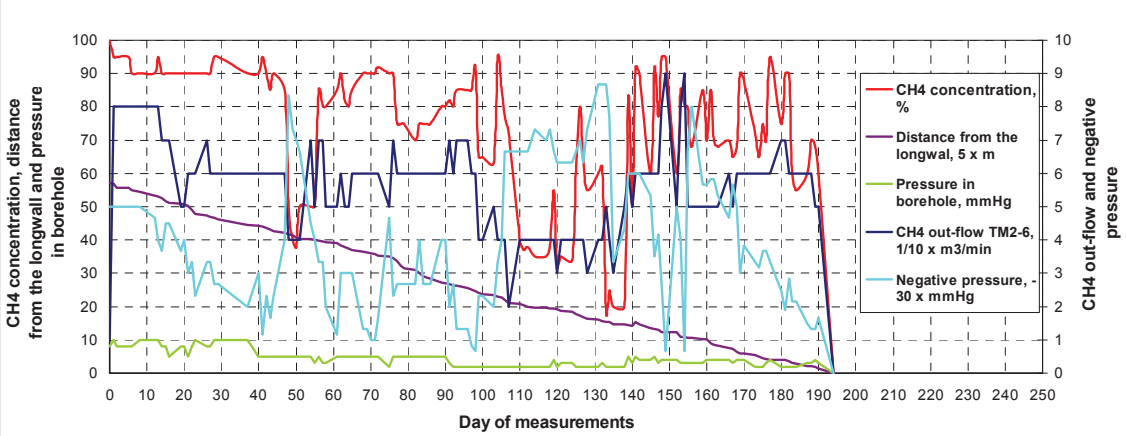
Alternative selection following fire: Longwall 121 in seam 364

Appendix 13 Results from methane borehole daily measurements

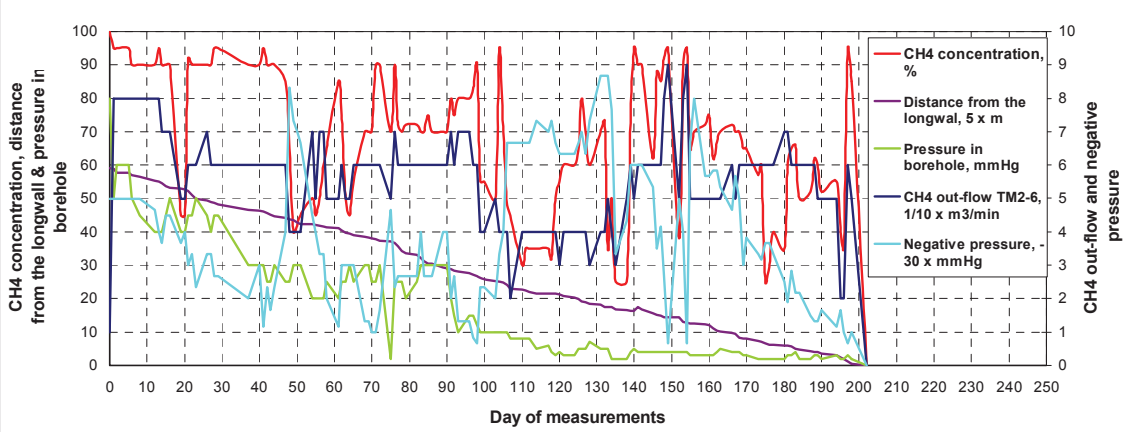
(Task 2.4)



Methane drainage borehole TM1: mesurement results

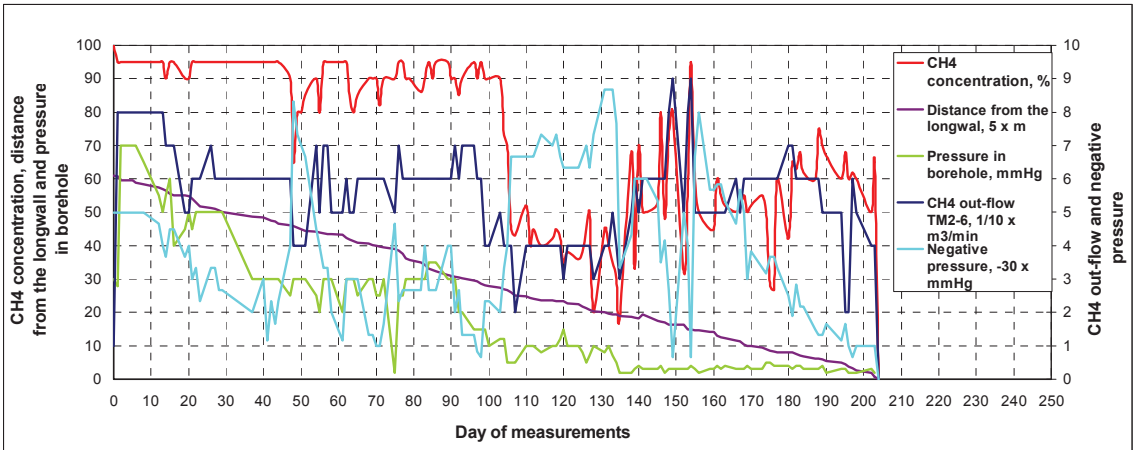


Methane drainage borehole TM2: mesurement results

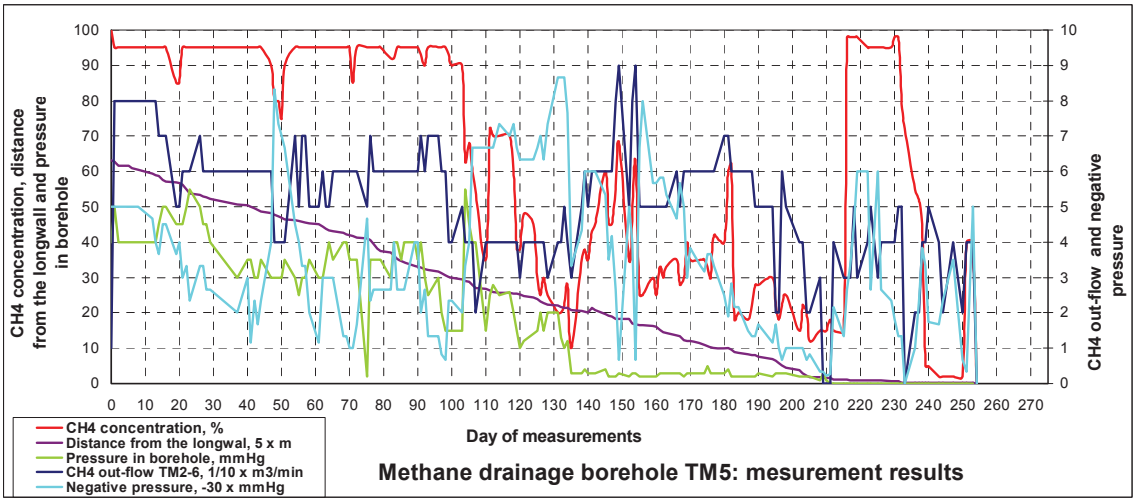


Methane drainage borehole TM3: mesurement results

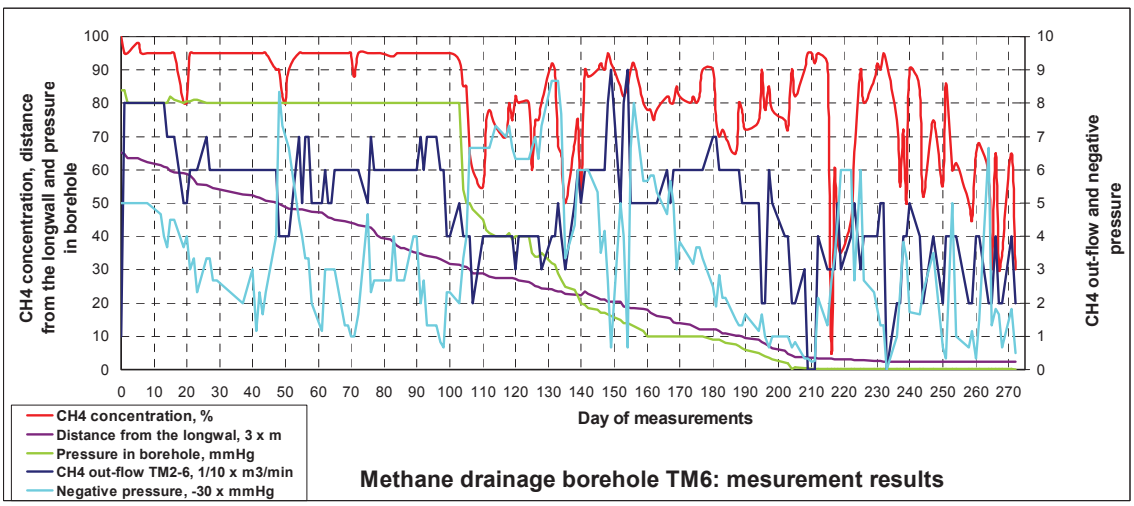
Continued....



Methane drainage borehole TM4: mesurement results



Methane drainage borehole TM5: mesurement results



Methane drainage borehole TM6: mesurement results

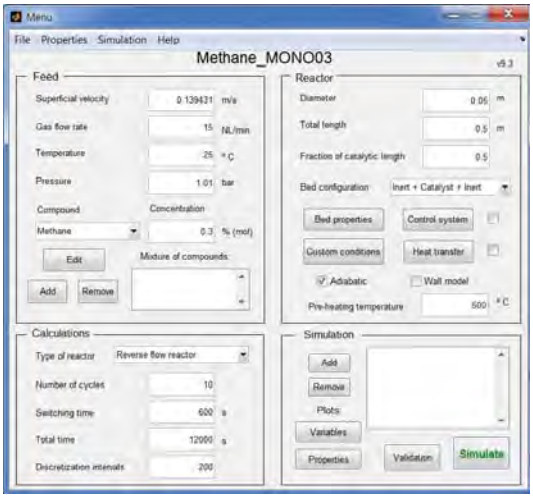
Appendix 14 Heat load survey

(Task 3.1)

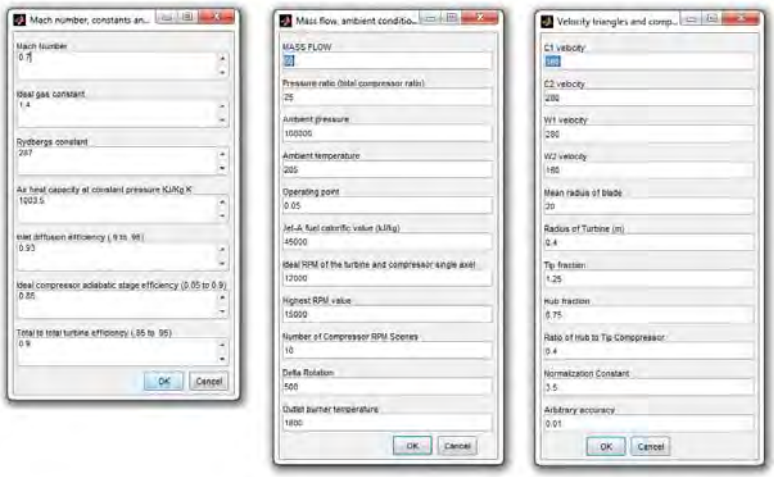
HEAT LOAD INFORMATION AT UNDERGROUND COAL MINES IN EUROPE	
N.B. Underground coal mining now only takes place in the European Union in Poland, UK, Germany, Spain, lignite in Czech Republic, Slovenia and small amounts in Bulgaria and Slovakia. Further mine closures are anticipated and Poland presents by far the most robust industry, although some decline is also likely there due to the more accessible reserves dwindling.	
Czech Rep	No data available.
Germany	Hot mines – in 1980s ~300MW of cooling power was in use. No recent data to date on reduced industry. All mines scheduled for closure.
Poland	Surface air temp (Katowice) typically 25°C summer : -18°C winter. Mining stops at occasional extremes (33°C underground : -20°C surface). A mining group working 53 longwalls over 15 mines (deepest at 1,200m) provides around 40MW of electrical power to chiller units. Also heating when required by hot water from adjacent coal power stations. Another mining group with 6 mines runs gas engines running on drained mine gas for mine energy, providing a total of 24MWel, as well as 1.4MWth in water boilers. At one mine, an especially well-developed tri-generation system (part of the above) provided 57MWh of heat and 27MWh of cooling in a year (2009) in addition to electricity. (It is not known to what extent the loads at this group may require other energy sources for additional heat / cooling.)
Slovenia	To deal with cold winters, a very large mine has reported 11.5MW heating units installed working on hot water from an adjacent coal power station. Around 15,000MWh per annum consumed in heating downcast air in winter; approx 6,000MWh for surface facilities heating and showers.
Spain	Report of no active heating or cooling from a major mining company.
UK	Minor loads: Occasional use of propane heaters in shafts in winter to prevent water pipes freezing; chillers on closed loop to cool conveyor motors; water boilers heated by mine methane (as well as electricity generation from gas engines). A proposed industrial estate next to a large UK mine, as yet still undeveloped, might offer the prospect for co-ordinated planning of user industries, although methane concentrations may not be sufficiently high for good quality heat provision.

Appendix 15 MATLAB Models of Flow Reversal Reactor and Gas Turbine

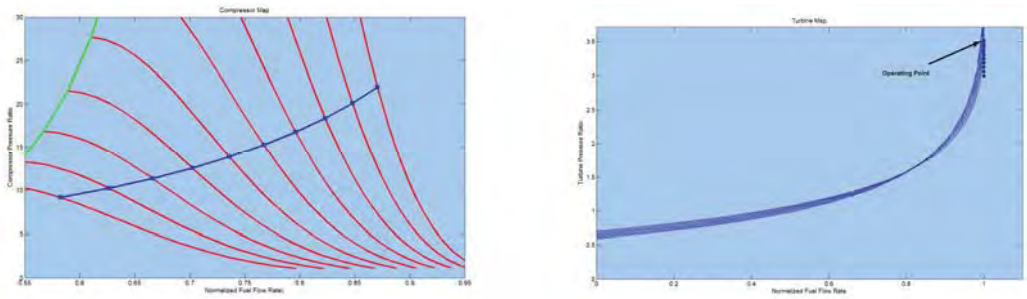
(Task 3.1)



Flow reversal reactor main data entry screen



Gas turbine data entry screens

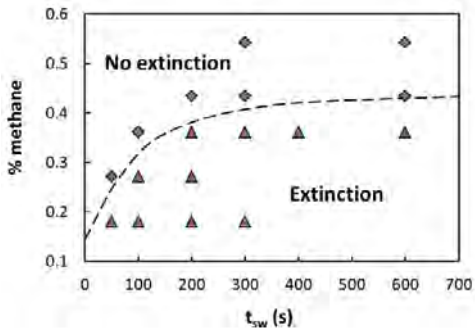


Gas Turbine Compressor and Turbine maps from the settings above

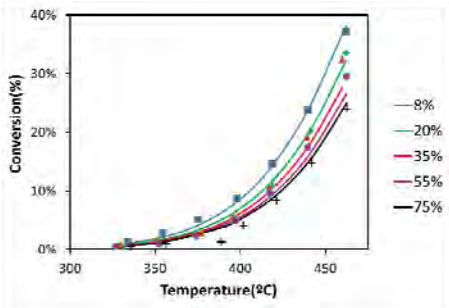
(x – Normalised fuel flow rate; y – pressure ratio)

Appendix 16 Catalytic Flow Reversal Reactor results

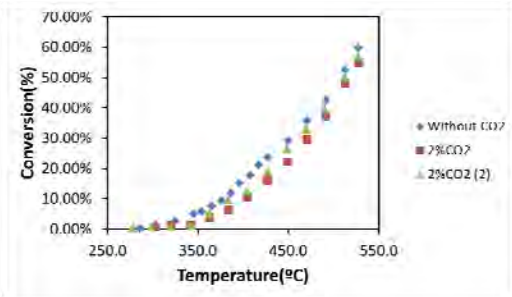
(Task 3.1; 6.1)



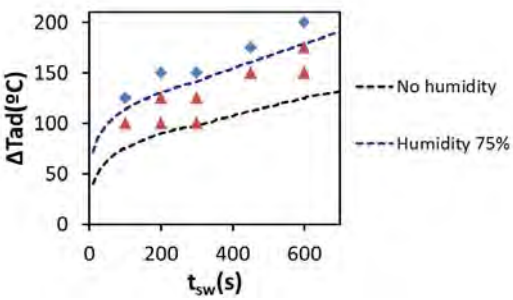
Stability region of a foam bed CFRR – CH_4 concentrations / switching times



CFRR light-off curves for different relative humidities (isothermal conditions)



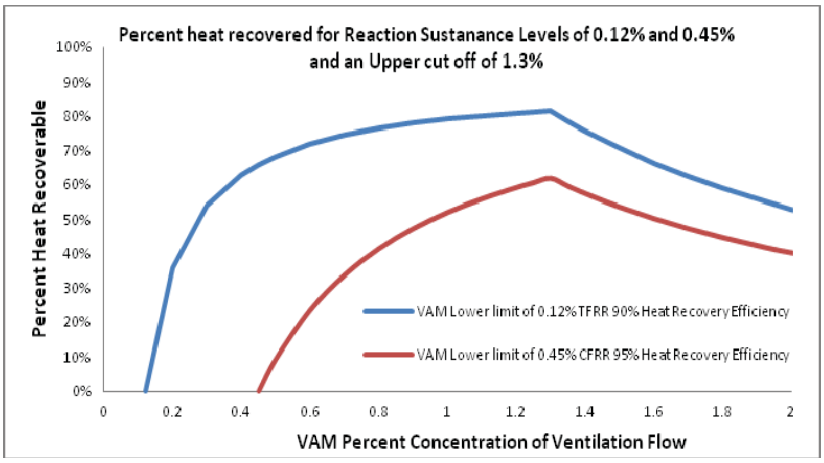
Effect of CO_2 on CFRR reaction



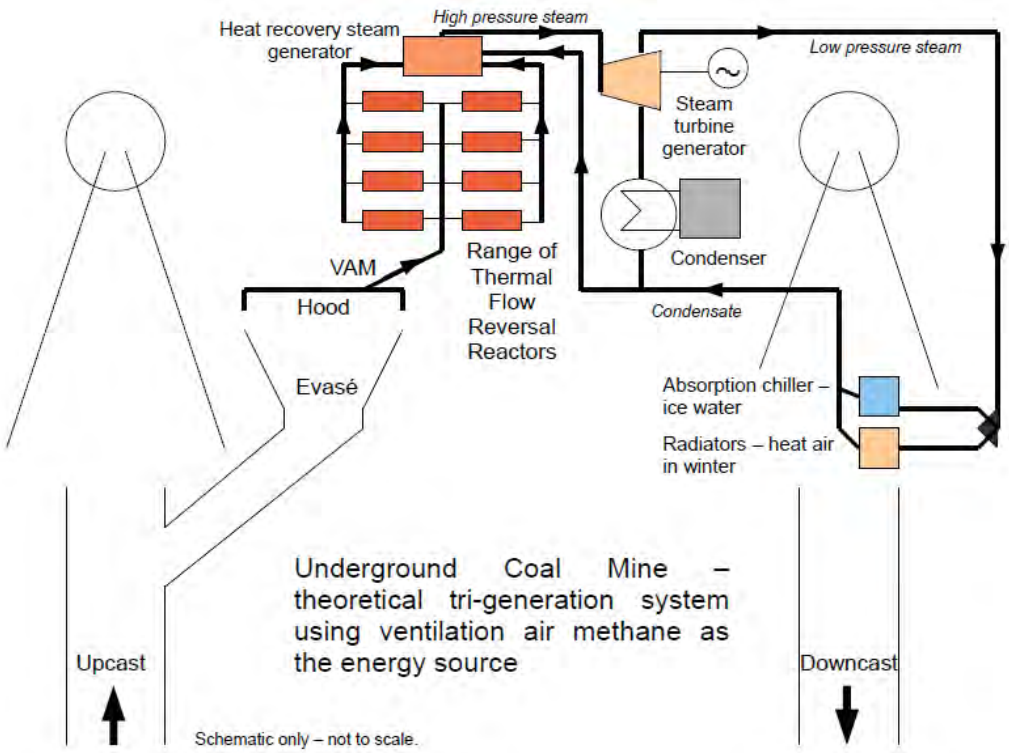
Comparison of CFRR reaction extinction with and without humidity

Appendix 17 Tri-generation from ventilation air methane

(Task 3.3)



Recoverable heat energy – TFRR sufficient for electricity at >0.45% VAM (<10% of mines)



Diagrammatic representation of a potential tri-generation system at an underground coal mine

Appendix 18 World Market opportunities for VAM technologies

(Task 3.4)

Country, 2009 Coal Production (10^6 MT/yr)	VAM Output 10^9 m^3	Mining Conditions	Market Comments
China 2,666	18.3	Numerous underground coal mines are very gassy.	CDM mechanism in place with carbon value extended until 2020 to enable return on investment.
India 529		Most underground mines are small scale and not gassy, currently offering poor conditions for VAM abatement.	Coal India plans to dramatically increase coal production by 2020.
United States 378	2.8	Large gassy mines give favourable conditions for VAM abatement projects.	Weakened global voluntary carbon market (VCM) presents high financial risk hurdles.
Russia, Ukraine, Kazakhstan 300[a]	6.2[a]	Wide range of mining techniques, mining conditions and levels of mechanisation.	Significant potential for VAM abatement projects.
South Africa 250		Mines tend to be shallow, thus not gassy.	Good future potential as coal mines access deeper reserves.
Australia 117	1.3	A number of gassy underground mines are in operation.	Recent adoption of carbon tax could provide a mechanism for return on investment.
Mexico 12		Majority of mines use modern longwall techniques and exhibit favorable VAM abatement characteristics such as large volumes and high VAM concentrations.	CDM and VCM mechanisms in place with carbon value till 2020 to enable return on investment.

[a] Combined values for 3 countries.

World market opportunities for VAM technologies.

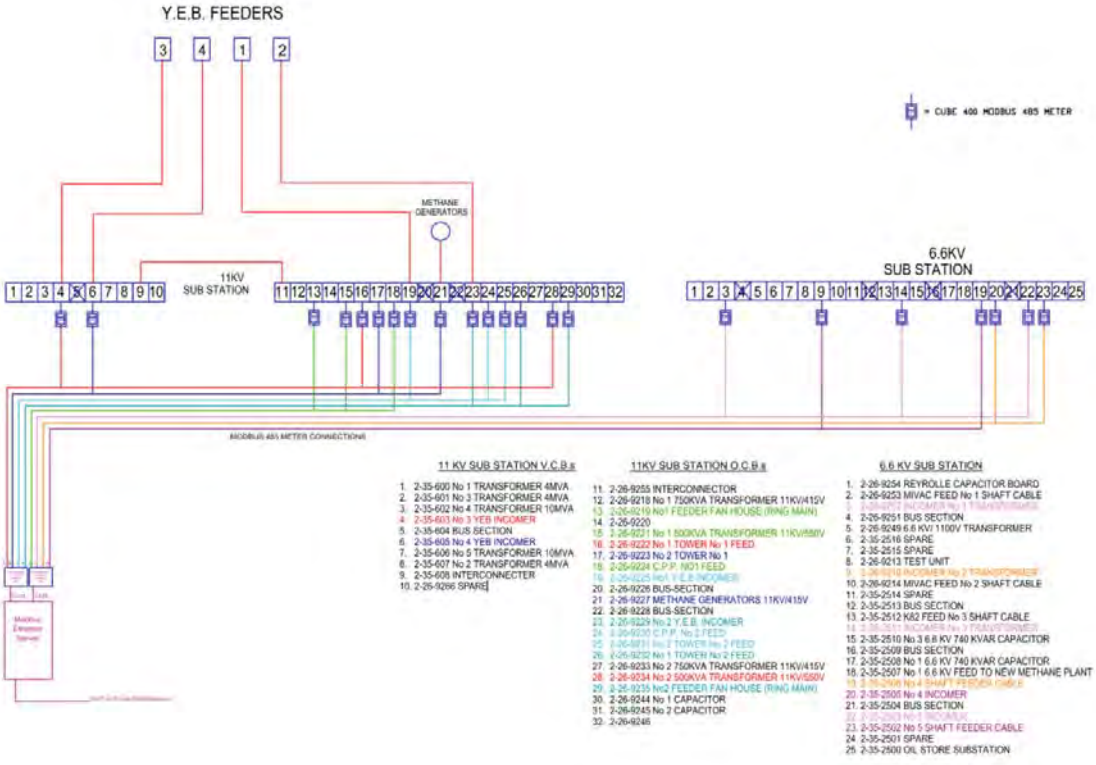
Source: Somers J; A 2012 update on the world VAM oxidizer technology market; US EPA

Appendix 19 Installation of digital metering system at Kellingley Colliery

(Task 4.1)

Original Energy Meter

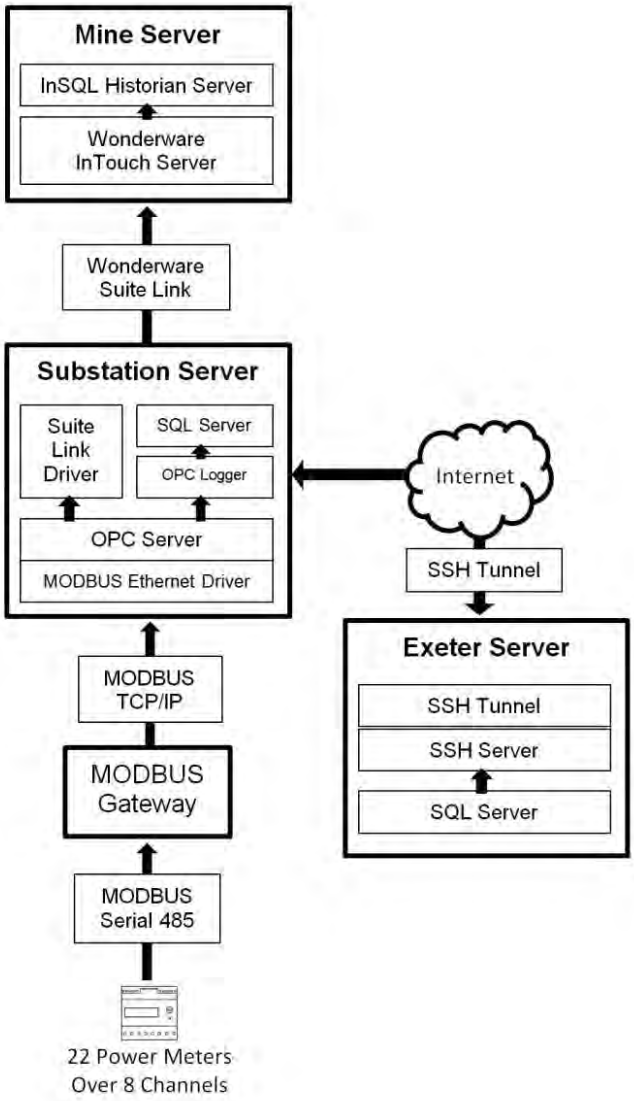
New Digital Power Meter



Modbus Communication Wiring

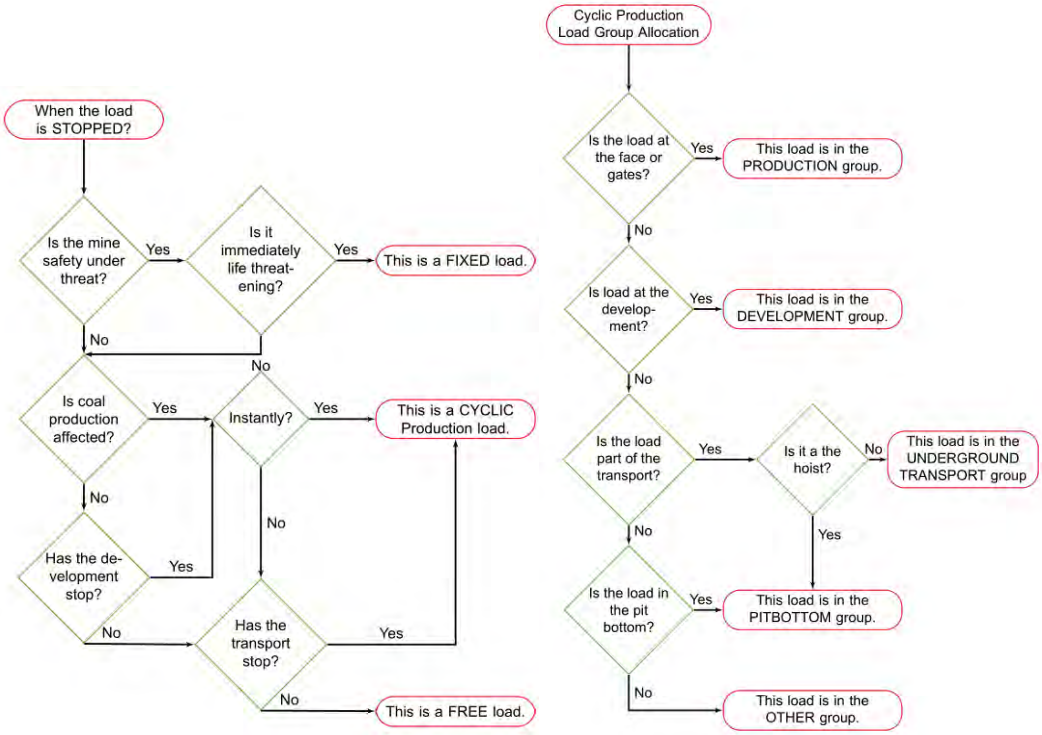
Appendix 20 Data Acquisition Software System

(Task 4.1)

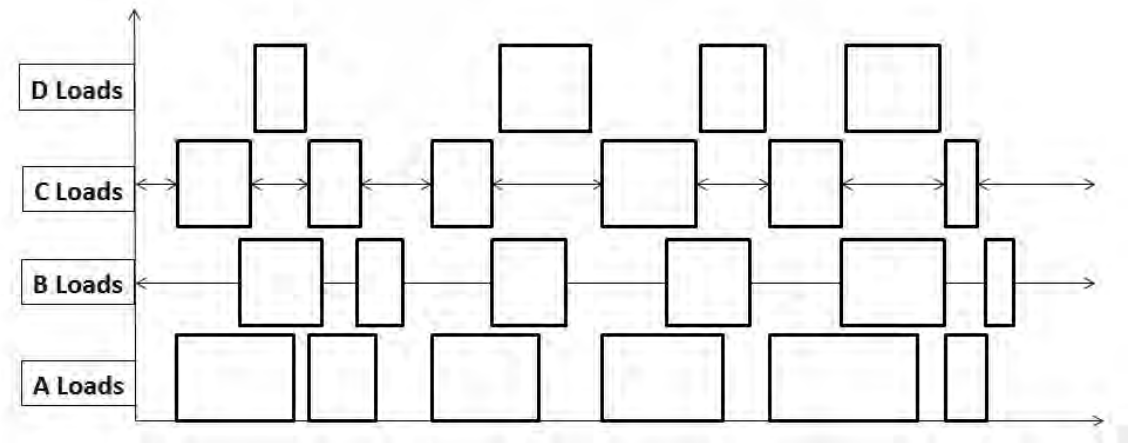


Appendix 21 Load classification & scheduling

(Task 4.2)



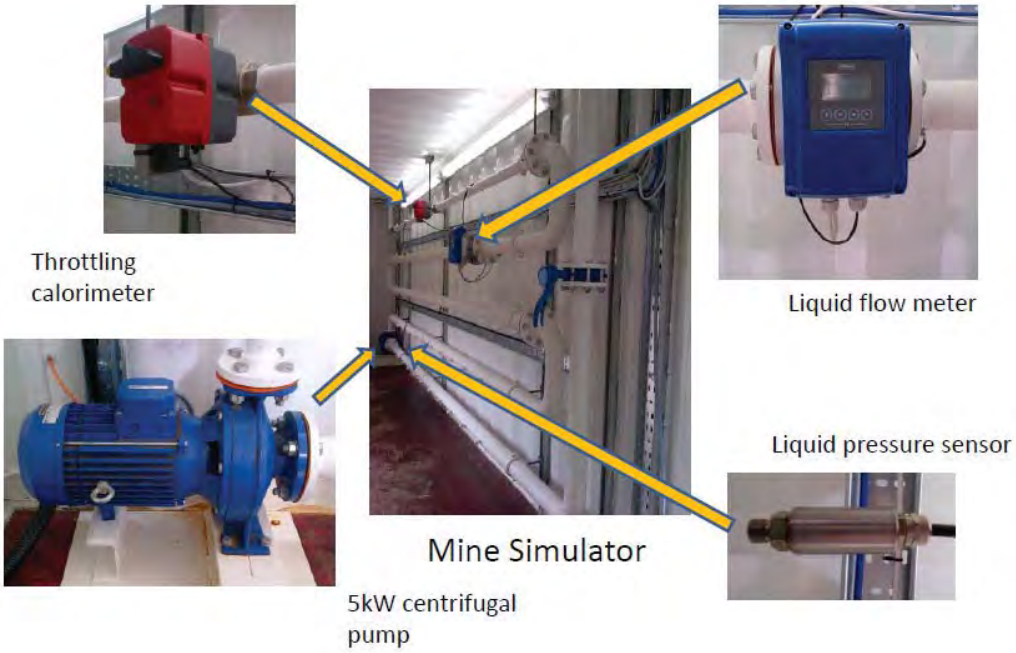
Flow diagrams; *left* Main category allocation *right* Load group allocation



LC Matrix Scheduling

Appendix 22 Mine Simulator housing and installed equipment

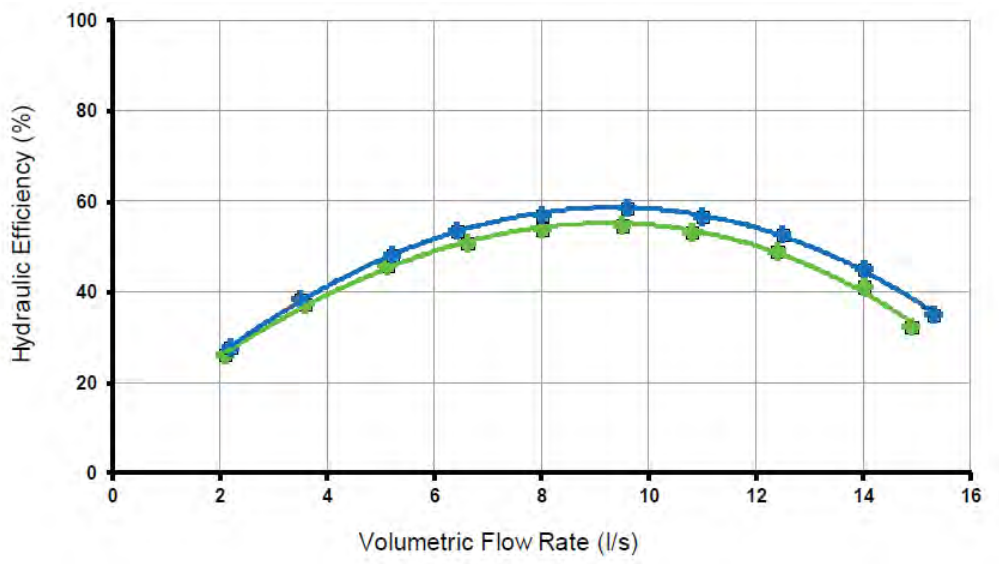
(Task 4.4)



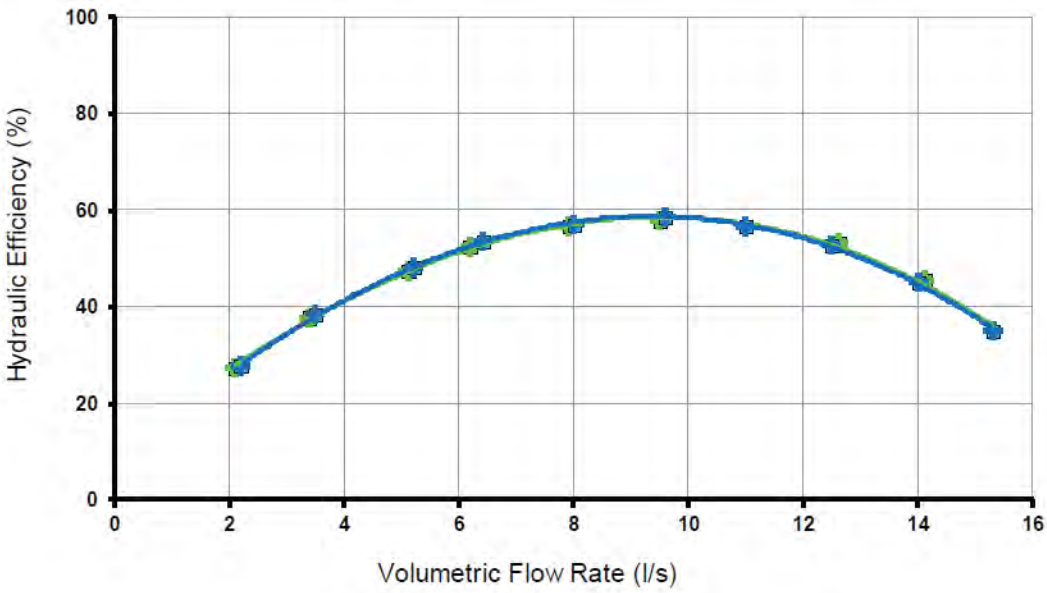
Appendix 23 Comparison of hydraulic efficiency – Poirson vs conventional method

(Task 4.4)

a) Mine simulator - before adjustment for motor efficiency



b) Mine simulator - after improvement in motor efficiency rating



N.B. The actual efficiency for this test pump shows as low (25%- 60%) due to the mismatch of pump capability with head in the simulator as well as system 'throttling' to investigate and to emphasise the comparative research outcome.

Appendix 24 Turnaround efficiency and heat storage for a compressed air storage facility

(Task 5.2)

In the normal compressed air energy storage (CAES) system, both electricity and gas are required as inputs to the system to get electricity as output from the system. Therefore, the turnaround efficiency (η) can be determined simply as

$$\text{Turnaround Efficiency (\%)} = \frac{\text{Amount of Electricity as Output}}{\text{Amount of (Electricity + Gas) as Input}}$$

Let's consider the case of Huntorf CAES plant (in Germany), which requires 0.8 kWh electricity and 1.6 kWh gas as inputs for an output of 1 kWh electricity. Therefore,

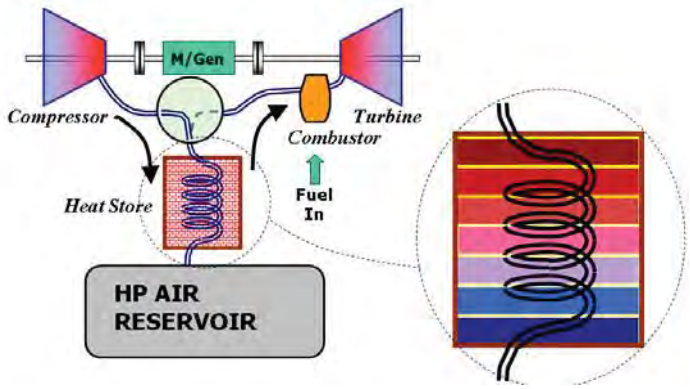
$$\text{Turnaround Efficiency} = \frac{1}{0.8 + 1.6} = 42\%$$

In calculating the efficiency of a plant, care should be taken over the value of input energy, particularly the value placed upon gas because 1 kWh of gas cannot simply be converted into 1 kWh of electricity – if 1 kWh of gas is used in a combined cycle gas turbine with a realistic efficiency of 55%, only 0.55 kWh electricity will be generated. Using this 55% efficiency, the effective turnaround efficiency of Huntorf CAES plant becomes

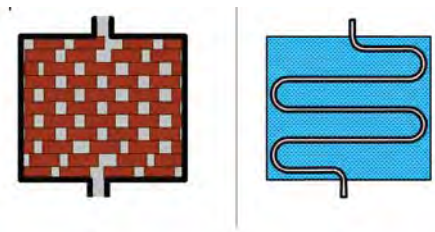
$$\text{Effective Turnaround Efficiency} = \frac{1}{0.8 + 55\% \times 1.6} = 60\%$$

Therefore, the Effective Turnaround Efficiency can be defined as

$$\text{Effective Turnaround Efficiency (\%)} = \frac{\text{Amount of Electricity as Output}}{\text{Amount of (Electricity + Gas} \times \text{Turbine Efficiency) as Input}}$$



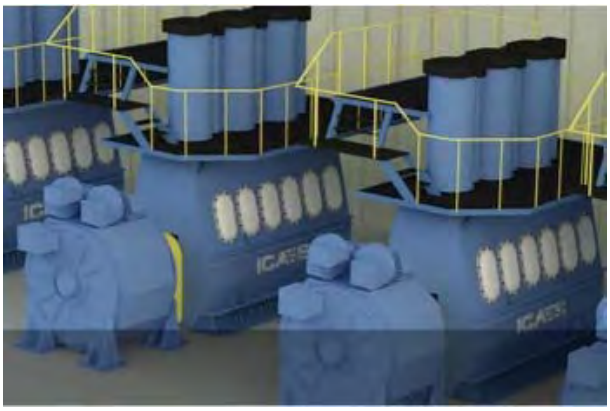
Thermal storage function (above) and thermal store options (below) for CAES



Appendix 25 Some equipment required for a compressed air energy storage facility

(Task 5.2)

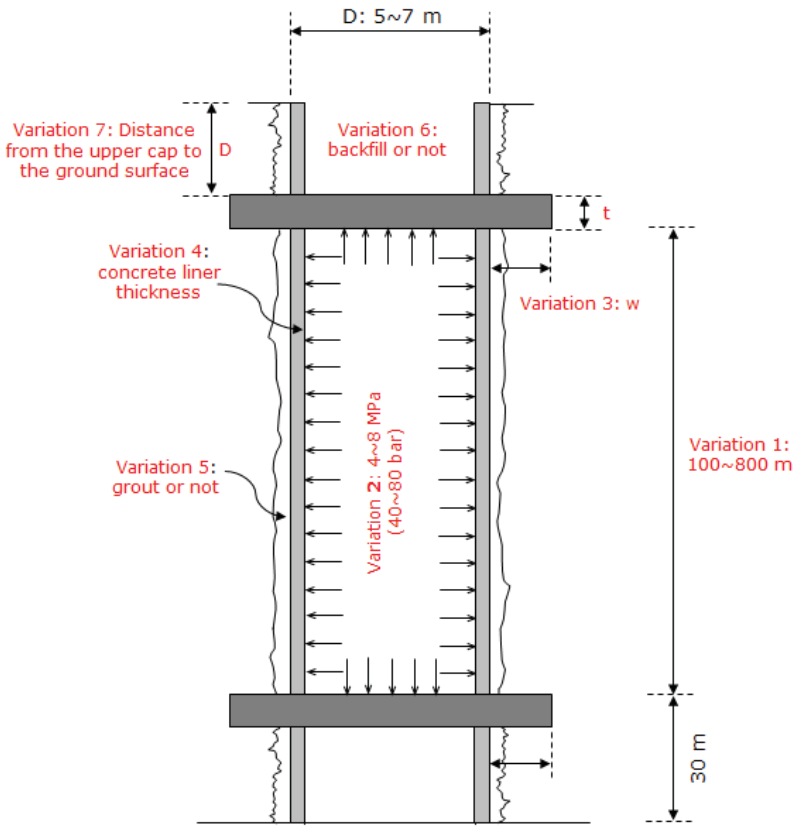
Hot gas expander (source GE)



Reversible motor / generator (Source Sustain-X)

Appendix 26 Factors and variations for CAES shaft sensitivity study

(Task 5.3)



Shaft cap thickness **t** ≥ 0.45 m

Shaft cap depth into the surrounding rock **d** ≥ 0.75 m

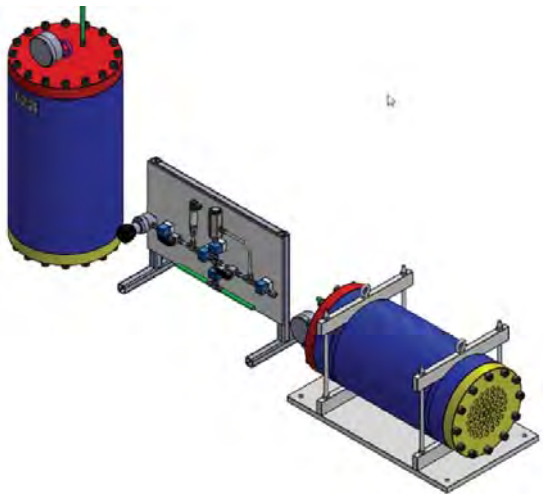
Shaft cap diameter ≥ 2D

Variation		**Different values**
V1: Shaft length for storage		100, 200, 400, 800 m
V2: Compressed air pressure		4, 6, 8 MPa
V3: Shaft cap material and dimension	Thickness: t	0.5, 0.75, 1, 1.25 m
	Width into surrounding rock: w	0.8, 1.2, 1.6, 2 m
	Materials*:	NSC, HSC
V4: NSC concrete liner thickness		0.4, 0.5, 0.6, 0.7, 0.8 m
V5: Grout or no grout behind the shaft liner		Yes , no
V6: Backfill or no backfill above the upper shaft cap		Yes, no
V7: Distance from the upper shaft cap to the ground surface		5, 10, 15, 20, 30 m

*NSC – normal strength concrete with uniaxial compressive strength 35 MPa
HSC – high strength concrete with uniaxial compressive strength 100 MPa
Underlined values – those used in the basic models

Appendix 27 Concrete testing for CAES shaft lining

(Task 5.4)



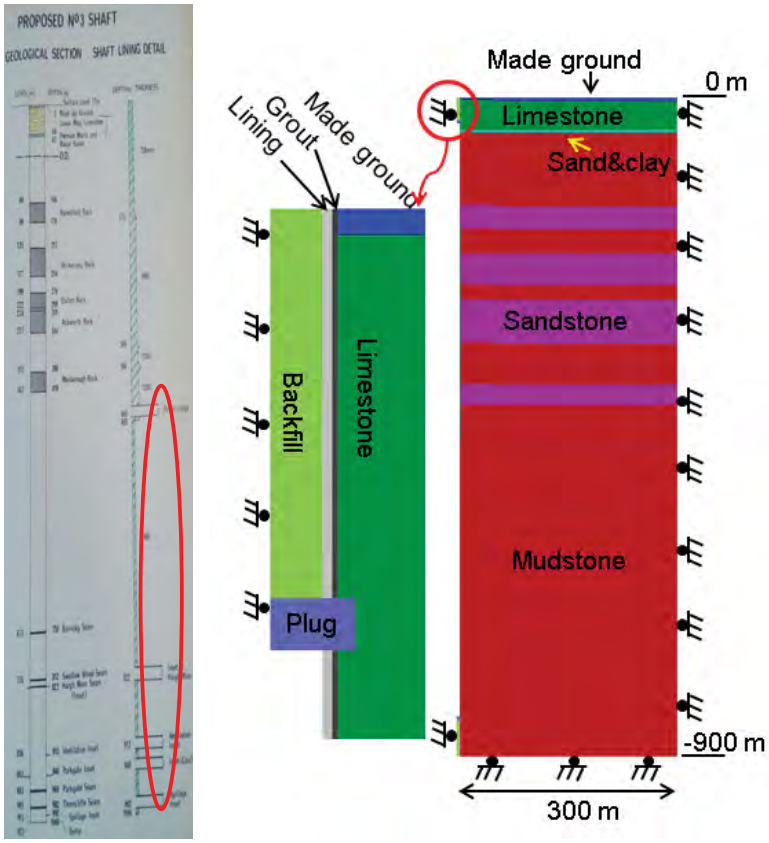
Schematic of concrete test rig

Left: Compressed Air Reservoir, Centre: Manifold, Right: Sample Vessel in Cradle

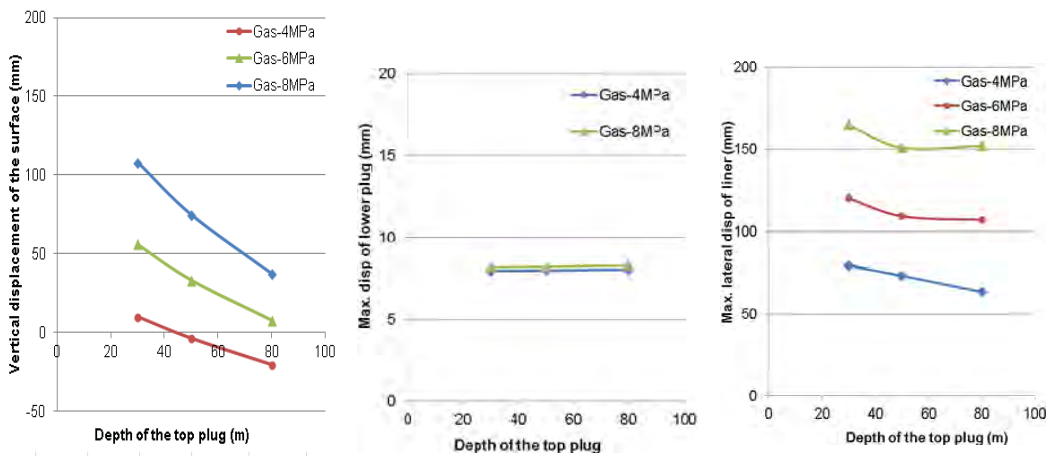
Concrete sample prepared

Appendix 28 Geotechnical modelling of selected shaft Maltby 3 for CAES

(Task 5.4)



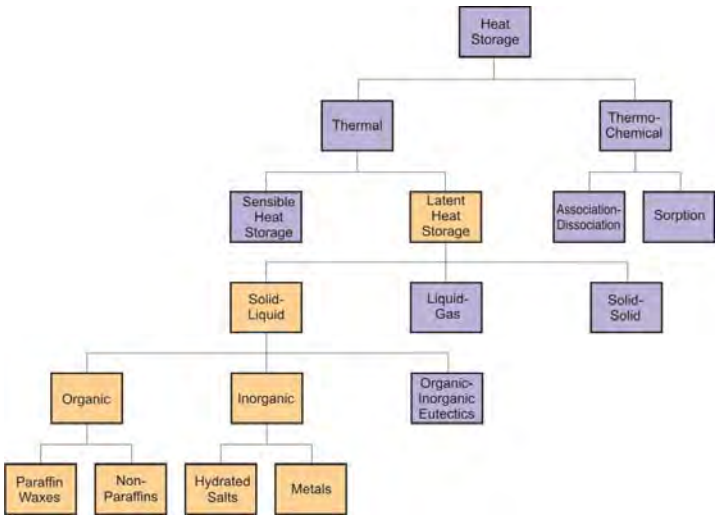
Original shaft plan and 2D model for CAES assessment



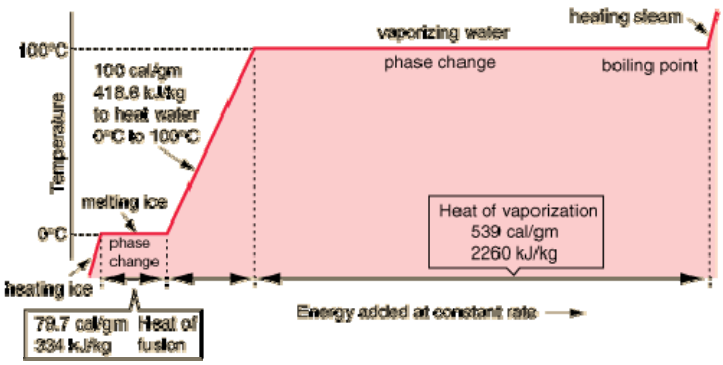
Modelled displacements of ground surface, lower plug and shaft liner

Appendix 29 Review of heat storage methodologies

(Task 5.5)



Categories of heat storage



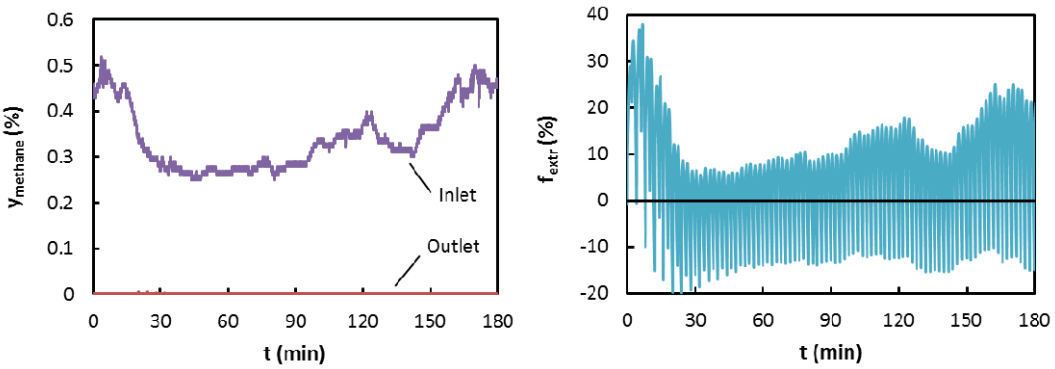
Sensible vs Latent (phase change) heat – H_2O as a demonstration

System	Advantages	Disadvantages
Active storage: direct steam generation	- Intermediate heat transfer fluid and steam generation exchanger is not necessary, improving the efficiency loss in steam generation. - Simpler overall plant configuration. - Lower investment and O&M costs.	- High pipe cost (high pressure). - Need of auxiliary protective heating systems for start-up, maintenance and recover from frozen conditions. - Instability of the two phase flow inside the receiver tubes.
Passive storage: concrete/ceramics	- Very low cost of thermal energy storage media. - High heat transfer rates into and out of the solid medium. - Facility to handling of the material. - Low degradation of heat transfer.	- Increase of cost of heat exchanger and of engineering. - Long-term instability.

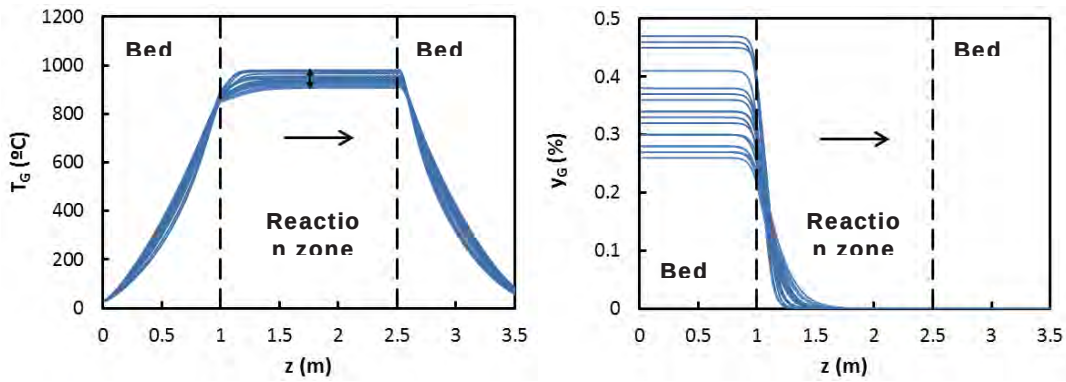
Advantages and disadvantages of sensible system options

Appendix 30 Simulations of heat storage system coupled to a TFRR

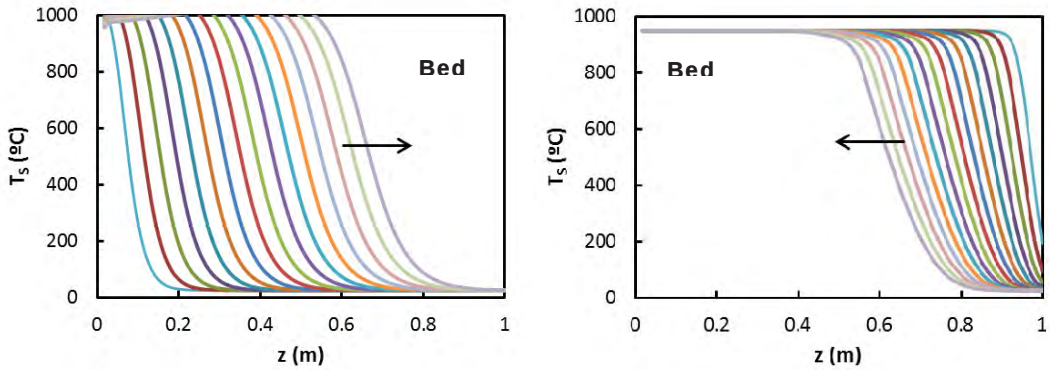
(Task 5.5)



Evolution of CH_4 concentration (left graph) and fraction of gas flow drained from the TFRR to the heat storage bed (right graph)



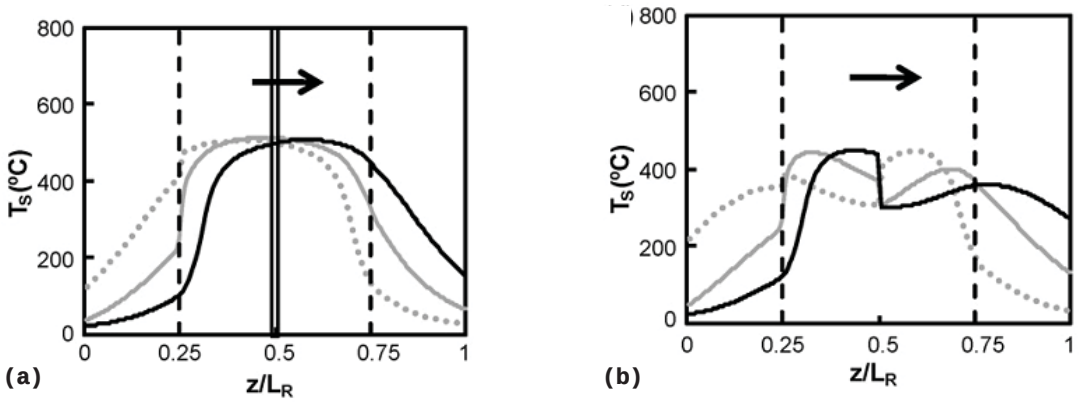
Temperature (left graph) and CH_4 concentration (right graph) profiles in the TFRR



Temperature profile in the heat storage system (bed C) at high (bed charging: left graph) and low (bed discharging: right graph) concentration periods.

Appendix 31 Flow reversal reactor temperature profile simulations with heat extraction strategies

(Task 6.1)



Comparison of energy recovery strategies

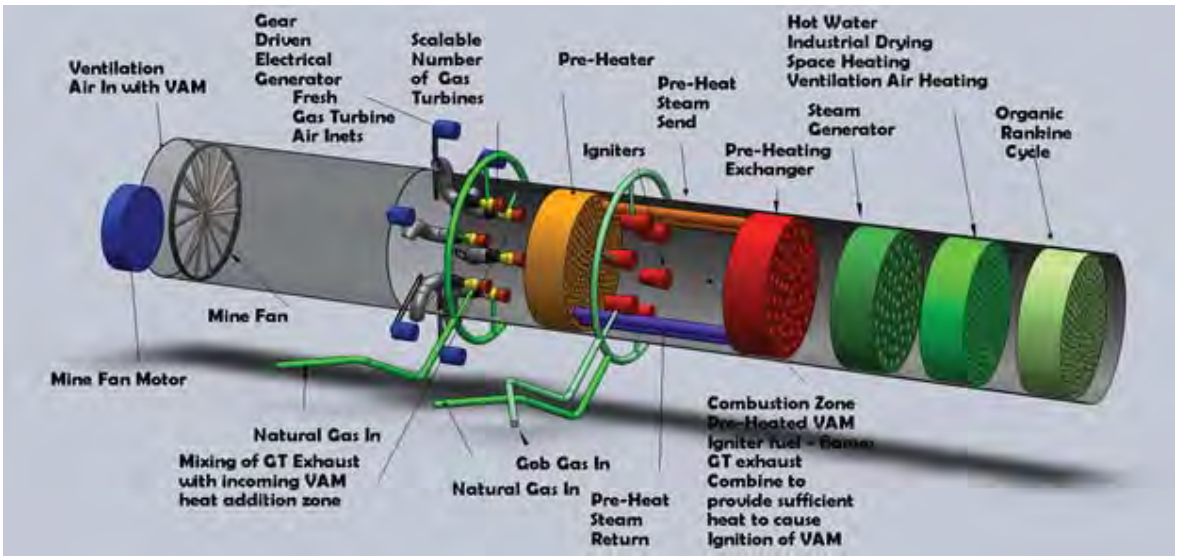
Hot gas withdrawal with no gas return to the reactor (a) produces very little disruption on the reactor performance, apart from the desired temperature decrease.

Heat exchange with gas return option (b) breaks the parabolic profile of the reactor, resulting in a loss of symmetry and hence a decrease on the reactor efficiency.

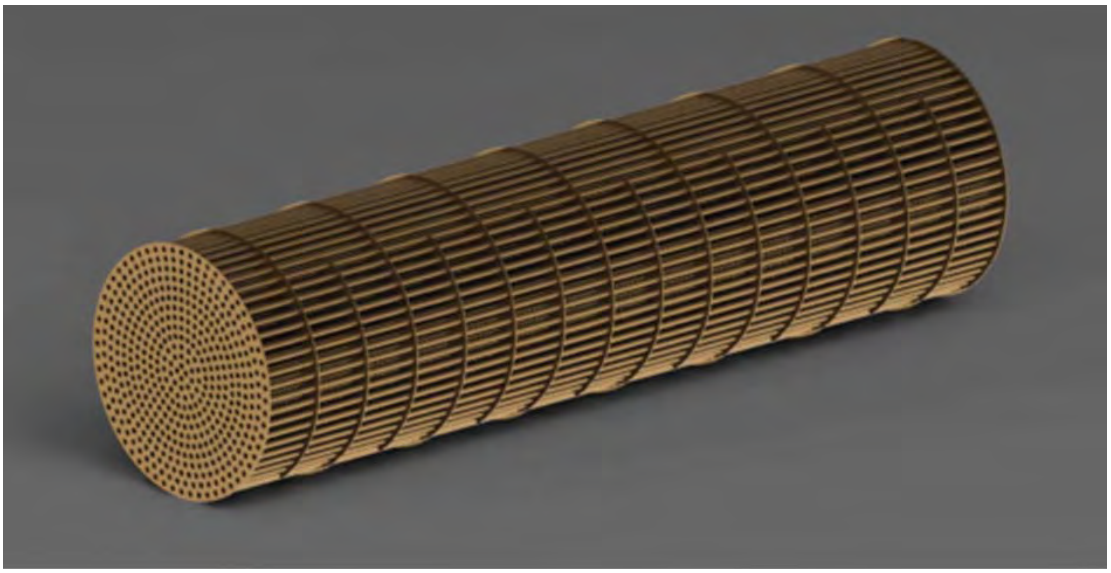
However, (b) operates within a temperature range of 450-500ºC (catalytic flow reversal reactor) or 850-900ºC (thermal flow reversal reactor), whereas (a) operates within 25-500ºC or 25-900ºC, respectively. The higher the temperature reaches, the higher the subsequent efficiency for loads such as electricity generation.

Appendix 32 Gas Turbine configuration for energy exploitation and VAM oxidation

(Task 6.1)



Overall system for VAM oxidation and energy exploitation

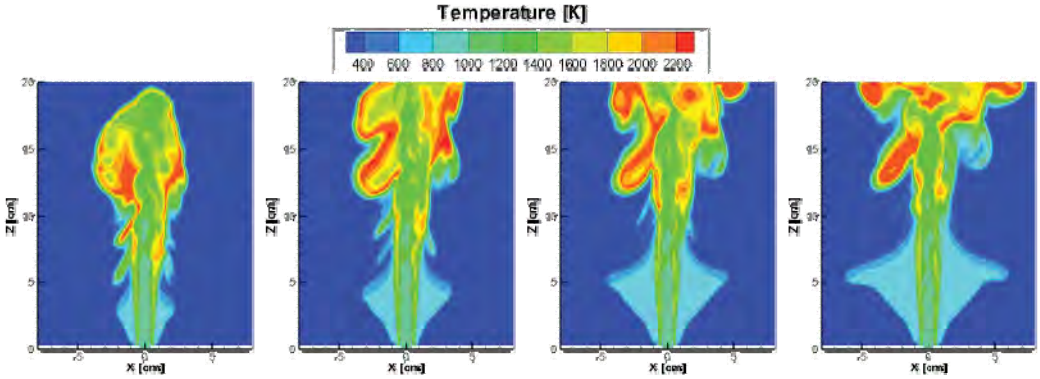


Example heat exchanger for combustion zone Solidworks: 30mx8m, baffle separation 2m

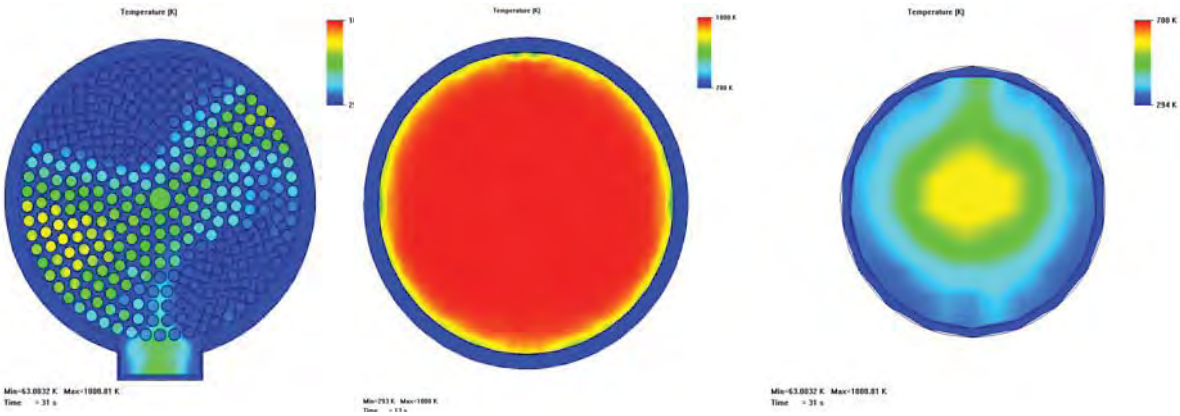
Appendix 33 Modelling the LOWCARB gas turbine concept

(Task 6.1)

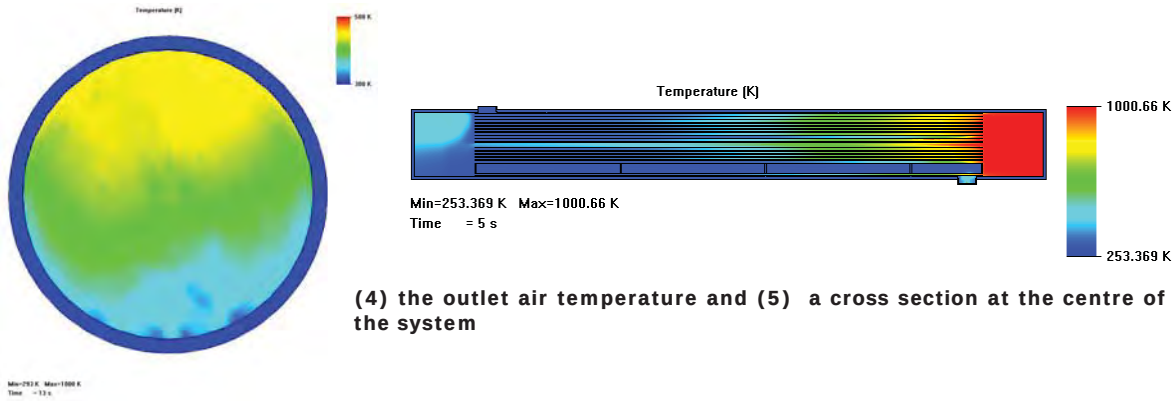
Example of combustion process modelling; 2.0 % VAM at 300°K pre-heated



Examples of heat exchanger function for the combustion zone



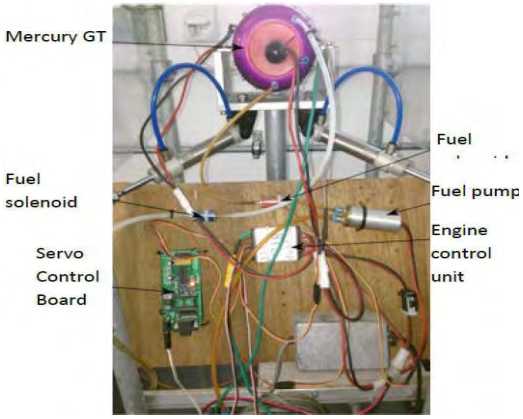
(1) A cross section of a shell and tube heat exchanger centered on the water outlet, (2) The air flow at the air inlet kept at a constant 1000 K and (3) a slice through the water outlet. Time of measurements for 1 , 2, and 3 are 31 seconds from start-up.



(4) the outlet air temperature and (5) a cross section at the centre of the system

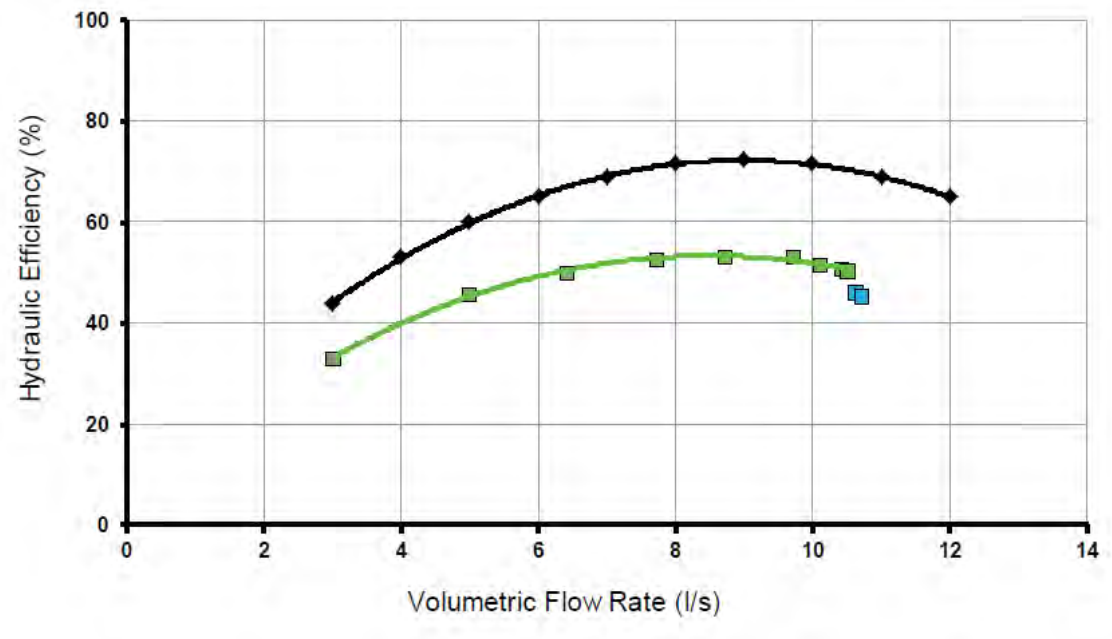
Appendix 34 Images of field trial of the gas turbine VAM oxidation concept

(Task 6.1)



Appendix 35 Performance of the CSM Test Mine pump using the Poirson method

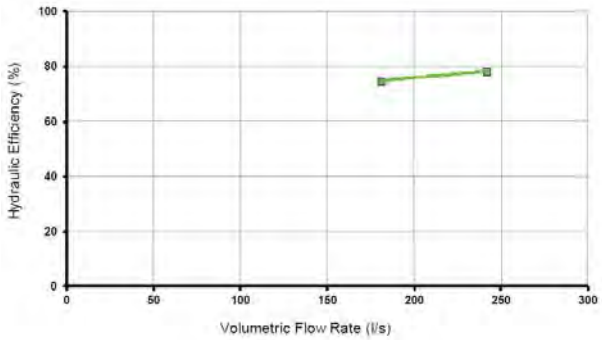
(Task 6.1)



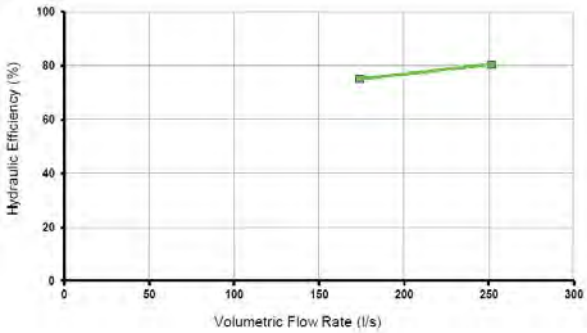
Note that the measured hydraulic efficiency (green) is well below the estimated original equipment manufacturer (OEM) performance (black) across the range of flow rates for this very old pump.

Appendix 36 Performance of four clay site pumps using the Poirson method

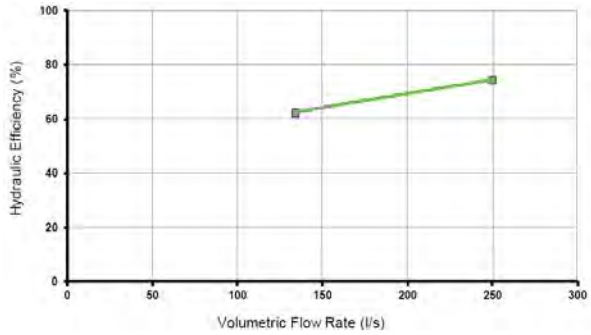
(Task 6.1)



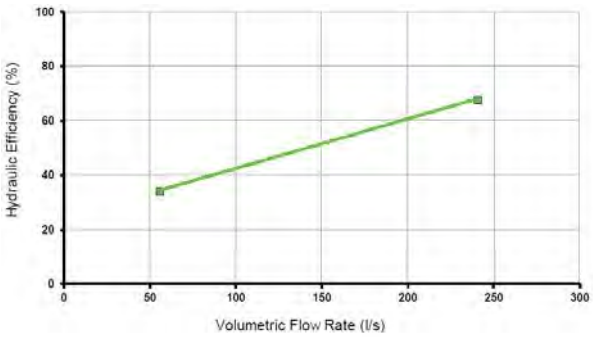
1.



2.



3.

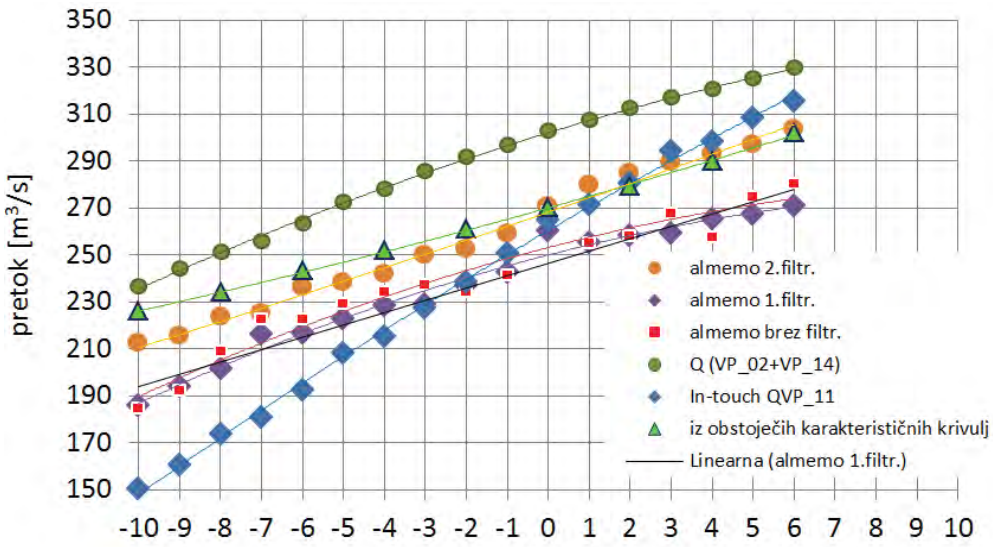


4.

N.B. Pumps 1 and 2 show acceptable efficiency; pump 3 requires minor improvement work; pump 4 requires major refurbishment

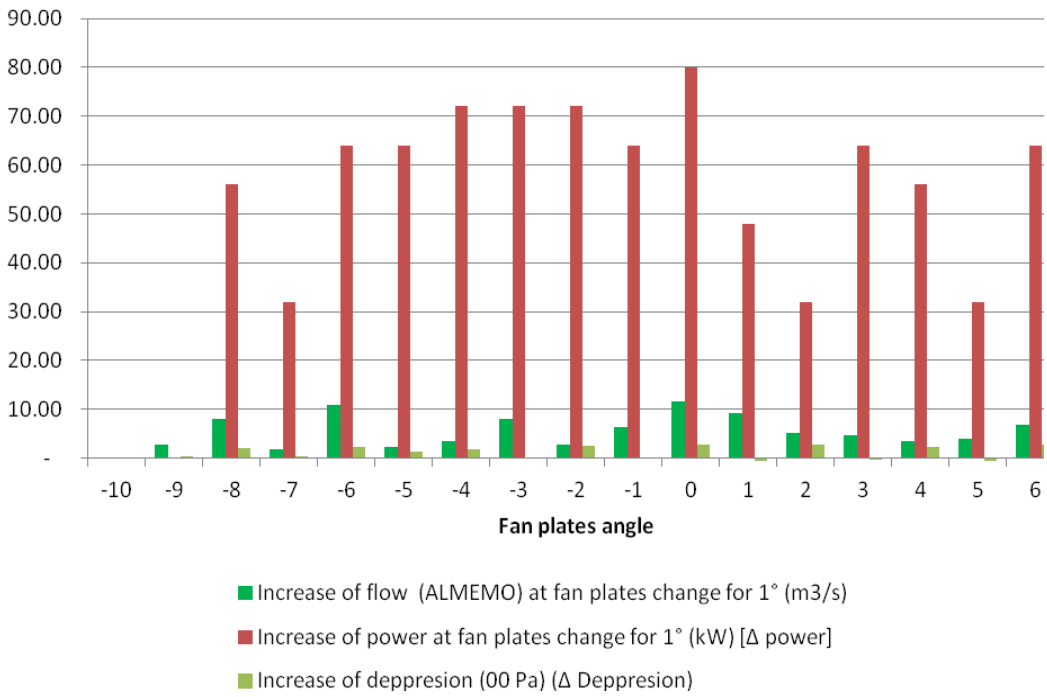
Appendix 37 Velenje mine ventilation – flow rate measurement comparisons

(Task 6.1)



Flow rate at different fan angles by different measuring methods

This shows data measured with ALMEMO (original, 1. Filtration and 2. Filtration of data), data from fixed air speed sensors in the mine, data from VTIS and estimated data from the existing characteristic curve based on measured pressure drop. Measurement with ALMEMO at the fan suction pipe was compared with a summary of two roadways airflows (VP 14 and 2) which are jointing in to the air exiting shaft.



Changes in flow, power (cost) and pressure drop at different fan angles

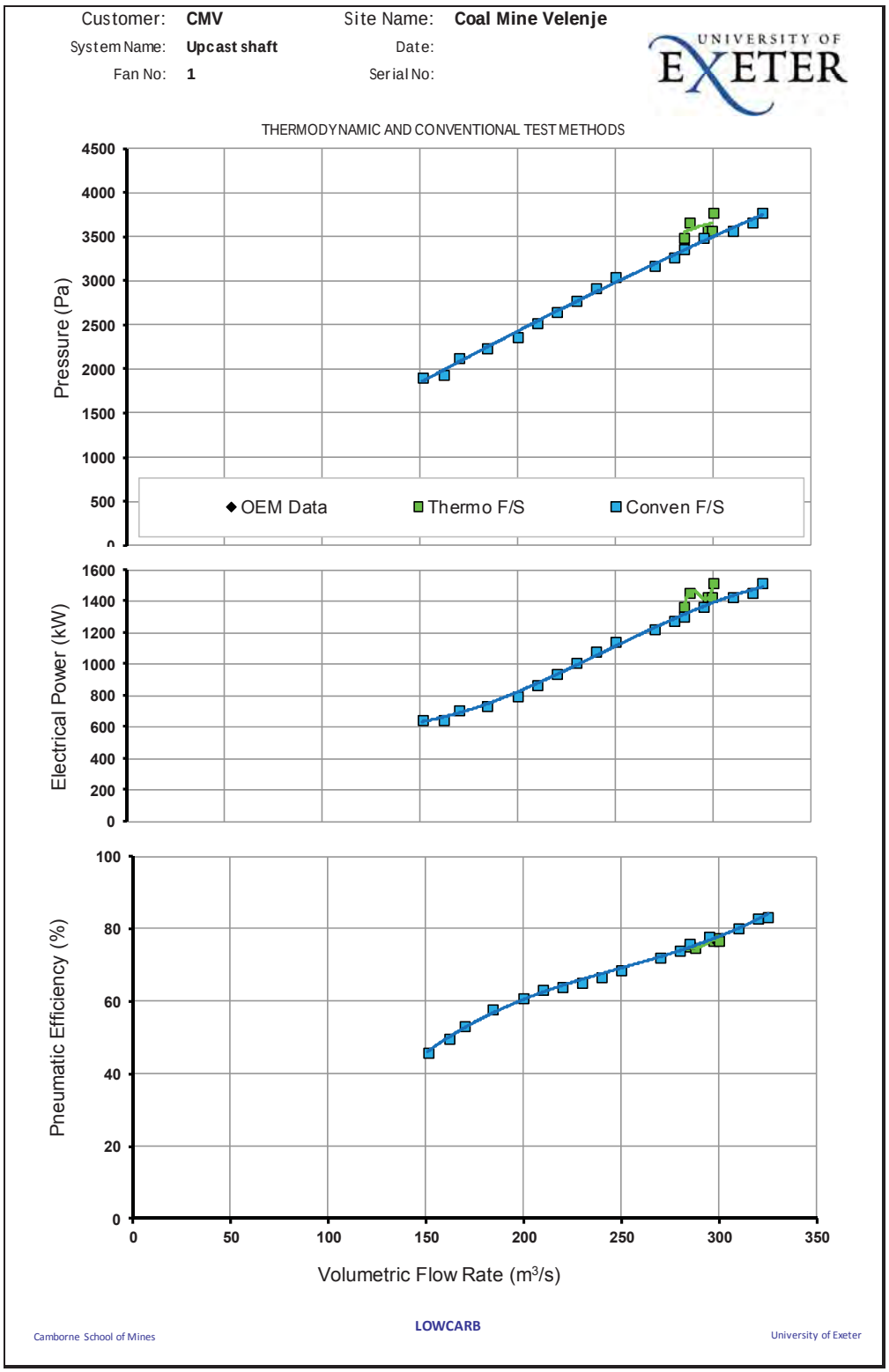
Appendix 38 Installation of CSM sensors across Velenje mine fan

(Task 6.1)

		
EQUIPMENT LIST	**INSTALLING TEMPERATURE SENSORS**	**INSTALLATION OF PRESSURE SENSORS**
		
PRIMING MANOMETER TUBE	**LAPTOP AND DAU SET UP**	**FAN STATION LAYOUT**

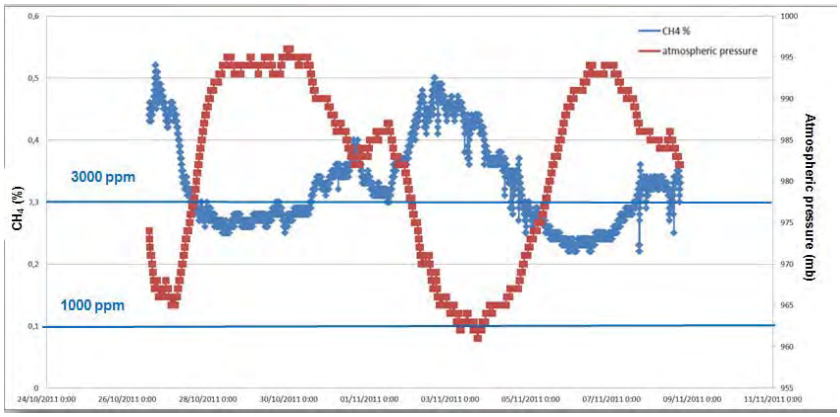
Appendix 39 Comparison of conventional and thermodynamic fan efficiency measurement

(Task 6.1)

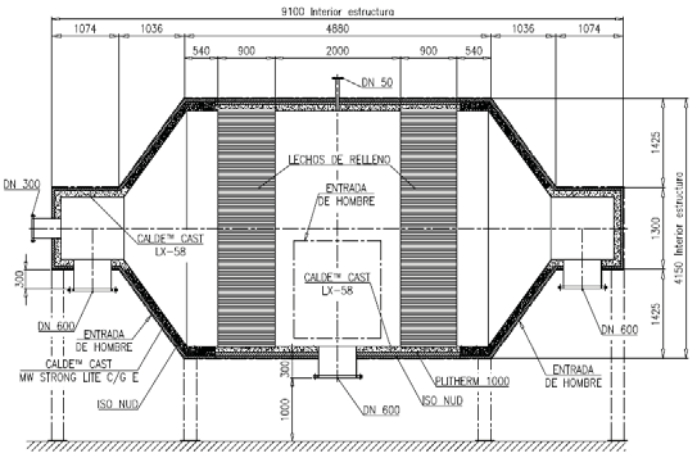


Appendix 40 Design work for the Thermal Flow Reversal Reactor

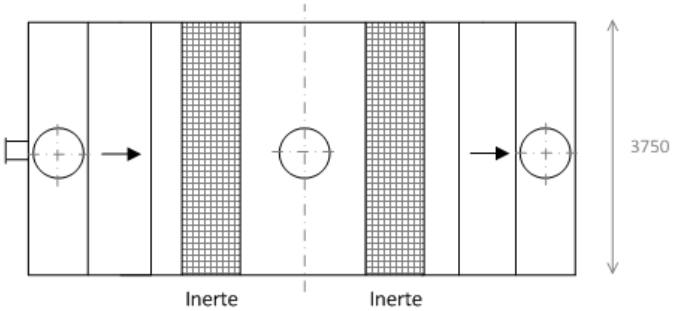
(Task 6.3)



Maria Luisa upcast shaft; CH4 and atmospheric pressure readings vs time



TFRR plan view (mm)

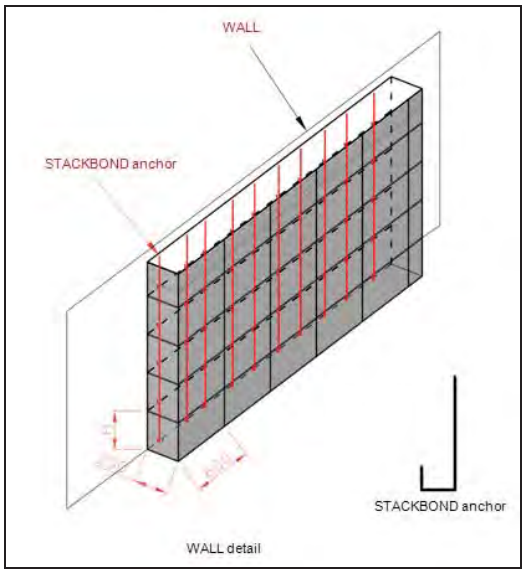


TFRR elevation view (mm)

Continued

Honeycomb bricks

The dimension of each bed will be 3.75 m x 3.75 m x 0.9 m and it will be built with 25 x 25 x 3 (height x thickness x length) standard blocks of cordierite.



Combustion chamber wall

Walls:

- Refractory bricks, thickness 115 mm.
- Dense bricks quality 40/42% Al_2O_3.
- Biosoluble ceramic fibre (insulation layer); thickness 19 mm, density 128 kg/m^3
- Insulating panel of calcium silicate; thickness 100 mm.
- Anchors to connect bricks into wall.

Floor:

- Refractory bricks, thickness 115 mm.
- Dense bricks quality 40/42% Al_2O_3.
- Insulating panel of calcium silicate; thickness 100 mm.

Roof:

- Prefabricated modules of ceramic fibre over-compressed. Density 160 kg/m^3, T 1260 ºC, thickness 250 mm, with anchors type stackbonds.

Appendix 41 Pump energy savings estimated from using the CSM Poirson technology

Parameter:	Pump 1	Pump 2	Pump 3	Pump 4	Total	Units
Head	125.9	125.7	124.9	125.5	-	m
Flow rate	180.8	173.9	134.1	55.6	-	l/s
Electrical power	310.4	297	272.5	209	-	kW
Hydraulic efficiency	74.8	75	62.6	34.4	-	%
Operating time	50%	50%	50%	50%	-	%
Volume pumped	2851	2742	2114	877	8584	Ml/year
Energy	1359552	1300860	1193550	915420	4769382	kWh/year
Tariff	0.094	0.094	0.094	0.094		£/kWh
Specific power	476.9	474.4	564.5	1044.2	555.6	kWh/Ml
Energy cost	127,798	122,281	112,194	86,049	**448,322**	£/year

Four pumps – current annual energy cost

N.B. £448,322 per annum = €537,986 per year at an exchange rate of 1.2

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KI-NA-26832-EN-N

This underground coal mining project addressed THE emissions of the greenhouse gas methane from the mining process and opportunities to improve the efficiency of energy use and hence secondary carbon dioxide emissions. Following extensive underpinning research into the specified problems, a methane reserves assessment for a mine and a study of low emission methane utilisation were carried out, while work on coal properties from Polish mines has provided detailed results, with a comparative study of Slovenian coal. Computer simulation was used to plan in-mine pre-mining drainage tests, while indicators for outburst prediction have been specified and tests with ground penetrating radar to detect methane in seams carried out. For ventilation of air methane, work has been undertaken on oxidation technologies, including scientific analysis of the catalytic version of the process, and computer models have been developed for the major options. Tri-generation potential and gas cleaning and concentration techniques have been studied. An analysis of mine electricity use for profiling loads led to simulation to enable optimisation strategies. A ventilation improvement initiative at a complex mine improved understanding of gas movements and air flows, linked to software development, while a mine simulator for pumping efficiency provided the basis for actual pump and fan measurements. An energy storage study and work on compressed air systems and the geotechnical potential of abandoned shafts was applied to a theoretical case study and heat storage studies supported several technological areas. A techno-economic analysis was applied to selected processes. Only the energy efficiency initiatives are genuinely viable in the current economic circumstances.

Studies and reports

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