A Tier 3 method to estimate fugitive gas emissions from surface coal mining

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Abstract

Fugitive coal seam gas emissions from open cut coal mines (surface mining) arise when coal and associated strata are blasted, fractured and disturbed as part of the mining process. This liberates the seam gas that is trapped within coal seams. The Intergovernmental Panel on Climate Change recommends using an emission factor (*EF*) approach as the basis for estimating fugitive coal seam gas emissions. The *EF* is the volume of gas (m^3) released per tonne of coal produced. Three levels of accuracy are associated with estimation of these emissions: Tier 1, 2 and 3, which each have increasing levels of accuracy. Tiers 1 and 2 provide average *EF* values for the whole country or the coal basin, while Tier 3 provides *EF* values specific to a coal mine. Until recently, Australian open cuts used nominal *EF* values of 3.2 m^3/t and 1.2 m^3/t for the two main coal producing states of New South Wales and Queensland, respectively. These values were used for all mines in these states, irrespective of the level of 'gassiness' of specific coal seams and strata. During the last few years, we have developed a new method for Australian open cut mining that is specific to each mine site. The proposed Tier 3 method, which has been adopted by the National Greenhouse and Energy Reporting, considers the coal seams and clastic rock horizons as gas reservoir units that release their gas during mining. The primary data required are the in situ gas content and gas composition of coal and carbonaceous rocks contained within the column of strata called the 'gas release zone'. In this methodology, regions of similar gas content and reservoir properties are termed a 'gas zone'. A small number of drillings are required to characterise the gas zone and to provide inputs to the model.

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This paper describes this new method of estimating fugitive gas emissions from surface coal mining. To illustrate the procedure of the calculations, the method is applied to an active Australian open cut coal mine.

Keywords: *Coal seam gas; CSG; open cut mine; fugitive emissions; Tier 3 method; gas content.*

1. Introduction

Australia is a major producer of coal. It has about 72 billion tonnes of identified bituminous coal resources, with about 363 million tonnes (Mt) of saleable bituminous coal production for the 2009–10 financial year (Resources and Energy Statistics, 2012). About 275 Mt (76%) of the total saleable coal was produced in open cut mines. Fugitive emissions from all coal mines for the year 2009 were estimated at 28.7 Mt carbon dioxide equivalents (CO_2-e), which is 5.3% of Australia's total man-made greenhouse gas emissions (Australian National Greenhouse Accounts, 2011). While the bulk of emissions are from underground coal mining, about 30% of emissions come from open cut mining. In 2007, the *National Greenhouse and Energy Reporting (NGER) Act 2007* was established, which required Australian coal mines to report their annual fugitive emissions. The first annual reporting period was from 1 July 2008. A carbon pollution tax has also been introduced by the Australian government to reduce greenhouse gas emissions. It takes effect from 1 July 2012.

Developing accurate and practical methods to estimate emissions from coal mining in Australia has been an ongoing research activity since the early 1990s, following the Kyoto conference on climate change (Williams and Saghafi, 1993; Williams et al., 1996; Saghafi and Williams, 2002). In underground mining, gas is released into the mine's drainage and ventilation systems. Therefore, total emissions from coal can be estimated with relative ease. In contrast, in open cut mining, coal seam gas (CSG) is released from diffuse sources and emitted directly to the atmosphere. Therefore, estimating CSG emissions is not straightforward and can be uncertain. Open cut mines have a multitude of gas emission sources, including open exploration boreholes, extracted coal seams, removed overburden carbonaceous and other gas-bearing sedimentary rocks, thin and ashy coal seams tipped in spoil piles, standing highwall, and underburden coal and rocks.

These emission sources are spread widely across the mine lease, and the diffuse nature of emissions makes direct measurement inaccurate and difficult.

Emissions from mining are generally quantified using the concept of an Emissions Factor (*EF*), or specific emission. This terminology is similar to the one used for mine safety purposes in underground mining, where specific emission presents the volume of gas released into the coal face for each tonne of coal extracted (see e.g. Boxho et al., 1980; Creedy, 1993; Creedy et al., 1997). To quantify fugitive emissions from mining, generic *EF* values have been suggested and used while measured established values have been unavailable. The Intergovernmental Panel on Climate Change (IPCC, 2006) recommends using *EF* with a tier qualifier. The three levels of accuracy, namely Tier 1, 2 and 3, are defined as follows:

- Tier 1: *EF* is a generic number. Tier 1 numbers are only applied if no data on gas content and emissions are available for the coal basin and country.
- Tier 2: *EF* is basin specific. Its value is an average number for one or several coal basins.
- Tier 3: *EF* is mine specific. It is determined by measuring emissions from an individual mine and is therefore the most accurate (see e.g. the IPCC 2006 report for more official definitions of these terms).

In the following sections, after a brief history of the evolution of methods of measuring and evaluating CSG emissions from open cut mining in Australia, the details of the new Tier 3 method are presented. The new Tier 3 model has been adopted by the Australian NGER (National Greenhouse and Energy Reporting, 2009) and recommended to open cut coal mines for evaluation of their CSG emissions.

2. History of methods for estimating fugitive emissions from open cut mining in Australia

In the last two decades, several studies have been undertaken in the coalfields of eastern Australia to develop a method for evaluating emissions from open cut coal mining. Two methods for direct measurement of emissions, here referred to as global emissions measurement and spot emissions measurement – were initially devised and applied to some open cut mines in two main coal regions of Australia. Global emissions measurement uses air pollution techniques to assess emissions by measuring gas concentration downstream of winds sweeping through the coal mine. This method also requires measurement of wind velocity and assumptions about the shape of the plume and wind velocity distribution in the plume. Spot emissions measurement uses a chamber technique to measure emissions emanating directly from exposed surfaces, such as uncovered coal seams, spoil piles and ground surface. A purpose-built chamber covers the targeted surface, and gas accumulation in the chamber is measured to estimate gas flux emitted from the exposed surface. In the next sections, these two methods are described in more detail.

2.1. Global emissions measurement method

A global emissions measurement method was developed and applied to 17 open cut coal mines in the Sydney and Bowen Basins in the early 1990s (Saghafi and Williams, 1992; Williams et al., 1993). This method was developed using air pollution techniques, which determined emissions by measuring wind speed and gas concentration in the proximity of emissions sources (one or a group of coal mines). In this method, the crosswind profile of the CSG plume was assumed. Wind speed and gas concentration were measured using an instrumented vehicle (Figure 1). The traverses were made in the early morning (around sunrise), when mixing heights were about 50 to 100 m and methane (CH_4) concentration in the plumes was at its highest. Wind direction was measured by releasing small helium-filled balloon at the start of each traverse, while wind speed was measured with a tethersonde, which was raised to about 100 m above the ground. Our measured data were supplemented with routine meteorological measurements that some of the mines carried out as part of their operations. The ability to carry out the traverse was limited by adequate access around the perimeter of the mine, and the timing of the vehicle traverses was subject to mine operational requirements.

Figure 1. Schematic of direct measurement of surface mine emissions using an air pollution technique (Saghafi and Williams, 1992; Williams et al., 1993)

If a rectangular plume with a width of *w* and a height of *h* is assumed, then the rate of gas emissions, *Q*, can be calculated as follows:

$$Q = chwv \tag{1}$$

where *c* is the average cross-wind CH_4 concentration, and *v* is the wind speed. The vehicle was driven crosswind along available public access roads in the proximity of the mine lease. CH_4 concentration was averaged using a Gaussian distribution function.

Global emissions measurements were undertaken using the technique illustrated in Figure 1 at ten mines of the Bowen Basin and seven mines of the Sydney Basin (Hunter Coalfield). CH_4 was detected from all 17 open cut mines visited. The concentrations of CH_4 due to mine emissions were generally limited to 0.05–0.40 ppm for the 10 Bowen Basin mines and 0.10 to 5.50 ppm above the background concentration for the seven Sydney Basin open cut mines.

Figure 2 shows the distribution of measured CH_4 emissions over the coal production for the ten Bowen Basin mines. These measurements were undertaken during the CSIRO field investigation campaign of 1990–92 in the Bowen Basin. Table 1 reports the raw coal production and measured emissions for these ten mines, which underwent the direct measurement of emissions. The visited mines produced 4.0 to 12.5 Mt of raw coal per year at the time of measurement (1992–93). Total coal production from these ten mines was approximately 68 Mt in the financial year 1991–92.

Table 1. Measured fugitive CH_4 emissions and calculated emission factors for 10 open cut mines in Queensland (Saghafi and Williams, 1992; Williams et al., 1993)

Figure 2. Distribution of measured fugitive CH_4 emissions for ten open cut coal mines in Queensland, Australia (modified from Saghafi and Williams, 1992; Saghafi, 2008)

Measured CH_4 emissions from these open cut mines varied largely from 1.5 to 25.4 million m^3 (Mm^3) per year, totalling about 78 Mm^3 for all 10 mines. As seen in Figure 2, no positive relationship between the volume of gas emitted and tonnage of coal produced is evident. The low-production coal mines have even larger emissions than the high-production coal mines; mines that produced less than 6 Mt of coal per year emitted the most gas. This shows that parameters other than coal production (such as gas content and local geological conditions) could have determined the magnitude of gas emissions from these mines. The *EF* for individual mines was calculated by dividing the measured emissions from plumes by mine production. The calculated *EF*s varied from ~0.1 to~4.6 m^3/t (in terms of CH_4 emissions). Subsequently, an average *EF* of 1.2 m^3/t was established (coal production weighted average) for the surface mines of the Bowen Basin. Table 1 gives details of data used to establish an individual and an average *EF* for Queensland open coal mines. A similar method was used to establish an average *EF* for surface mining in the Sydney Basin. Emissions from seven mines in the Hunter Coalfield, with a total production of more than 28 Mt (in 1992), were measured. The calculated *EF*s for these mines varied from 0.4 to8.7 m^3/t. Based on total emissions and total coal production, an average value of 3.2 m^3/t was established for the open cut mines of the Hunter Coalfield. This work led to the definition and quantification of two *EF*s for these coal basins.

The *EF* gas volumes correspond to CH_4 seam gas at atmospheric pressure and 20 °C conditions. In terms of CO_2-e emissions, the average *EF* for Queensland surface mines is 0.017 tonnes of CO_2 per tonne of run of mine (ROM) coal; for New South Wales, it is 0.045 tonnes CO_2 per tonne of ROM. These values were obtained by assuming that the global warming potential (GWP) of CH_4 is 21, and that its density is 0.6685 kg/m^3 at atmospheric pressure and 20 °C. These values were subsequently adopted by the NGER and were recommended for the estimation of emissions, under the titles of 'Method 1' or 'National average estimate' (National Greenhouse and Energy Reporting, 2009).

2.2 Spot emissions measurement method

Though global emissions from open cut coal mines can be obtained from air pollution techniques, this approach has been abandoned, as contribution from other sources – including underground mines in the region – could not be readily quantified. Furthermore, the air pollution methods proved to be costly, complex and time dependent. Specific environmental and metrological conditions were required for measurements to be taken, and the results were very sensitive to meteorological conditions, which affected assumptions of the origin of the emissions and the shape of the plume.

Consequently, direct mine emissions methods of measurements were designed and applied to coal mines. Methods such as local surface measurement of emissions were introduced. These methods used chamber techniques used previously in the study of spontaneous combustion of coal and spoils in open cut mines (see e.g. Carras et al., 2000). Though the results were promising (Saghafi et al., 1995), the techniques were found difficult to apply, mainly because of the requirements for staff and equipment to be present at the site of measurement. In many areas of a mine, access was prohibited, and safety or production requirements made spot measurements of emissions impossible. However, some important insights were gained through measuring gas emitted from uncovered and blasted coal.

3. Development of a new method for emissions estimation based on gas reservoir concepts

Since the early 2000s, as part of several studies (Saghafi et al., 2003; Saghafi et al., 2004; Saghafi, 2008) we have aimed to develop a Tier 3 method with a gas reservoir view of coal and strata in open cut mining.

In a reservoir view, the coal seams and surrounding strata are considered as gas reservoir units, which totally or partially emit all their gas during coal mining. This approach assumes that the total volume of emissions –including those from exploration boreholes, mining, spoil piles, and transportation and haulage of coal products – is equal to the volume of gas initially trapped in the reservoir. If a coal mine advances at a certain regular pace over its life, this approach assumes that emissions would be equal to the volume of gas contained in a reservoir 'slice' or column of strata of constant width (Figure 3). The column height is equal to the open cut depth (*h*: ground surface to the base of the mining) plus the depth of mining-induced fractured and disturbed zone below the base of the mine (δ*h*). The approach assumes that the sum of individual emissions, Q_1 to Q_n(Figure 3) produced from different sources is equal to or less than the volume of gas (*Q)* initially trapped in a column of strata above and below the base of the pit. This gas is released partially or totally, depending on mining conditions. This approach is mathematically expressed by:

$$Q_1 + Q_2 + Q_3 + Q_4 + \ldots \leq Q \qquad (2)$$

Figure 3. Schematic of sources of fugitive coal seam gas evolved from an open cut (strip mining in this diagram); total emissions from various sources are equal to or smaller than the volume of gas initially contained in the strata above and to a certain distance below the base of the mine (diagram not to scale)

In this approach, the reservoir slice from which all emitted gas is sourced is partitioned into various emission layers (gas-bearing layers), all of which constitute a 'gas release zone' (Figure 4). The extent of the gas release zone below the mine base is specific to the mine site and is affected by mechanical properties of the strata and mining method. The new method requires data on the in situ gas content of coal and rocks contained in the gas release zone. A simple way of identifying the various emission layers is to delineate them according to the type of material they contain. A refinement of the layering strategy is to subdivide them further according to their gas content and/or composition. Figure 4 identifies the emission layers based on the type of sedimentary unit from which they are made. Each layer is an individual gas reservoir with a specific emission regime. For instance, a coal seam and a shale layer are different emission layers. However, a gas-bearing coal layer can also contain thin bands of rock and other carbonaceous material. Similarly, a rock layer may contain thin bands of coal. The layers are characterised by their thickness, density and gas content and composition.

Figure 4. Conceptual model of gas emissions from open cut: gas is released from a vertical column, called the gas release zone, which includes the overburden and strata below the mine base. This column is divided into individual emission layers based on the nature of materials and their reservoir properties (diagram not to scale; black layers are coal, gray layers are rocks)

The new method consists of first calculating the emission from each layer and then integrating emissions from all layers. Emissions from an individual layer can be calculated as follows:

$$q_i = \beta_i c_i \rho_i h_i \tag{3}$$

where q_i is gas emission volume from layer i (m^3 of gas per m^2 of ground surface), and parameters c_i, ρ_i and h_i are gas content (m^3/t), density (t/m^3) and thickness of layer (m), respectively. In Eq. (3), β_i is the emission coefficient and varies between 0 and 1. It is introduced to account for the fact that not all of the contained gas is necessarily released. The value of β depends on the depth and reservoir properties of the layer. In general, it is plausible to assume that the overburden layers will release all their gas (β=1). However, the layers in underburden would only partially release their gas during mining (β<1). At depths larger than a given distance below the base (δh), one can assume that no gas would be released (β=0). The value of δh depends on the extent of the fractured zone and water table level below the pit floor. In the calculations that will be presented, δh is assumed to be about 20 m. If the water table is near the surface or pit base, this would cause β to vary sharply near the floor and be strongly non-linear in style. Figure 5 shows schematically the effect of the water table and the extent of fracturing of the ground below the pit floor on the shape of β. Function (1) corresponds to the case with little fracturing of the floor and/or high water saturation; function (3) corresponds to a highly fractured and/or drained water underburden; and function (2) corresponds to intermediate ground conditions. For simplicity, if a linear relationship with depth is used, then β is:

$$\beta(z) = 1 - \frac{z-h}{\delta h} \tag{4}$$

Figure 5. The emission coefficient β can be assumed to be a function of depth. In this example, β=1 for overburden layers. For the underburden, the shape of the function depends on the intensity of fracturing below the pit base and the water table level

Assuming that data on gas content, density and thickness are available, the emissions from each individual layer are calculated using Eq. (3). Then, all individual emissions are summed to deliver the total site-specific emissions:

$$Q = \sum_{1}^{n} q_i \tag{5}$$

Q has the units of volume of gas emitted per unit area of the ground (m^3/m^2), which can be called the 'surface emission factor' or 'emission density'. The latter term is used in this paper to express emissions in terms of the volume of gas released per unit of ground surface area. This is a useful way of quantifying emissions. Once an average value for the emission density in a mine lease or a gas zone is established, the total annual emissions can be estimated by multiplying Q by the area of ground to be mined over a year. If multiple Q values are available across the mine lease then a contour map of missions density can be established and used for emissions estimate.

3.1. Emission coefficient β and residual gas content

The emission coefficient, β, is also an indicator of the residual gas content of coal after mining. This is particularly useful in estimating the footprint of fugitive gas from coal transportation and haulage. If transportation emissions need to be separated from mine emissions, then an emissions coefficient of β<1 is used for the mined seams. For instance, if 10% of initial in situ gas remains in ROM materials, then β is 0.9 and is used in the model. The residual gas content and in situ gas content are related through parameter β by:

$$\frac{c_r}{c_i} = 1 - \beta \tag{6}$$

where c_r is the residual gas content and c_i is the initial virgin gas content of layer i. The residual gas content of mined coal can be directly measured using a standard gas content testing method; therefore, the footprint of coal transportation and haulage is quantified.

3.2. Emission density in terms of CO_2 equivalent

The gas content of various layers in the developed model is reported in m^3 per tonne of coal (m^3/t), and the mixed nature of CSG is presented by quantifying each component in terms of per cent. Hence, the calculation of gas emitted from a mine using the above model will produce emissions in terms of total volume per unit ground surface or emission density (m^3/m^2), as well as the percentage of the two main components of CSG, i.e. CO_2 and CH_4. For greenhouse gas inventory purposes, gas emissions are reported as CO_2-e. Therefore, the CH_4 emissions are converted to CO_2 using the GWP factor for CH_4, which is a measure of its global warming impact relative to CO_2. The value selected for GWP varies according to the time span chosen (usually 100 years), reflecting the lifetimes of CH_4, and CO_2 in the atmosphere (Climate Change, 1995). For the model proposed here, the GWP factor used is for 100 years time span. For CH_4, the GWP in terms of equivalent mass of CO_2 is 21 and in terms of equivalent volume of CO_2 is 8.36. For shallow coals, the seam gas mainly contains CO_2 and CH_4. For deeper coals, small volumes of ethane (C_2H_6) can be also present, and its effect on CO_2-e needs to be accounted for. The other component of the seam gas is nitrogen (N_2), for which GWP=0.

3.3. Estimate emission factor

EF is commonly used to express emissions from coal mines relative to coal production. It is the volume of gas emitted per tonne of coal produced. This factor can be calculated by extending the procedure for calculating emissions from individual layers in the gas release zone to coal production from these layers. An equation for the mass of coal produced from layer *i* is:

$$p_i = \alpha_i \rho_i h_i \tag{6}$$

The coefficient α, which is called the 'production coefficient', takes on a value of either 1.0 or 0. If the layer is a coal horizon and is mined, then α=1.0; otherwise, if the layer is spoiled, α=0. As for emissions, all individual productions are summed to estimate the total coal production per unit of ground surface area (t/m^2):

$$P = \sum p_i \tag{7}$$

Finally, the *EF* for the specific site would be:

$$EF = \frac{Q}{P} \tag{8}$$

This model is easily amenable to a spreadsheet calculation. The input data are gas content, composition, thickness, density and emission coefficient for the various layers. The output data are emission density and emission factor.

3.4 Effect of temporary stoppage of mining on gas emissions

The model takes into account temporary stoppage of mining. By using *EF*, gas emissions are proportional to production; if a pit is stopped temporarily, coal production reduces, consequently reducing the estimated emissions.

If mining is completely stopped, gas will still diffuse through strata and into the standing highwall at a diminishing rate. The applicability of the Tier 3 model stops at this stage as this model cannot be used for abandoned mines and rehabilitated terrain. Emissions would be of a different scale, and if still significant, must be dealt with using a different model.

4. Discussion on the uncertainty of fugitive gas emissions estimates

The basic equation used in this method is Eq. (3), which quantifies the emissions volume from a given sedimentary layer. As the mathematical expressions show, the data required are thickness and density of the layer and in situ gas content. The thickness and density of the layer can be readily – and relatively accurately – quantified from the analysis of the geological and geophysical logs, combined with coal quality data routinely produced for estimating coal reserves. However, determination of gas content and composition of gas-bearing layers are not usually included in routine exploration drilling programs, and therefore need to be measured for fugitive emissions calculations. As drilling and coring are cost-intensive activities only a small number of gas exploration boreholes could be drilled to determine the gas content and gas composition of gas-bearing units (coal seams and carbonaceous sedimentary layers). The gas content and composition are the most error prone input data for the model. An inter-laboratory comparison of gas content testing in Australia (Saghafi et al., 1998) showed that the variability in results between two individual laboratories can be as high as 30%, though in general it was below 15%. Moreover, the detectability limit of gas content for open cut coals with low gas content can introduce further errors of gas content. In cases where the gas content is to be estimated, the error would be even higher. We discuss a method of estimating uncertainty due to error of measurement or estimations of gas content in the next section of this paper (Section 4.1). For gas content determination we recommend that core drilling span the totality of the overburden certain depth of the underburden (approximately 20-25 m below the base of the pit). When such a core hole is drilled, the coal gas content should be measured using a direct method of such that the one described Standards Australia documents (Australian Standard, 1998).

In eastern Australia, the CSG of some coalfields contains large volumes of CO_2. For these areas, we suggest that gas content and composition be determined using the fast desorption method of gas content testing. This method is preferable because it more accurately estimates the CO_2 content of CSG (Saghafi, 1998). For carbonaceous material, the standard method of gas content testing can still be applied. For clastic rocks, however, measurement of gas content is almost impossible, because most gas is in the free phase. If necessary, the maximum gas content of these layers can be estimated using porosity data (measured or determined from geophysical logs) and assuming theoretical values for gas pressure and water saturation. At shallow depths, the density of free-phase gas is quite low, and the gas content of rock strata for open cut mines can be omitted. For example, at a depth of 100 m, the CH_4 content of rock strata with an available porosity of 1% (pores are otherwise filled with water) can reach a maximum of 0.04 m^3/t.

4.1 *Calculation of uncertainty of emissions estimate*

The Tier 3 method described in this paper estimates the total emissions from strata (Q) by summing the individual emissions (q_i) from individual gas-bearing layers. Assuming that the uncertainties of emissions for layers are independent of each other, the uncertainty of total emissions can be calculated by quadrature addition of each layer's uncertainties (for more details on error calculation, see metrology guides such as JCGM documents):

$$\delta Q = \sqrt{\sum \delta q_i^{\,2}} \tag{9}$$

where δQ is the uncertainty of emissions, and δq_i are the uncertainties associated with emissions from individual gas-bearing layers. Using Eq (3), the uncertainty of an individual layer's emissions can also be calculated by the quadrature addition of errors associated with each parameter used in Eq (3), i.e.:

$$\delta q_i = q_i \sqrt{(\frac{\delta c_i}{c_i})^2 + (\frac{\delta h_i}{h_i})^2 + (\frac{\delta \rho_i}{\rho_i})^2 + (\frac{\delta \beta_i}{\beta_i})^2} \tag{10}$$

Assuming that all errors, except error of gas content measurement, are negligible, then:

$$\delta q_i = \varepsilon_i q_i \tag{11}$$

where $\varepsilon_i = \delta c_i / c_i$ is the relative error of measurement of gas content for samples collected from gas-bearing layer i. The uncertainty of the total emissions from the column of strata can then be expressed in terms of gas content errors, i.e.:

$$\delta Q = \sqrt{\sum \varepsilon_i^{\,2} q_i^{\,2}} \tag{12}$$

If relative errors of gas contents for all gas-bearing layers were of similar magnitude, then Eq (12) is further reduced to:

$$\delta Q = \varepsilon \sqrt{\sum q_i^{\,2}} \tag{13}$$

where ε is an average relative error associated with the measurement of gas content. δQ is the uncertainty of emissions based on data from a single borehole; emissions densities and associated uncertainties from different boreholes would obviously differ and estimate of uncertainty of emissions from the whole mine lease may require further uncertainty calculation. This takes us to the concept of gas domains or gas zones, where it is assumed that an average gas emission density (Q) could be applied. This concept is explained in Section 5.

5. Gas zones and geological domains

In mines with complex geology and hydrological regimes, gas content can vary widely in space, and gas distribution in strata is largely location specific. Esterle et al. (2006) compiled gas contents of coal samples from some 1300 exploration boreholes drilled in 16 coal mines in Central Bowen Basin, and found that gas content as a function of depth varied substantially between the mines. Therefore, estimating emissions from the data measured or estimated from a single borehole may not accurately present emissions from the whole mine lease, because gas content may vary extensively across even a single mine lease where complex geology and hydrology is dominant. In an area with complex geology, a larger number of boreholes need to be drilled to accurately reflect mine emissions. In the new Tier 3 method, we call areas with similar gas distribution patterns 'gas zones'; the data from boreholes within a gas domain are used to estimate emissions for that domain. Similar concepts as the one presented here has been previously used by researchers to subdivide coalfields in domains of similar gas behaviour. For example, Williams et al. (2000) studied gas content variation in Queensland underground coal mines for safety and drainage purposes, and partitioned the mining area in number of 'gas domains'. They defined gas domains as 'areas where the gas content magnitude varies according to a set of defined conditions, i.e. gas content for the domain is a function of say depth and ash and the variability is tight enough to distinguish it from other domains'.

In the context of emissions from open cut mines, we first divide the mine lease into a number of 'geological domains' (based on local geology such a faults, structure and hydrology). Then, using any existing data on gas in the area, the geological domains can be subdivided into one or a number of 'gas zones'. In each gas zone, the gas content distribution within the strata is expected to follow similar patterns. The size of a gas zone depends on the geological and reservoir complexity of the area, but is generally in the order of a few square kilometres. One or two boreholes may be drilled in each gas zone to determine the main parameter of the model required for calculation of *EF*.

The main purpose of the gas zone concept is to enable the use of a single *EF* value within each zone. Therefore, the major identifiers of a gas zone are those properties that strongly affect the value of *EF*. The identifier can be gas content versus depth, if a dominant gas is present in seam gas. Figure 6 shows two gas zones based on the depth gradient of gas content where CH_4 is the dominant gas. Alternatively, gas zones can be delineated based on gas composition due to large differences in the GWP of CH_4 and CO_2 in CSG.

In low gas content mines, properties such as coal thickness or gas saturation can also delineate gas zones. Hence, the first step in evaluation of *EF* is to divide the mine lease into geological domains, and then build gas zones based on these domains.

Figure 6. A mine lease is divided into a number of gas zones, in which gas composition and gas content follow similar patterns with respect to depth

Where no virgin coal gas content data (from exploration boreholes) are available, measurement of gas composition and content of blasted coals can be useful for an initial delineation of gas zones. This can be supplemented by data on previous exploration drilling and hydrological tests (such as piezometry) within the lease. A gas zone can also be delineated vertically, giving a three-dimensional entity.

Once gas zones are identified, gas drilling could be undertaken to collect the necessary coal and rock samples. The samples should correspond to the layers in the gas release zone in the model. One or two core boreholes may be drilled in each gas zone. In practice, the gas exploration boreholes should be included in the routine coal exploration program, thereby generating data for both coal reserves and fugitive emissions calculations.

6. Example application of the new method to estimate emissions from a surface coal mine

As discussed in Section 5, the new Tier 3 approach can be applied to any surface mine, provided that some data on gas and the lithology of strata at the location are known. To reduce the number of drilling, the mine lease is compartmentalised into gas zones, based on existing geological, geophysical and gas data. An initial borehole is drilled and cored in each zone. Samples are collected from coal seams and carbonaceous sedimentary rocks and tested for gas content and composition. A second borehole is then drilled to confirm the delineation of the gas zone. If the data from the second borehole differ from that of the first borehole, then the boundary of the gas zone should be adjusted accordingly.

In this section, we discuss the application of the new method to an open cut coal mine in a coal region in eastern Australia. The rank of coal in this region is medium-to-high volatile bituminous, and it belongs to the Upper Permian geological age. The non-coal strata consist mainly of sandstone, with some siltstone and mudstone units. Some seams are divided into 'splits', in which an otherwise continuous coal seam separates into two layers, enclosing a discrete 'pocket' of mineral matter. To obtain accurate data to apply the new method, samples were collected from all coal seams and their splits, and their carbonaceous and clastic rocks. The procedure of drilling, sampling and measuring, and the results, are presented in detail in the next sections.

6.1. Gas drilling and core samples

Two boreholes were drilled and cored in a gas zone in shallow strata of up to ~230 m. The CSG was rich in CH_4, with small volumes of CO_2 in seams closer to the ground surface. The two boreholes were drilled approximately 350 m apart. Seven main seams were intercepted by the drilling. However, nine coal horizons were traversed by the drilling, including coal seams and their splits. A total of 84 core samples were collected from Borehole 1, with 37 samples from coal layers and 47 samples from rock horizons, including sandstone, siltstone shale and carbonaceous materials. As we expected that gas behaviour would be relatively similar within the gas zone, Borehole 2 was mainly chip drilled in clastic horizons and was fully core drilled in coal horizons. A total of 27 core samples were collected, which also included four non-coal samples.

Any sample with a density below 1.9 m^3/t was considered coal. The ash yield of the coal samples varied from 3.6–69.8%. In terms of dry ash free, the volatile matter content of the coal samples varied from 29.0–39.2%.

6.2. Gas content measurement of coal and non-coal materials

The gas content of core samples was measured using either a rapid crush method or fast desorption method (Williams et al., 1992; Saghafi et al., 1998; Australian Guide, 1999). This method consists of three stages, termed Q_1, Q_2 and Q_3. In each stage, part of the gas contained in coal is liberated and measured.

Gas lost during drilling (Q_1 component) is estimated by measuring the gas desorption rate in the field on fresh core samples immediately after being available at the surface. The rate of gas desorption from the sample is measured by monitoring the displacement of the water in an inverse measuring cylinder over a period of 20 to 30 minutes, which is about the time taken to drill the individual cores and bring them to the surface. The other two components of gas content are Q_2 and Q_3. Q_2 is the volume of gas desorbed after the coal is sealed in a gas-tight canister in the field following Q_1, until the coal is removed from the canister and crushed to liberate its remaining gas. Q_3 is the volume of gas released during crushing. Measurements of gas composition are also required at the Q_2 and Q_3 stages, as seam gas – particularly at shallow depths – can be rich in CO_2. Measurements of gas composition in the field for Q_1 are not undertaken, because the gas volume released is small; also, using a gas chromatograph in the drilling site requires complex logistics, so accurate results are not always obtained. The 'measured gas content', Q_m, is defined as the sum of the three components:

$$Q_m = Q_1 + Q_2 + Q_3 \tag{14}$$

All coal and non-coal samples were measured for Q_1 and Q_2 components of gas content. However, crushing sandstone and siltstone samples for Q_3 component measurement was impracticable.

6.2.1 Gas content of coal, Borehole 1

Nine coal layers (coal seams and their splits) were crossed by drilling and sampled in Borehole 1. Figure 7 plots gas content against depth for the coal samples. The measured gas content for the coal samples varies from 0.3 to 9.2 m^3/t. Measurements of the composition of desorbed gas showed that seam gas in this area consists of almost pure CH_4 (with less than 1% CO_2), regardless of the depth of the coal sample. The composition of seam gas has a large impact on the emission inventory, considering that the GWP of CH_4 is 21 times higher than that of CO_2 (although this value can be changed by the IPCC and can be higher).

Figure 7. Measured gas content of coal core samples against depth. The rectangles show the position of coal seams

6.2.2 Gas content of non-coal sedimentary rocks, Borehole 1

In general, sandstone samples showed no release of gas when they were placed in sealed canisters. This is presumably because any gas held in these samples would have been released as soon as the drill reached the sample, and during retrieval and transfer of the rock to the measuring canisters. The carbonaceous interburden sedimentary rocks, however, showed non-negligible amounts of gas, although much less than that of coal from a similar depth (<0.1 m^3/t). Figure 8 shows the results of measuring gas content of interburden samples as a function of depth for Borehole 1. For rock materials in which non-zero gas content was measured, the samples contained either carbonaceous shale or small amounts of coal and coaly shale materials, and were in close proximity to a coal seam (roof or floor). The gas composition of the interburden, as for coal, consisted mainly of pure CH_4 (<1% CO_2).

Figure 8. Gas content of interburden core samples (carbonaceous rock layers). The rectangles show the position of coal seams

6.2.3 Gas content of coal, Borehole 2

As discussed in Section 5, to establish the extent of the gas zone, a second borehole (Borehole 2) was drilled, approximately 350 m away from Borehole 1. In Borehole 2, most drilling was not followed by coring. All coal horizons were, however, cored and sampled, an approach supported by the very low gas contents of non-coal strata found in Borehole 1. Figure 9 shows the distribution of gas content against depth for core coal samples from Borehole 2. In total, as for Borehole 1, nine coal horizons were also sampled in Borehole 2.

Figure 9. Measured gas content of coal core samples against depth. The rectangles show the position of coal horizons crossed by Borehole 2

6.3. Gas zone and gas content pattern

To verify whether the two boreholes belonged to the same gas zone, we compared the gas distribution patterns using gas content data from both boreholes. One way of quantifying the similarity of the patterns is to use the average gas content of each coal seam from each hole and then try to fit a curve through the aggregated gas content data from both boreholes. Table 2 presents the two sets of gas content data from the two boreholes. For coal seams where multiple gas contents were measured, minimum, maximum and average gas content is reported. Average gas content is simply the arithmetical average of measured gas content values. The standard deviation is calculated when four or more cores across a single seam are measured for their gas content. Standard deviation for the average gas content is calculated by taking the square root of variance divided by the number of samples measured. These data are also illustrated in Figure 10, which shows the average gas content against depth.

Table 2. Gas content of coal samples from coal seams traversed by Borehole 1 and Borehole 2

Figure 10. Average gas content of the main coal seams intersected by Boreholes 1 and 2; error bars correspond to plus or minus one standard deviation.

Some rules can be established to indicate whether or not the data are from a single gas zone. For example, if the coefficient of determination (r^2) is larger than an agreed limit and/or the standard deviation is lower than an agreed limit, then one can consider that the two boreholes are drilled in the same gas zone.

For the data in Figure 10, a linear relationship between gas content and depth fits the data:

$$c = 0.046h - 1.81 \tag{15}$$

where c is the average gas content (m^3/t) at a seam depth of h (m). The coefficient of determination for this equation is 0.96 and the standard deviation is 0.59 m^3/t. This presents a strong correlation between the gas content and depth for the mixed data from the two holes. These data indicate that similar patterns of gas distribution exist in the two locations if they satisfy the agreed set of thresholds for coefficient of correlation and standard deviation; e.g. if the standard deviation (σ) is used as a measure of the uncertainty of gas content and for delineation of the gas zone. For example, if a threshold of σ=0.7 m^3/t is accepted, then Eq. (15) can be used to evaluate the gas content for emissions calculation in all locations in that gas zone. Various factors could be used to establish a mine-specific threshold for gas content uncertainty. This threshold could be based on the uncertainty of the gas content testing system available to the mine. For instance, some commercial laboratories in Australia use a limit of detectability of 0.3–0.7 m^3/t. This constraint tohether with the cost (drilling, coring, sampling and testing) could be used to establish a threshold.

For shale and carbonaceous materials (Figure 8), there was no clear relationship between gas content and depth. However, a more detailed study could analyse the relationship between organic matter content and gas content for these materials.

6.4 Calculation of emissions

In this section, we calculate the emission density and emission factor for the area represented by Borehole 1 using the procedure described in Section 3. We have used the data from Borehole 1 for emissions calculation in this area, because it had a more complete set of data than Borehole 2.

As described in Section 3, we divide the ground strata at location of Borehole 1 into a number of emission layers, each of which is characterised by its gas content, density and thickness. When calculating emissions, we have considered only those layers with a gas content of 0.01 m^3/t or higher (0.01 m^3/t is the limit of gas content detectability of the equipment and method we used). The emissions from each layer are then calculated using Eq. (3). For the coefficient of emission, β, we have assigned a value of 1.0 for all overburden layers (all gas contained in these layers is released during mining), and a value between zero and 1.0 for layers below the base of the mine. β is 1.0 for the floor of the mine, and zero for a location 25 m below the floor in the underburden. Table 3 shows the spreadsheet calculation of the emissions based on the tier 3 model for the data represented by Borehole 1. We assumed that the base of the mine would be located at a depth of ~117 m. We also assumed that non-carbonaceous rocks do not contribute to the emissions, or that their contributions are insignificant. Table 3 demonstrates that the Tier 3 model is readily amenable to a spreadsheet calculation. The emission density for the area is the sum of all entities in column q_i, which gives a total emission density of Q=29.05 m^3/m^2. Emission density can be also presented as a function of depth to identify areas of high emissions. In Figure 11, the cumulative emission density ($\sum q_i$) is plotted against the depth, which shows that the maximum emissions are produced from the gas-bearing layer between 108 and 117 m (steepest gradient of the curve). As expected, the gradient of emission density for underburden strata is very small.

Table 3. Application of the Tier 3 emission model to evaluate emission factors for a mining area characterised by the data from Borehole 1

Figure 11. Cumulated emission density ($\sum q_i$) as a function of depth. Emissions from individual layers (q_i) are added to each depth to quantify the increase in emissions as depth of mining increases.

Uncertainty of estimate of emission density

The uncertainty of emissions estimations can be calculated using Equations 9 to 13 in Section 4.1. We assume that gas content measurement/estimate is the main source of errors, and that the relative error of measurement of gas content increases as the value of measured gas content reduces. Typical relative errors for measurement of gas content are assumed to be as follows: for gas contents greater than 1.0 m^3/t, the relative error of measurement is 20%; for gas contents ranging between 0.5 and 1.0 m^3/t, the relative error is 30%; for gas contents between 0.1 and 0.5, the relative error is 40%; and for gas contents smaller than 0.1 m^3/t, the relative error is 60%. In Table 3, the column identified by δq_i contains the error (uncertainty) for the estimate of emissions from individual gas-bearing layers. Using equations presented in Section 4.1, the uncertainty of emissions from mining in area characterised by Borehole 1 would be δQ=1.73 m^3/m^2. Therefore, emission density, based on data from Borehole 1, is: Q=29.05±1.73 (m^3/m^2). This value of emissions can be used for an area of mining (a gas zone) for which the use of gas content and other data from Borehole 1 is justified.

Emissions factor based on data from Borehole 1

Coal production from a column of strata is calculated by summing all individual p_i values from the various emission layers (the last column of Table 3). This yields a total coal production of P=9.99 tonnes per m^2 of the ground surface. EF is then calculated according to Eq. (8), which gives EF=2.91 m^3/t. Accounting for the uncertainty of emissions, EF can be expressed as: EF= 2.91±0.17 (m^3/t).

To appreciate the differential contribution of coal and carbonaceous rock layers in total emissions, the emissions densities (volume of gas emitted per unit ground surface) for gas-bearing layers are integrated for overburden and underburden and are shown in Table 4. The data in this table show that for this case, 3% of total 29.05 m^3/m^2 emissions is sourced from underburden. Emissions from coal layers in overburden are 93% of total overburden emissions, with the remaining 7% from non-coal carbonaceous rocks. In contrast to overburden, emissions from coal layers in underburden are only 24% of the total underburden emissions, and the remaining 76% are from carbonaceous rocks. The main reason for the predominance of carbonaceous rock emissions in underburden is their close distance to the base of the mining, whereas coal layers are much deeper below the base of the open pit.

Finally, we need to express gas emissions in terms of CO_2-equivalent volume. Assuming that CH_4 GWP in terms of equivalent volumes of CO_2 is 8.36, and that the CH_4 concentration in CSG at Borehole 1 is ~98%, then the *EF* for this location in terms of CO_2-e would be: $EF_{CO2\text{-}e}$ = (1-0.99)*2.91*8.36 + 0.01*2.91 = 24.10 m^3/t.

Table 4. Contribution of coal and rocks in overburden and underburden to potential emissions from mining (based on Tier 3 model) for a mining area characterised by the data from Borehole 1

7. Cost of applying the Tier 3 method to open cut mining

One main concern that may be raised by some mine operators is the cost of using the Tier 3 method. The Tier 1 or 2 methods did not require any measurements, and emissions were estimated using recommended *EFs* for the different coal basins in Australia. The new method, as outlined in this paper, is location specific. It therefore requires drilling, coring and gas testing at a number of locations over the mine lease. The real advantage of this method – contrary to other methods – is that it enables mine operators to accurately estimate the volumes of gas emitted from their mining operation over the mine life. This means that when the tax on carbon is in place, the cost to the mines could be correctly evaluated. In contrast, using a coal basin-based average *EF* would penalise many mines that have small emissions. Another benefit of this method is that the distribution of *EF* can be plotted (on a contour map) over the mine lease, showing areas of low and high *EFs*. Coal mine operators may then decide to pre-drain areas with high emission densities.

As discussed in Section 5, in mines with complex geology, the spatial variation of gas content and composition – as well as the coal thickness – may vary largely. As discussed in Section 5, an efficient strategy to reduce the amount of drilling and sampling needed in these conditions is to divide the mine lease into gas zones that are expected to have similar gas content and composition patterns. Depending on the size and accuracy of the available data, this approach can significantly reduce the cost of using the new method.

In areas where the splits are thin, several thin sections can be used to form a single sample for measurement of gas content and composition. Furthermore, for thin interburden seams enclosed between two thicker seams, the gas content and composition of the interburden seams can be estimated by interpolating the data from the adjacent, thicker seams.

For locations with low gas content (<0.5 m^3/t), the cost of gas measurements could be further reduced by using some of less variable data from previous holes. Finally, the cost of gas drilling can be largely reduced by including it in routine exploration drilling. For example, in one of the drilling operations for emissions determination using this method, combining gas drilling with routine exploration drilling reduced the cost by approximately 40%.

8. Conclusions

This paper proposes a Tier 3 model for estimating CSG emissions from surface coal mining. The new model is based on gas reservoir concepts. It assumes that gas adsorbed and trapped in strata is released during mining, either totally or partially. Gas is emitted from a 'gas release zone', which consists of all gas-bearing horizons in the overburden, and in a section of the underburden to approximately 20–25 m below the mine base. An important concept in the new model is that of the emission layers that make up a reservoir slice. Each layer is an individual gas-bearing unit with a specific emission regime. The layers are characterised by their thickness, density, and gas content and composition.

The main inputs to the Tier 3 model are gas content and gas composition of the coal seams and carbonaceous rocks in the overburden and the mining-affected section of the underburden. The outputs of the model are gas emission factor, expressed in terms of m^3 of gas released per tonne of coal extracted, and/or gas emission density, expressed in terms of m^3 of gas released per m^2 of ground surface.

In view of the spatial variation in gas content at shallow depths of open cut mining, gas content and composition data are determined by direct desorption measurement of core samples collected from a small amount of core drilling in the lease before mining. The number of the boreholes to drill is reduced by partitioning the mine lease into a number of 'gas zones', in which the local geology, strata layout and hydrology predict similar gas emission behaviour. To reduce the cost of drilling further, gas boreholes are undertaken as part of the mine routine exploration program, with the cost of drilling and coring shared between the two operations.

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10. References

Australian Coal Association, 2009. The Australian Coal Industry – Coal Production. Available online at <http://www.australiancoal.com.au/>.

Australian National Greenhouse Accounts, 2011. National greenhouse gas inventory accounting for the Kyoto target, December Quarter 2010. Published by Australian Department of Climate Change and Energy Efficiency. 36 pp. Also available online at <http://www.climatechange.gov.au/>.

Australian Standard, AS 3980-1999, 1999. Guide to the determination of gas content of coal seams – Direct Desorption Method. Published by Standards Australia.

Boxho, J., Stassen, P., Mucke, G., Noack, K., Jeger, C., Lescher, L., Browning, E. J., Dunmore, R., Morris, I.H., 1980. Firedamp drainage handbook for the coal mining industry in the European Community. Coal Directorate of the Commission of the European Communities. Verlag Gluckauf, Essen.

Carras, J.N., Day, S., Saghafi, A., Williams, D.J., 2000. Measurement of greenhouse gas emissions from spontaneous combustion in open cut coal mining. ACARP project C8059. 49 pp.

Climate Change, 1995. The Science of Climate Change: Summary for Policymakers and Technical Summary of the Working Group I Report, p. 22, also available at: http://unfccc.int/ghg_data/items/3825.php

Creedy, D.P., 1993. Methane emissions from coal-related sources. *Chemosphere* 26, 419–439.

Creedy, D.P., Saghafi, A., Lama, R.D., 1997. Gas control in underground coal mining. IEA CR/91. London, UK. International Energy Agency (IEA) Coal Research. 120 pp.

Esterle, J., Williams, R., Sliwa, R. , Malone, M., 2006. Variability of coal seam gas contents that impacts on fugitive gas emissions estimations for Australian black coals. In: Proceedings of the 36 Sydney Basin Symposium, 27–29 November 2006, University of Wollongong, Australia, pp. 213–222.

Joint Committee for Guides in Metrology (2008). JCGM 100:2008 Evaluation of measurement data – Guide to the expression of uncertainty in measurement. Joint Committee for Guides in Metrology. Document produced by Working Group 1 (JCGM/WG 1), 120 pp.

Intergovernmental Panel on Climate Change (IPCC), 2006. IPCC Guidelines for National Greenhouse Gas Inventories. Volume 2: Energy. National Greenhouse Gas Inventories Programme Publications. Also available online at <http://www.ipcc-nggip.iges.or.jp/public/2006gl/vol2.html>.

Kissell, F.N., McCulloch, C.M., Elder, C.H. 1973. The direct method of determining methane content of coalbeds for ventilation design, US Department of the Interior, Bureau of Mines RI 7767, National Technical Information Service No. PB221628.

National Greenhouse and Energy Reporting (NGER) Act 2007, 2009. National Greenhouse and Energy Reporting (Measurement) Determination 2008. Prepared by the Office of Legislative Drafting and Publishing, Attorney-General’s Department, Canberra. 238 pp. Also available online at <http://www.comlaw.gov.au/Details/F2009C00576>.

Resources and Energy Statistics, 2012. Bureau of Resources and Energy Economics, Australian Government, Vol 2, No 3, December quarter 2011.

Saghafi, A., Williams, D. J., 1992. Estimation of methane emission from Australian coal mines. In: *Symposium on Coalbed Methane Research and Development in Australia.* Townsville, Australia. Vol. 5, pp. 7–13.

Saghafi, A., Williams, D.J., Lama, R.D., 1997. Worldwide methane emissions from underground coal mining. In: Ramani RV (ed.) *Proceedings of the Sixth International Mine Ventilation Congress*, May 1997. Pittsburgh, PA, US. Society for Mining, Metallurgy, and Exploration, Inc., pp. 441–445.

Saghafi, A., Williams, D.J., Battino, S., 1998. Accuracy of measurement of gas content of coal using rapid crushing techniques. In: *Proceedings of the 1st Australian Coal Operators Conference COAL'98,* February 1998. Wollongong, Australia, pp. 551–559. Also available online at <http://ro.uow.edu.au/coal/273/>.

Saghafi, A., Day, S., Williams, D.J., Roberts, D.B., Quintanar, A., Carras, J.N., 2003. Toward the development of an improved methodology for estimating fugitive seam gas emissions from open cut mining. Australian Coal Association Research Program Project C9063, 49 pp.

Saghafi, A., Day, S.J., Fry, R., Quintanar, A., Roberts, D., Williams, D.J., Carras, J.N., 2004. Development of an improved methodology for estimation of fugitive seam gas emissions from open cut mining. Australian Coal Association Research Program Project C12072, 55 pp.

Saghafi, A., Day, S.J., Carras, J.N., 2005. Gas properties of shallow Bowen Basin coal seams and gas leaks to the atmosphere. In: Beeston JW (ed.) *Bowen Basin Symposium 2005: The Future for Coal: Fuel for Thought: Proceedings, Yeppoon, Qld: Geological Society of Australia Coal Geology Group and the Bowen Basin Geologists Group*, October 2005. Queensland, Australia. pp. 267–271.

Saghafi, A., Williams, D. J., Battino, S., 1998. Accuracy of measurement of gas content of coal using rapid crushing techniques. In: Proceedings of the 1st Australian Coal Operators Conference COAL'98, Wollongong, 18–20 February 1998, Australia, pp. 551–559, also available online at <http://outburst.uow.edu.au/presentations_publications/coal_1998/Saghafi-1998.pdf>.

Williams, D.J., Saghafi, A., Drummond, D.B., Roberts, D.B. 1992. Development of a new equipment for rapid determination of coal gas content. In: *Symposium on Coalbed Methane Research and Development in Australia.* Townsville, Australia. Vol. 4, pp. 21–30.

Williams, R, Casey, D., Yurakov, E, 2000. Gas reservoir properties for mine gas emission assessment, Bowen Basin Symposium 2000, The New Millennium – Geology Proceedings, Rockhampton, Queensland, Australia, 22–24 October 2000, pp. 325–333.

Table 1. Measured fugitive CH_4 emissions and calculated emission factors for 10 open cut mines in Queensland (Saghafi and Williams, 1992; Williams et al., 1993)

Mine	**Coal production (Mt/y)**	**CH_4 emissions (Mm^3/y)**	**Emission factor, *EF* (m^3/t)**
Mine 1	12.5	1.5	0.1
Mine 2	9.5	3.0	0.3
Mine 3	7.0	3.0	0.4
Mine 4	4.9	3.0	0.6
Mine 5	5.5	3.0	0.6
Mine 6	6.0	25.4	4.2
Mine 7	4.2	19.3	4.6
Mine 8	8.5	1.5	0.2
Mine 9	3.9	6.1	1.6
Mine 10	5.4	12.2	2.3
Total	67.4	78.2	
		Weighted average *EF*(m^3/t) =	1.2

Table 2. Gas content of coal samples from coal seams traversed by Borehole 1 and Borehole 2

	Borehole 1						Borehole 2					
	Core sample		Gas content (m^3/t)				Core sample		Gas content (m^3/t)			
Coal seam	Depth (m)	N of core samples	Min	Max	Average	Std deviation	Depth (m)	N of core samples	Min	Max	Average	Std deviation
1	44.1	1	0.26	0.26	0.26		35.4	1	0.00	0.00	0.00	
2	68.8	2	1.35	1.45	1.40	0.07	56.3	1	0.51	0.51	0.51	
3	113.0	8	1.90	4.81	3.51	1.14	114.4	4	1.23	2.78	1.92	0.66
4	142.8	8	4.64	6.79	5.82	0.76	139.0	4	3.44	6.47	5.11	1.13
5	173.0	4	3.72	7.94	5.93	1.57	173.7	4	3.76	7.89	6.00	1.89
6	201.7	6	5.74	9.21	7.53	1.32	195.8	4	5.09	9.24	7.21	2.02
7	220.1	4	5.48	9.08	7.26	1.42	211.4	4	7.11	9.85	8.47	1.32

Table 3. Application of the Tier 3 emission model to evaluate emission factors for a mining area characterised by the data from Borehole 1

	Emission (gas bearing) layers			Emission model input: layer thickness (h_i), gas content (c_i), density (ρ_i), emission coefficient (α_i), production coefficient (β_i)					Emission model output: layer emission (q_i), uncertainty (δq_i) and production (p_i)		
	Layer	Layer type	Depth (m)	hi (m)	c_i (m^3/t)	ρi (t/m^3)	α_i	β_i	q_i (m^3/m^2)	p_i (t)	δq_i
Overburden	1	Carb sandstone	30.8	0.60	0.01	2.30	0	1.00	0.02	0.00	0.01
	2	Carb sandstone	32.4	0.99	0.04	2.30	0	1.00	0.09	0.00	0.05
	3	Carb sandstone	38.4	0.86	0.12	2.30	0	1.00	0.24	0.00	0.09
	4	Coal	43.7	0.57	0.26	1.49	0	1.00	0.22	0.00	0.09
	5	Carb siltstone	44.6	0.23	0.01	2.30	0	1.00	0.01	0.00	0.00
	6	Carb siltstone	45.4	0.77	0.03	2.30	0	1.00	0.05	0.00	0.03
	7	Carb siltstone	48.1	0.91	0.02	2.30	0	1.00	0.04	0.00	0.03
	8	Carb siltstone	63.5	0.57	0.01	2.30	0	1.00	0.02	0.00	0.01
	9	Carb shale	67.7	0.58	0.04	2.30	0	1.00	0.06	0.00	0.03
	10	Coal	68.4	0.60	1.45	1.49	1	1.00	1.29	0.89	0.26
	11	Coal	69.1	0.91	1.35	1.50	1	1.00	1.84	1.36	0.37
	12	Carb siltstone	70.6	0.86	0.05	2.30	0	1.00	0.09	0.00	0.05
	13	Carb sandstone	76.9	0.54	0.05	2.30	0	1.00	0.06	0.00	0.03
	14	Coal	77.5	0.56	1.65	1.65	1	1.00	1.52	0.92	0.30
	15	Coal	85.1	0.58	1.68	1.64	1	1.00	1.60	0.95	0.32
	16	Carb siltstone	101.5	0.30	0.07	2.30	0	1.00	0.04	0.00	0.03
	17	Carb siltstone	108.4	0.31	0.36	2.30	0	1.00	0.26	0.00	0.10
	18	Coal	108.9	0.76	4.81	1.39	1	1.00	5.08	1.06	1.02
	19	Carb siltstone	109.5	0.27	0.66	2.30	0	1.00	0.41	0.00	0.12
	20	Coal	109.7	0.17	4.61	1.59	1	1.00	1.25	0.27	0.25
	21	Coal	110.2	0.59	4.76	1.35	1	1.00	3.79	0.80	0.76
	22	Coal	110.7	0.34	3.77	1.36	1	1.00	1.74	0.46	0.35
	23	Carb siltstone	111.1	0.40	0.08	2.30	0	1.00	0.08	0.00	0.05
	24	Coal	111.5	0.32	1.90	1.66	1	1.00	1.01	0.53	0.20
	25	Coal	115.5	0.96	2.06	1.44	1	1.00	2.85	1.38	0.57
	26	Carb siltstone	116.2	0.51	0.56	2.30	0	1.00	0.66	0.00	0.20
	27	Coal	116.7	0.31	3.75	1.36	1	1.00	1.58	0.42	0.32
	28	Coal	117.1	0.54	2.44	1.73	1	1.00	2.28	0.93	0.46
Underburden	29	Carb siltstone	122.4	0.99	0.30	2.30	0	0.79	0.53	0.00	0.21
	30	Carb siltstone	128.4	0.89	0.06	2.30	0	0.55	0.07	0.00	0.04
	31	Carb sandstone	130.2	0.96	0.03	2.30	0	0.48	0.03	0.00	0.02
	32	Carb siltstone	140.6	0.49	0.59	2.30	0	0.06	0.04	0.00	0.01
	33	Coal	141.2	0.56	4.62	1.38	0	0.04	0.14	0.00	0.03
	34	Coal	141.9	0.98	6.79	1.35	0	0.01	0.07	0.00	0.01
	35	Coal	142.7	0.63	4.64	1.40	0	0.00	0.00	0.00	0.00
	36	Coal	143.2	0.20	6.18	1.35	0	0.00	0.00	0.00	0.00
	37	Coal	143.6	0.61	6.59	1.36	0	0.00	0.00	0.00	0.00
	38	Coal	144.2	0.61	5.96	1.37	0	0.00	0.00	0.00	0.00
	39	Coal	144.6	0.28	4.93	1.46	0	0.00	0.00	0.00	0.00

	40	Coal	144.9	0.25	5.64	1.39	0	0.00	0.00	0.00	0.00
	41	Coaly siltstone	145.2	0.26	1.22	1.92	0	0.00	0.00	0.00	0.00

Table 4. Contribution of coal and rocks in overburden and underburden to potential emissions from mining (based on Tier 3 model) for a mining area characterised by the data from Borehole 1

Strata	Gas bearing sedimentary rocks	Total thickness (m)	Emission density (m^3 gas per m^2 of ground surface)
Overburden	Coal	7.21	26.06
	Carbonaceous rocks	8.70	2.11
	Total overburden	15.91	28.17
Underburden	Coal	4.12	0.21
	Carbonaceous rocks	3.59	0.67
	Total underburden	7.71	0.88

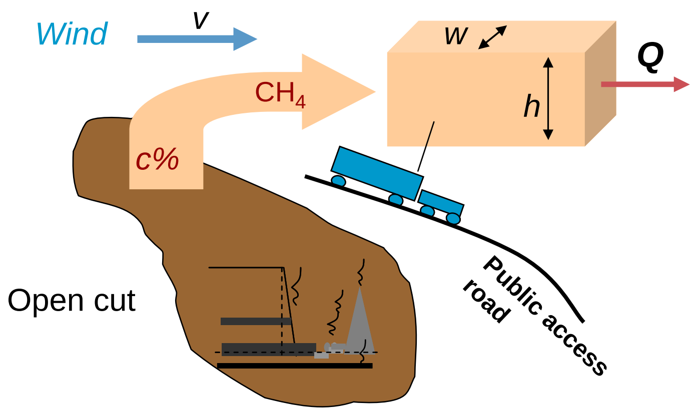

Figure 1. Schematic of direct measurement of surface mine emissions using an air pollution technique (Saghafi and Williams, 1992; Williams et al., 1993)

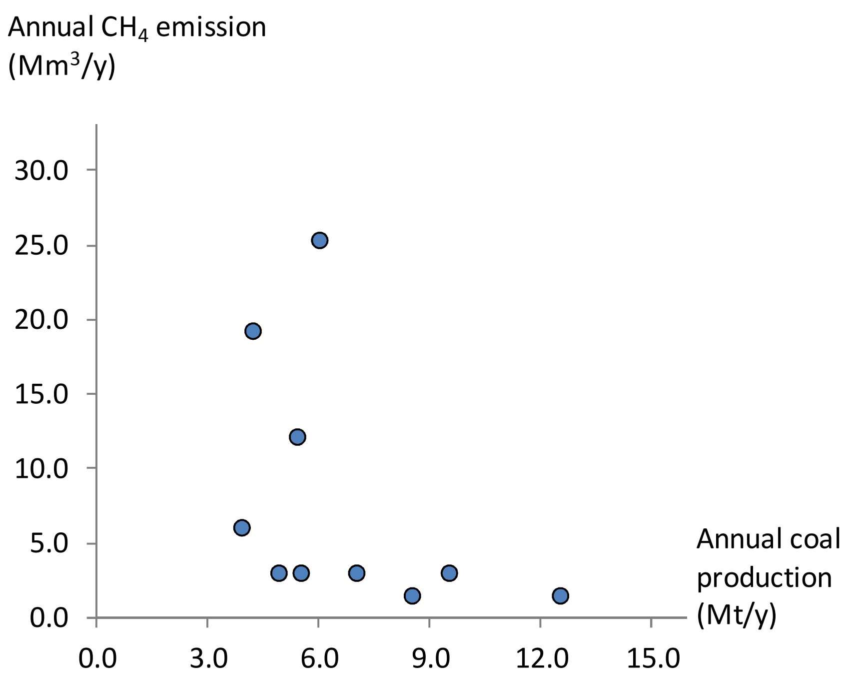

Figure 2. Distribution of measured fugitive CH_4 emissions for ten open cut coal mines in Queensland, Australia (modified from Saghafi and Williams, 1992; Saghafi, 2008)

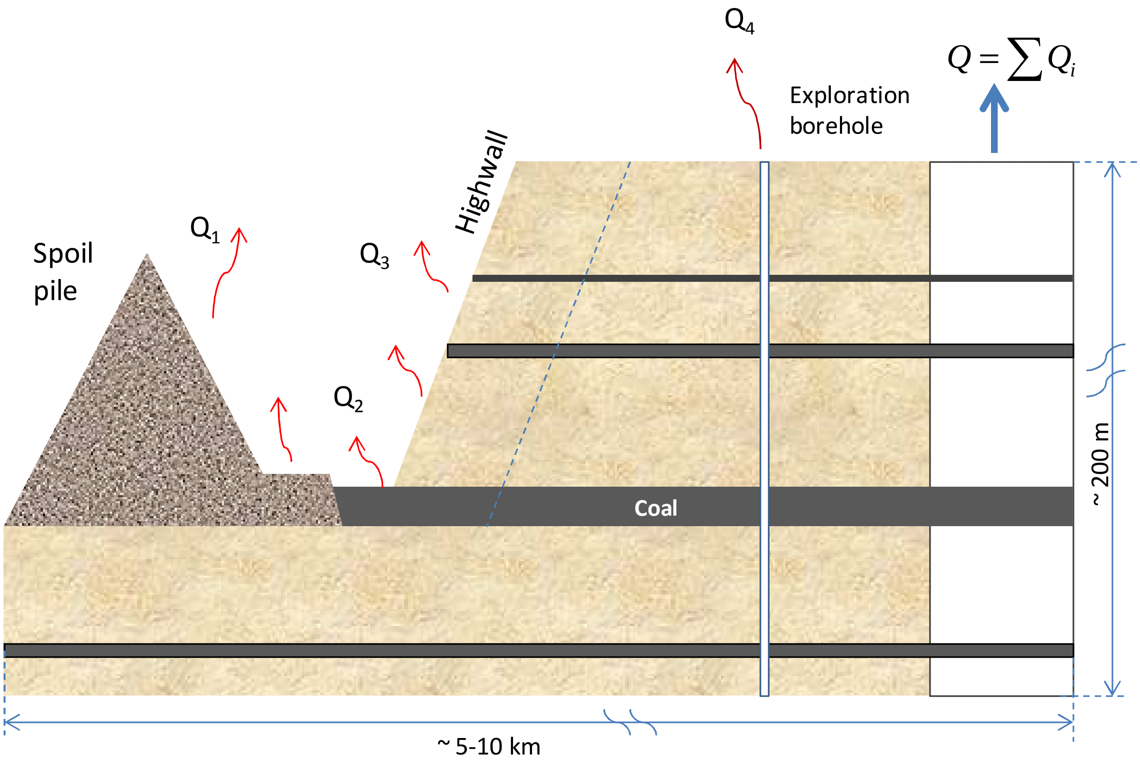

Figure 3. Schematic of sources of fugitive coal seam gas evolved from an open cut (strip mining in this diagram); total emissions from various sources are equal to or smaller than the volume of gas initially contained in the strata above and to a certain distance below the base of the mine (diagram not to scale)

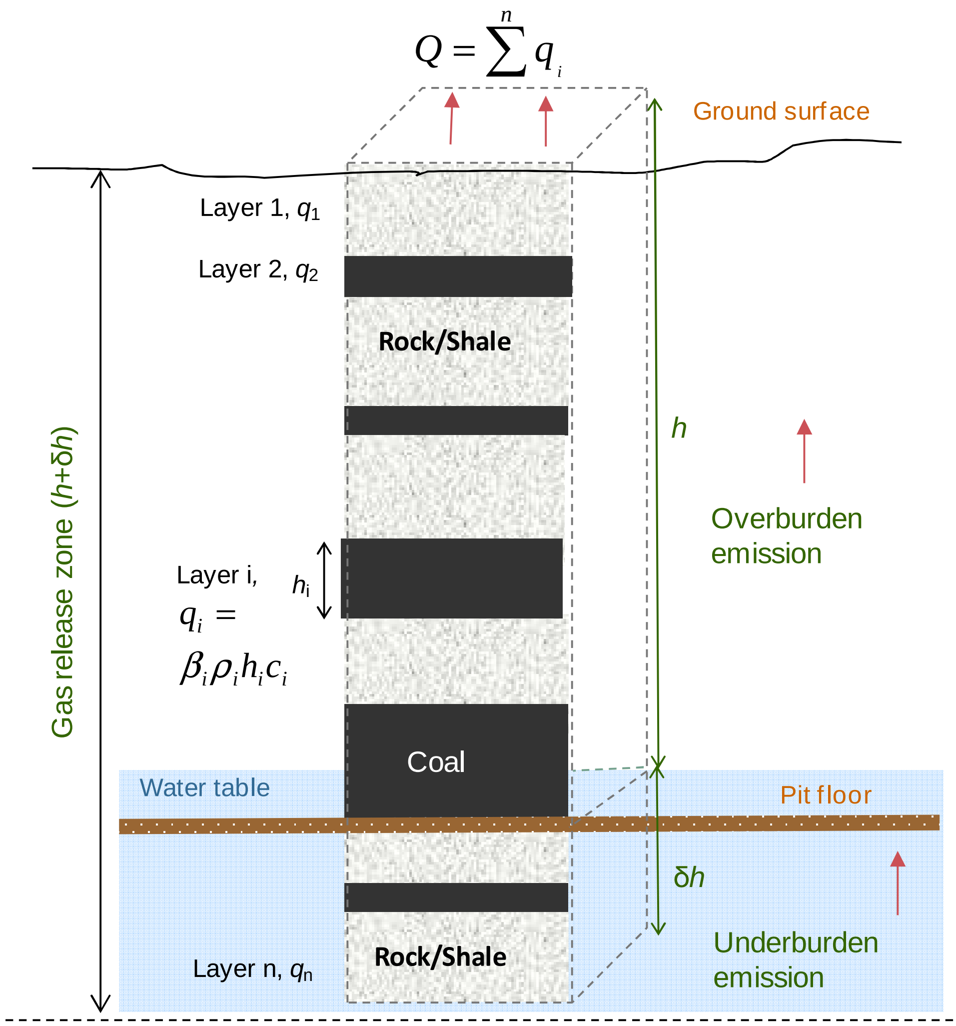

Figure 4. Conceptual model of gas emissions from open cut: gas is released from a vertical column, called the gas release zone, which includes the overburden and strata below the mine base. This column is divided into individual emission layers based on the nature of materials and their reservoir properties (diagram not to scale, black layers are coal; gray layers are rocks).

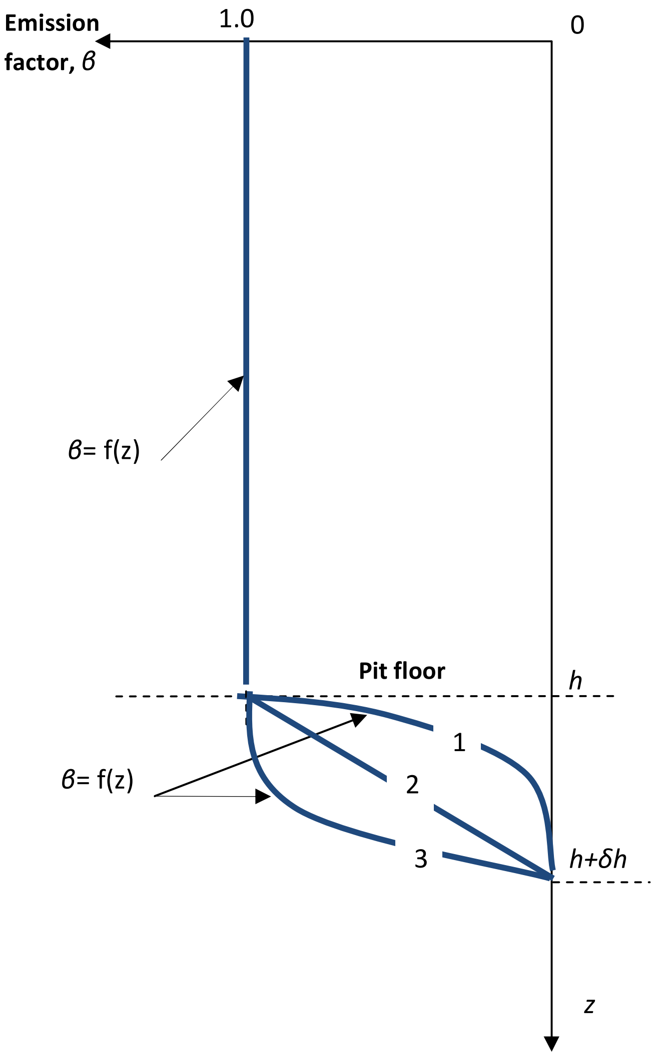

Figure 5. The emission coefficient β can be assumed to be a function of depth. In this example, β =1 for overburden layers. For the underburden, the shape of the function depends on the intensity of fracturing below the pit base and the water table level

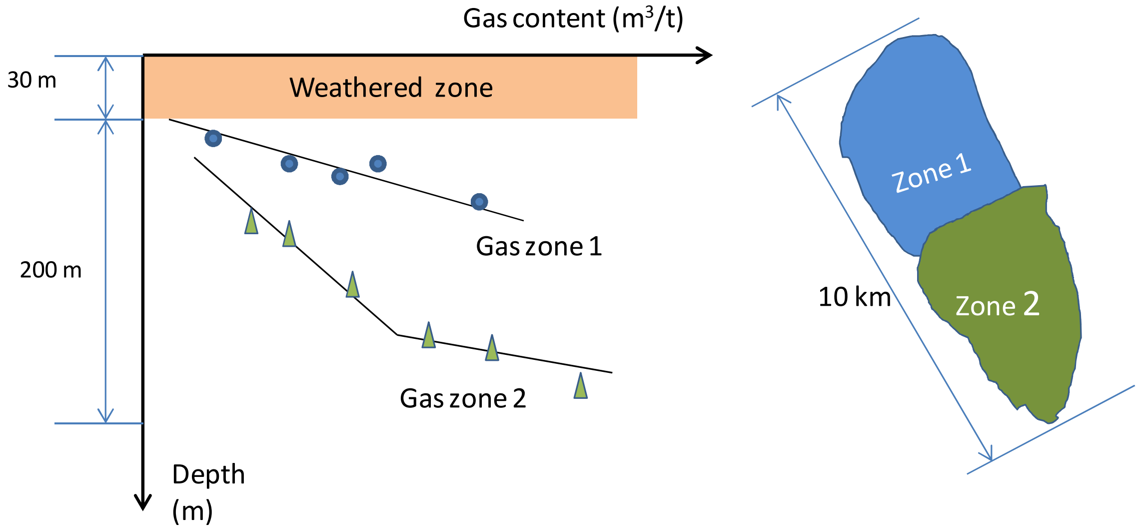

Figure 6. A mine lease is divided into a number of gas zones in which gas composition and gas content follow similar patterns with respect to depth

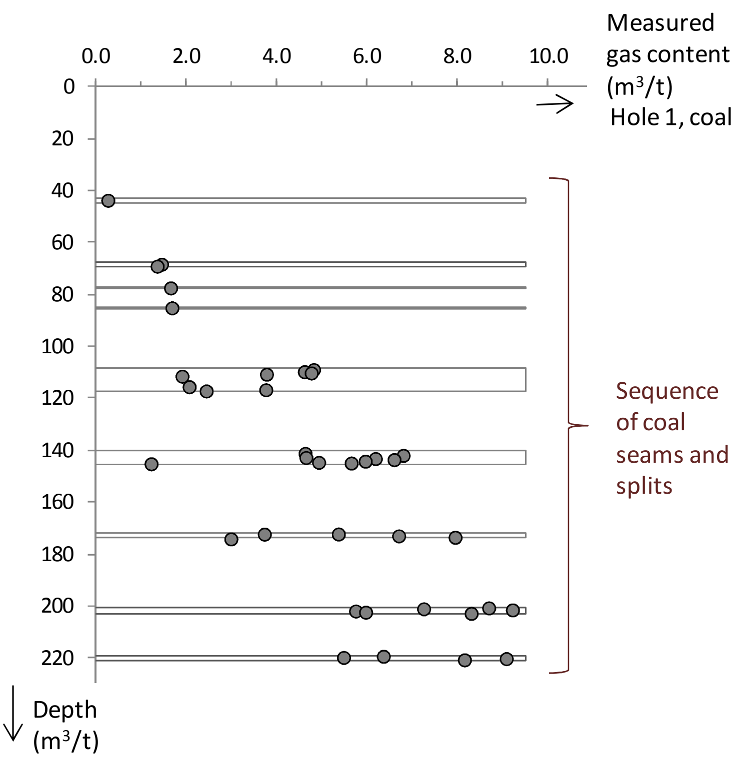

Figure 7. Measured gas content of coal core samples against depth. The rectangles show the position of coal seams

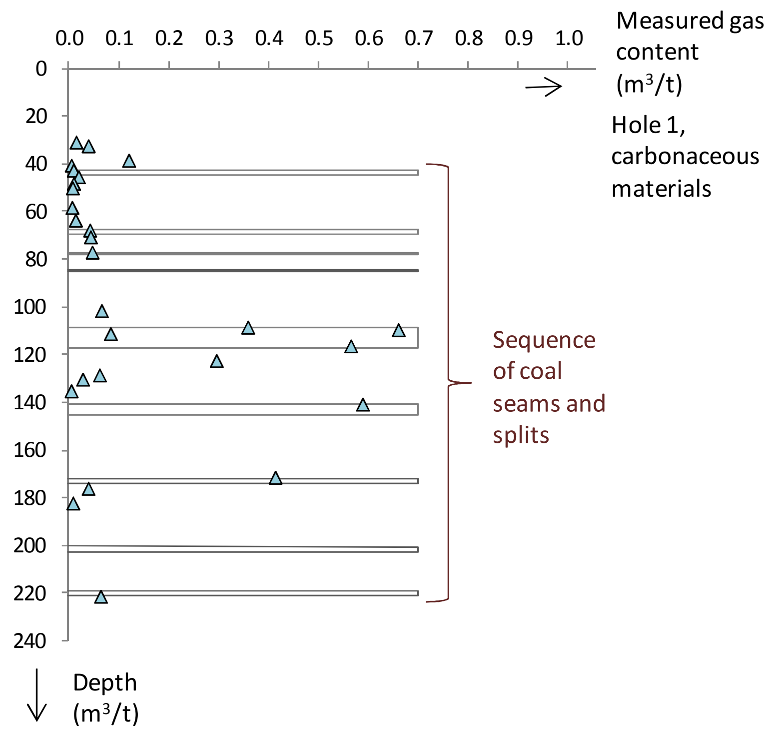

Figure 8. Gas content of interburden core samples (carbonaceous layers). The rectangles show the position of coal seams.

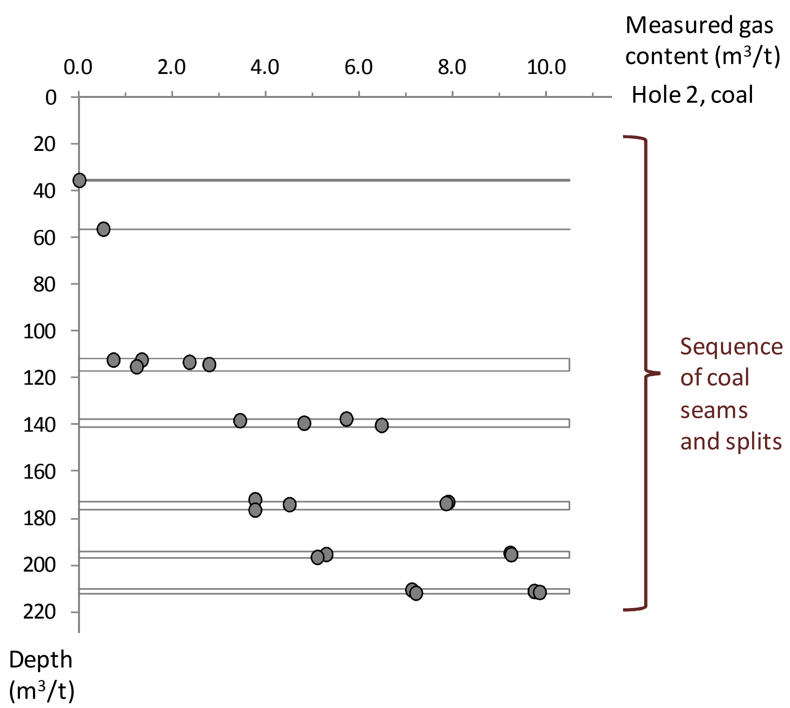

Figure 9. Measured gas content of coal core samples against depth. The rectangles show the position of coal horizons crossed by Borehole 2.

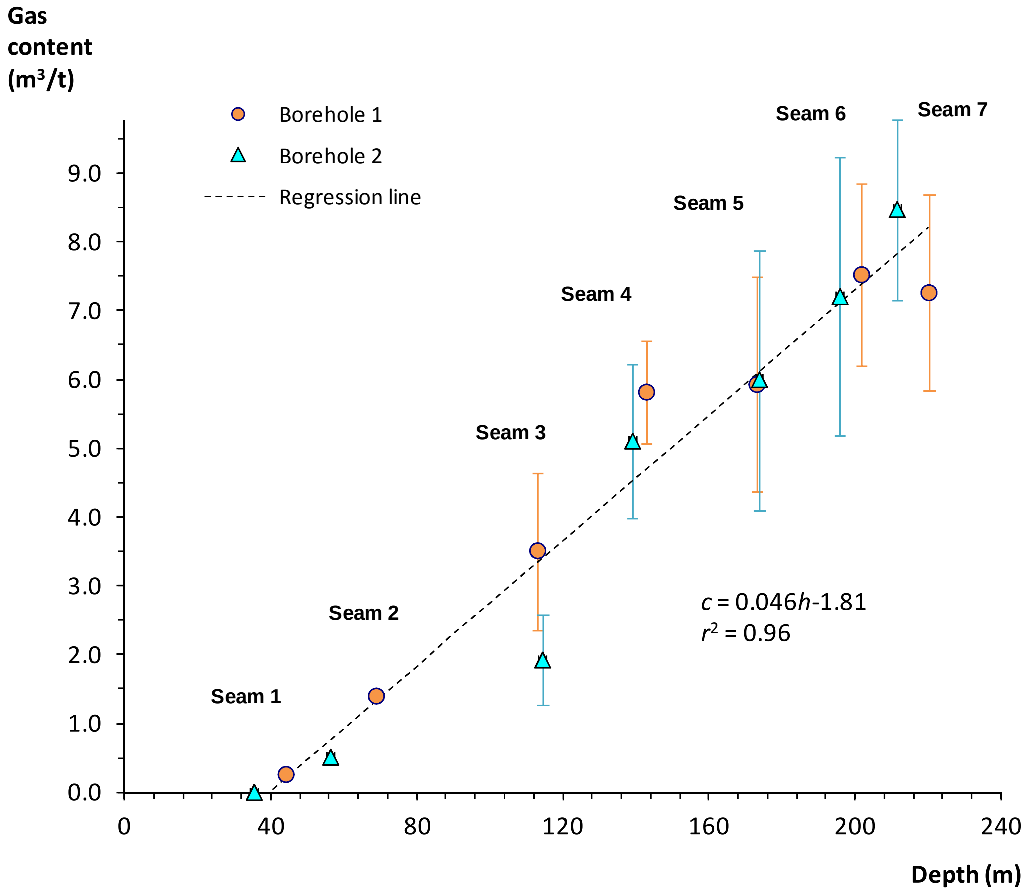

Figure 10. Average gas content of the main coal seams intersected by Boreholes 1 and 2; error bars correspond to plus or minus one standard deviation.

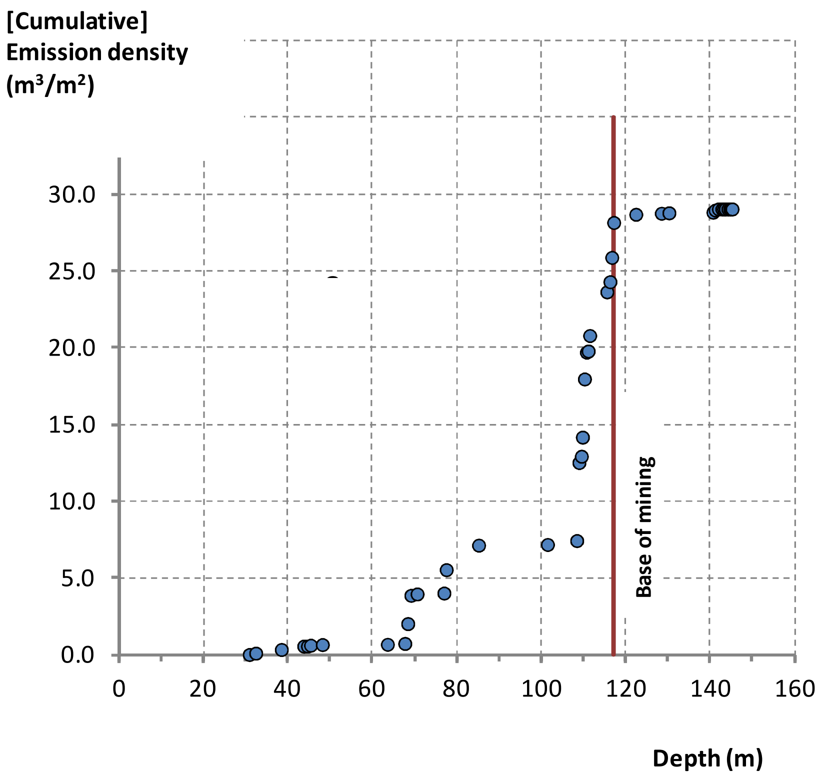

Figure 11. Cumulated emission density ($\sum q_i$) as a function of depth. Emissions from individual layers (q_i) are added to each depth to quantify the increase in emissions as depth of mining increases.